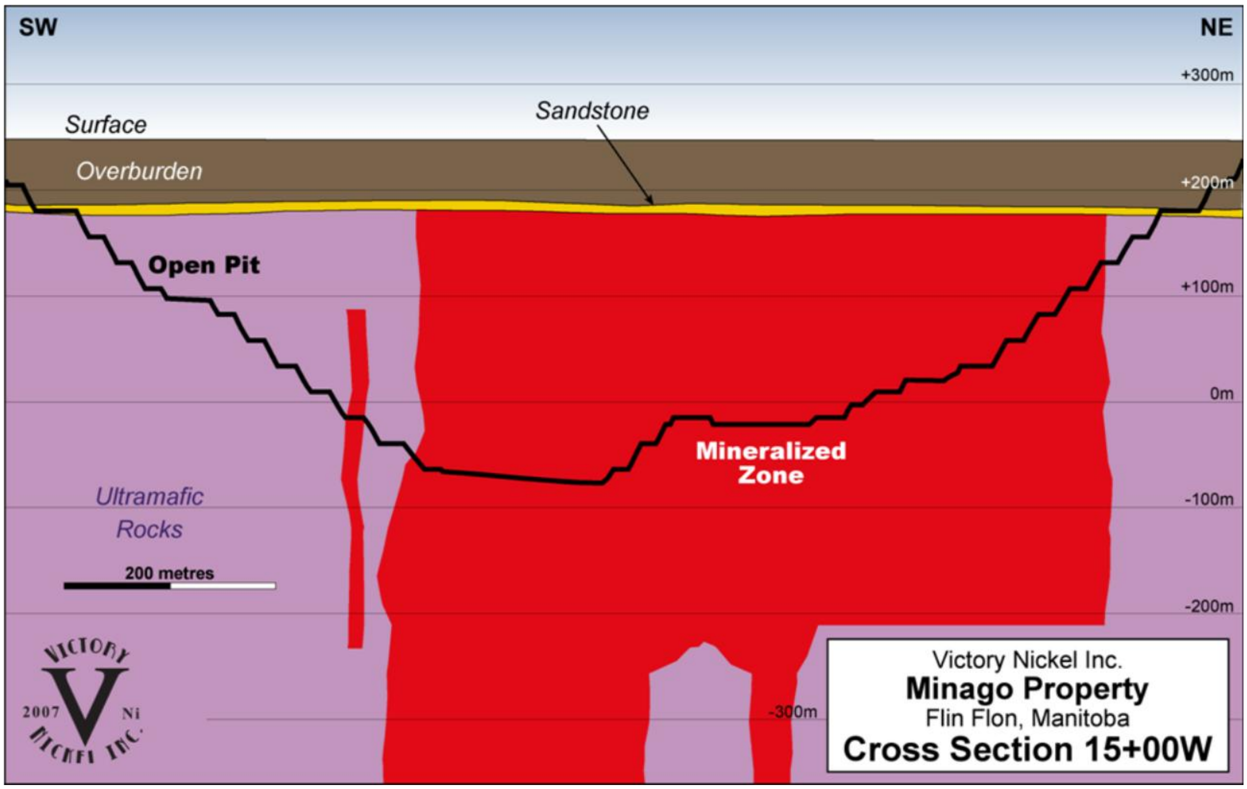


CHAPTER 2 – PROJECT DESCRIPTION

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

2. PROJECT DESCRIPTIONS

2.1 Project Overview

The Minago Property is located in Manitoba's Thompson Nickel belt, approximately 225 km south of Thompson, Manitoba, Canada (Figure 2.1-1).

In 2006, Nuinsco Resources Ltd. (Nuinsco) retained Wardrop Engineering Inc. (Wardrop) to provide the Preliminary Economic Assessment (PEA) of the Property. The PEA was completed in accordance with the National Instrument 43-101 (NI 43-101) requirements to identify the resources within economic open pit and underground mine designs.

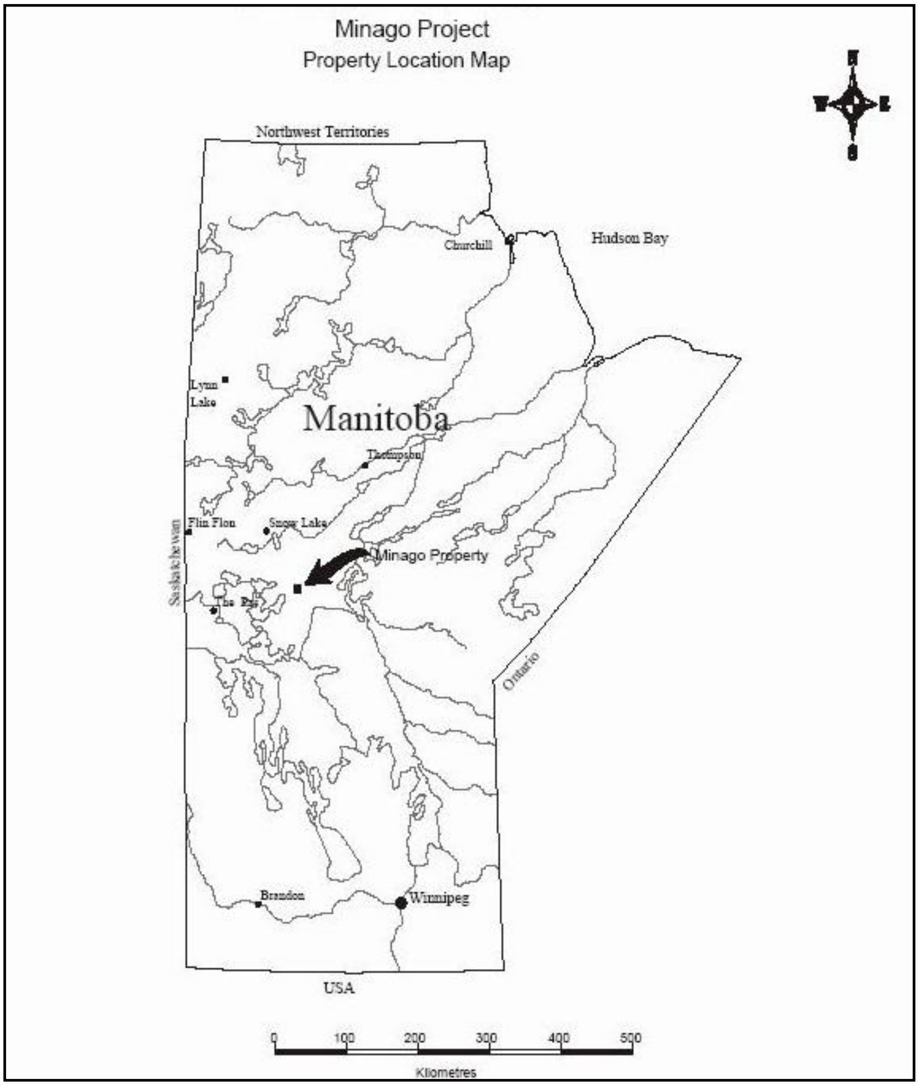
At the time the PEA was issued, Nuinsco owned 100% of the mining lease on the Property. In 2007, ownership of the Property was transferred to Victory Nickel Inc. (Victory Nickel), at that time, a wholly owned subsidiary of Nuinsco. On April 24, 2007, Victory Nickel engaged Wardrop to prepare the Minago Feasibility Study and a NI 43-101 compliant report. For this work, the resource estimation was provided by Wardrop in accordance with the Canadian Institute of Mining, Metallurgy, and Petroleum (CIM) Mineral Resource and Mineral Reserves definitions.

Wardrop found that the Minago deposit has potential as a large tonnage, low-grade nickel sulphide deposit amenable for open pit, and possibility for underground bulk tonnage mining methods. Significant parts of the deposit below a depth of 400 m require additional drilling to upgrade the resource class from inferred to indicated (Wardrop, 2009b).

Wardrop estimates that the Minago deposit contains a measured resource of 9.1 Mt grading 0.47% NiS above a cutoff grade of 0.2% NiS. In addition, the deposit contains 35 Mt of indicated resource at 0.42% NiS above a 0.2% NiS cutoff grade. An inferred resource of 12 Mt at 0.44% NiS above a 0.2% NiS has also been estimated (Wardrop, 2009b). The potential of the Minago Property is further supported by metallurgical testing in which very high grade concentrate was produced.

Wardrop also identified a sandstone horizon averaging ten metres thick above the unconformity of the main nickel bearing serpentinite. These well rounded silica sand particles in the sandstone formation were identified as being suitable for use as hydraulic fracturing sand, or "frac sand". When used as proppants in oil or gas wells these sands will improve the porosity of the shale beds leading to improved recovery and enhanced production. Currently, in onshore US wells, approximately 50% of the gas wells and 30% of the oil wells are hydraulically fractionated (Wardrop, 2009b).

The deposit has potential as a large tonnage, low-grade nickel sulphide deposit (25.2 Mt at 0.43% nickel (Ni), 0.20% cut-off grade) and contains 14.8 Mt million tonnes of marketable frac sand. The potential of the Property is supported by a recent metallurgical test program, where a very high



Source: Wardrop, 2006

Figure 2.1-1 Property Location Map

grade nickel concentrate was produced. The excellent recoveries for the ore from the open pit mine are substantiated by historical and current metallurgical testing data.

Together with the limestone-dolomite, the sandstone layer must be removed to access the nickel mineralization within the proposed open pit mine. To capture the value of this sand, Victory Nickel instructed Wardrop to include an assessment of frac sand within the Minago Feasibility Study. As a result of this additional work, the economic viability of commercial frac sand production has been established (Wardrop, 2009b).

The Property has a favourable location adjacent to the paved provincial Highway 6, which traverses north to Thompson. A 230 kV Manitoba Hydro power line runs parallel to the highway. The Property is only 60 km from the Omnitrax Canada railway line, which extends from Flin Flon and The Pas to Churchill. Grand Rapids is the closest township, located approximately 100 km south of the Property.

The mine life is estimated to be seven and two partial years, with frac sand being produced throughout the life of the mine and beyond. Frac sand will be processed for a period of ten years. Accommodation facilities and other associated facilities will be provided for the majority of the workforce, who will manage, operate, and maintain the mine on a rotational basis. To the extent possible, the workforce will be comprised of members of the local First Nations community.

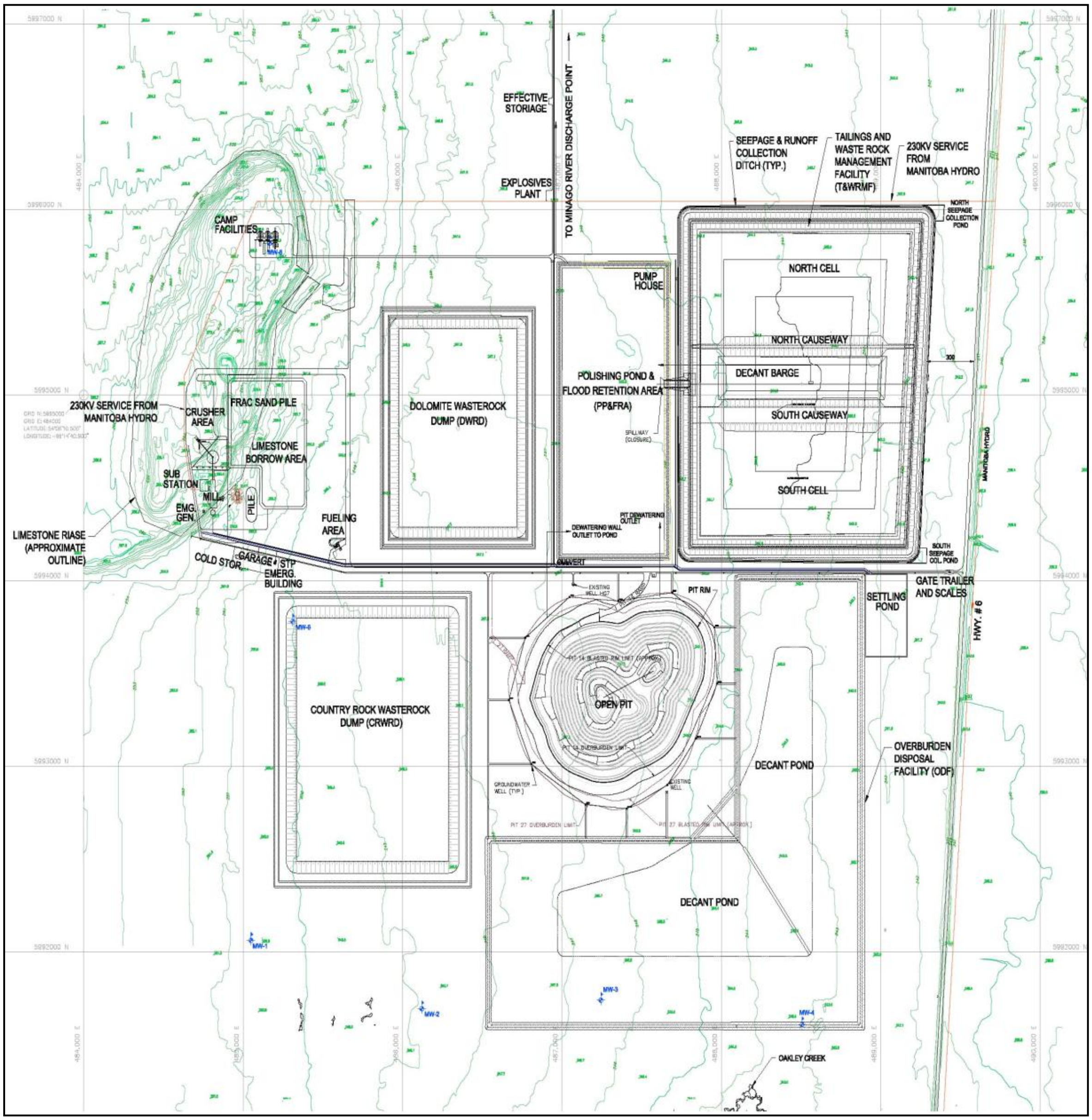
As currently configured, the proposed project will be comprised of an open pit mine, an Ore Concentrating Plant, a Frac Sand Plant, and supporting infrastructure (Figure 2.1-2). The Ore Concentrating Plant will process 3,600,000 t/a of ore through crushing, grinding, flotation, and gravity operations. This feed rate will produce approximately 49,500 t/a of 22.3% nickel concentrate on an average year before transportation losses and approximately 46,400 t/a after losses. The Frac Sand Processing Plant will be capable of producing between 1,500,000 t/a of various sand products including 20/40 and 40/70 frac sand, glass sand, and foundry sand products.

The mine site is situated within a topographically low area of water-saturated peat and forest terrain. The area is almost entirely swampy muskeg with vegetation consisting of sparse black spruce and tamarack set in a topographic relief of less than 3 m. Although this low area extends for significant distances to the north and east, elevated limestone outcrops exist to the south and west at a distance of 7 to 20 km from the site.

The site is located within the Nelson River sub-basin, which drains northeast into the southern end of the Hudson Bay. The basin has two more catchments, the Minago River and the Hargrave River, which enclose the project site to the north. There are two more tributaries, the William River and the Oakley Creek present at the periphery of the project area. The catchments of these two tributaries are within the Lake Winnipeg basin and drain northward into the Nelson River sub-basin.

The supporting infrastructure will include:

- a Tailings and Ultramafic Management Facility (TWRMF), rock dumps, and overburden dumps with supporting facilities;
- an Explosives Plant and explosives storage;
- a Potable Water Treatment Plant;
- local flood collection ponds and flood retention area with associated pumping systems;



Source: adapted from Wardrop, 2009b

Figure 2.1-2 General Site Plan of Minago

- de-watering systems with associated pipelines and pumping stations;
- roads and laydown areas;
- staff accommodations and facilities;
- open pit mining equipment including trucks, shovels, loaders, and drills; and
- truck repair and maintenance facilities.

The plant and infrastructure facilities have been located as close to the open pit mine as possible, based on a geotechnical investigation that identified the closest location with the best foundation conditions for the heavy equipment.

The plant and infrastructure facilities, shown in Figure 2.1-2, have been located as close to the open pit mine as possible on the limestone bluff to the west of the site. The escarpment will be cut back to a general elevation of 254 m.a.s.l. to ensure clearance above the water table for the plant facilities. The crusher will be located on the limestone bluff at a position where the elevation grade is most favourable. A more detailed sketch showing the plant and the camp facilities is given in Figure 2.1-3.

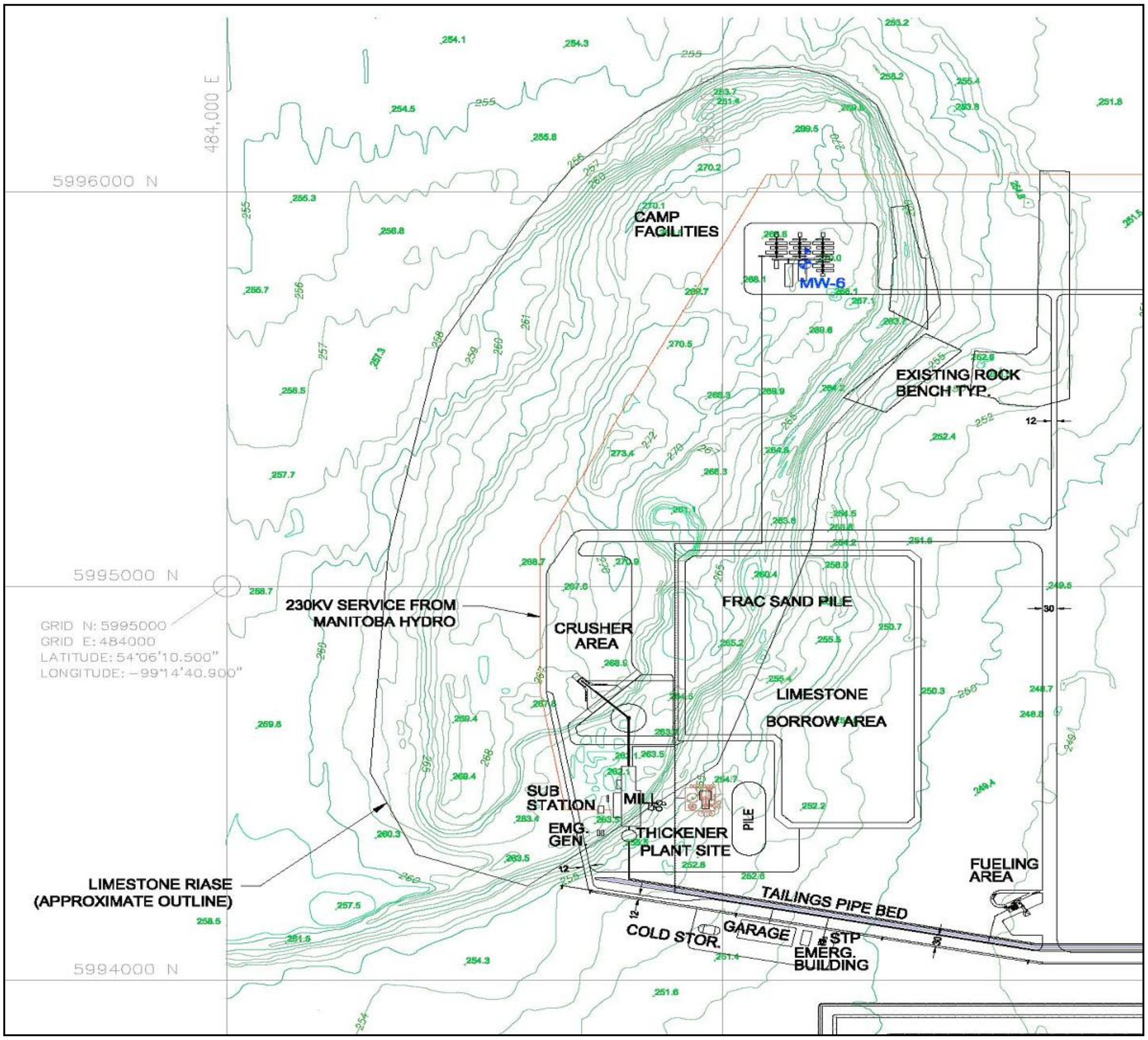
The Tailings and Waste Rock Management Facility (TWRMF) has been located on the east side of the side of the property where the geotechnical investigation has identified the best foundation conditions. At this location, to the northeast of the mine, the limestone founding strata was found to be between 3 to 4 m below the surface. Typically, on the balance of the site within close proximity of the open pit mine, the limestone horizon is 10 to 12 m below the surface.

The dumps for country rock, waste dolomite and the overburden were located around the pit to minimize the haul distances from the pit.

The road network was determined by the location of the dumps, facilities, and the ring road around the open pit mine, which will be used to access the de-watering wells. An access and maintenance road to service the discharge line to the Minago River was positioned in relation to the flood retention area and the associated pump houses. A similar discharge line will feed into a small tributary of the Oakley Creek.

2.1.1 Project Purpose and Need

At present, the world demand for nickel is exceeding the available supply. North America is not self sufficient in its nickel production. China and India have become the world’s largest consumers of Nickel. The demand for nickel in China will continue to grow as the World’s economies continue to improve. This suggests strong continued growth in nickel consumption. The long- term picture for nickel production shows no relief in sight for the current market trend. The increasing demand for nickel will continue to outpace the forecasted increases in production. The timing for the development of a nickel mine producing high grade nickel concentrate is excellent.



Source: Wardrop, 2009b

Figure 2.1-3 Plant and Camp Facilities

The market for nickel concentrates is strong, bringing favourable purchase terms and providing long-term security to project economics. Victory Nickel Inc. (VNI) intends to take advantage of this excellent market opportunity and the exceptional ore resource of the Minago Project to create profits for its shareholders. The Minago Project will provide a much-needed boost to the Manitoba economy, an economy that has experienced a serious downturn due to the current economic recession. The project will provide a solid tax base, support for infrastructure development, and workforce development opportunities for local communities.

2.1.2 Project Timing

The mine life is estimated to be seven full years and two partial years, with concentrate production mirroring ore production. The frac sand, which is to be mined at the start of mining will be produced throughout the life of the mine and beyond. The first partial year's ore production (2013) will be stockpiled pending commissioning of the Ore Processing Plant in 2014.

The construction phase is scheduled to commence with the overburden removal together with the open pit dewatering systems in the spring of 2011 (Year -3). Electrical supply installations are required to be in place before the spring of 2011 to provide power for the pit dewatering and dredging programs.

Construction can commence once all the permits are obtained from the MB Government. Victory Nickel anticipates to get the Environmental Act License approvals for mining and mill construction by August, 2010. Commencement of milling operations will commence in Year 2012 (Year -2) and into Year 2013 (Year -1). This is contingent upon receipt of the required licenses from the MB Government. Frac sand production will start in Year 2013 (Year -1) and Nickel production will start in 2014 (Year 1).

2.1.3 Overview of Project Components, Design Criteria and General Layout

The overall layout for the Minago Project is presented in Figure 2.1-2. The project has and will continue to be designed according to the following general criteria:

- The project must meet or exceed the highest standards of industrial health and safety and demonstrate minimum environmental impact. Existing industry guidelines, codes of practice, standards and regulations will be consulted and the most stringent will be applied.
- The project will mine and process 10,000 t/d of run of mine ore, including variable amounts of external dilution. In addition, the facility will produce frac sand.
- The project will be designed to operate continuously, 365 days per year with appropriate design allowances in each department for planned maintenance shut downs.
- Tailings and ultramafic waste rock will be co-disposed of in the Tailings and Waste Rock Management Facility to control potential for Acid Rock Drainage (ARD) and Metal Leaching (ML).
- The mining method will be drill and blast, and use electrical and diesel powered equipment. The mining method must be very adaptable, safe, and conserve the resource by achieving high performance standards.
- The process plant will use flotation methods to produce one nickel concentrate to agreed quality specifications. The concentrates will be sold to external smelters for processing to metal. The project will not produce marketable metal as there will be no smelter.

- Employees will be drawn from local communities and provided with hotel style accommodation at the mine camp.
- A nucleus of skilled experienced workers will be recruited for initial development and construction. Through local recruiting and comprehensive training, the company has set the goal of maximizing the percentage of Manitoba residents, and the Communities of Interest (COI) in particular.

When completed, the Minago project production facilities will consist of a 10000 t/d Open pit, flotation concentrator, process water treatment plant, waste rock dumps, and a subaqueous tailings and waste rock management facility. These production facilities will be supported by the following infrastructure: a maintenance workshop, warehouse, electric power supply, fuel and propane tank farm, offices, sanitary and changing facilities (dry), camp, water supply system, sewage plant, domestic and industrial waste disposal and transportation corridors.

2.2 Certificate of Title and Mineral Depositions

2.2.1 Mineral Rights

2.2.2 Mineral Depositions

The Property is comprised of one contiguous group of claims and one mineral lease, augmented by an isolated claim and a second adjacent mineral lease (Figures 2.2-1 and 2.2-2). The contiguous block consists of one mineral lease and 40 unpatented mineral claims with a combined surface area of 7,298.23 hectares (ha) (Tables 2.2-1 and 2.2-2).

Mineral Lease 2 and Mineral Lease 3, which were issued on April 1, 1992, for a period of 21 years and may be renewed after that time at the discretion of the Minister of Manitoba Industry, Economic Development, and Mines. The annual rental cost of the mineral leases is $1,984 for Mineral Lease 2 and $1,416 for Mineral Lease 3, both due annually on April 1.

Mineral claims KON 1 through KON 4 are in good standing until May 17, 2021 plus 60 days. Thereafter the cost to keep the KON mineral claims in good standing is $25.00/ha per year in the form of work conducted and submitted for assessment or payment in lieu thereof.

Mineral claims BARNEY 1 to BARNEY 6 inclusive are in good standing until September 24, 2022 plus 60 days. After that, the costs to keep the BARNEY claims in good standing is $25.00/ha per year in the form or work conducted and submitted for assessment or payment in lieu thereof.

The mineral claims MIN 1 through MIN 29 are in good standing until the dates indicated on Table 2.2-1. The earliest expiry date for this claim group is January 26, 2009. After expiry, the cost to keep the MIN claims in good standing is $12.50/ha per year until the year 2017 in the form of work conducted and submitted for assessment or payment in lieu thereof. Thereafter the cost to keep

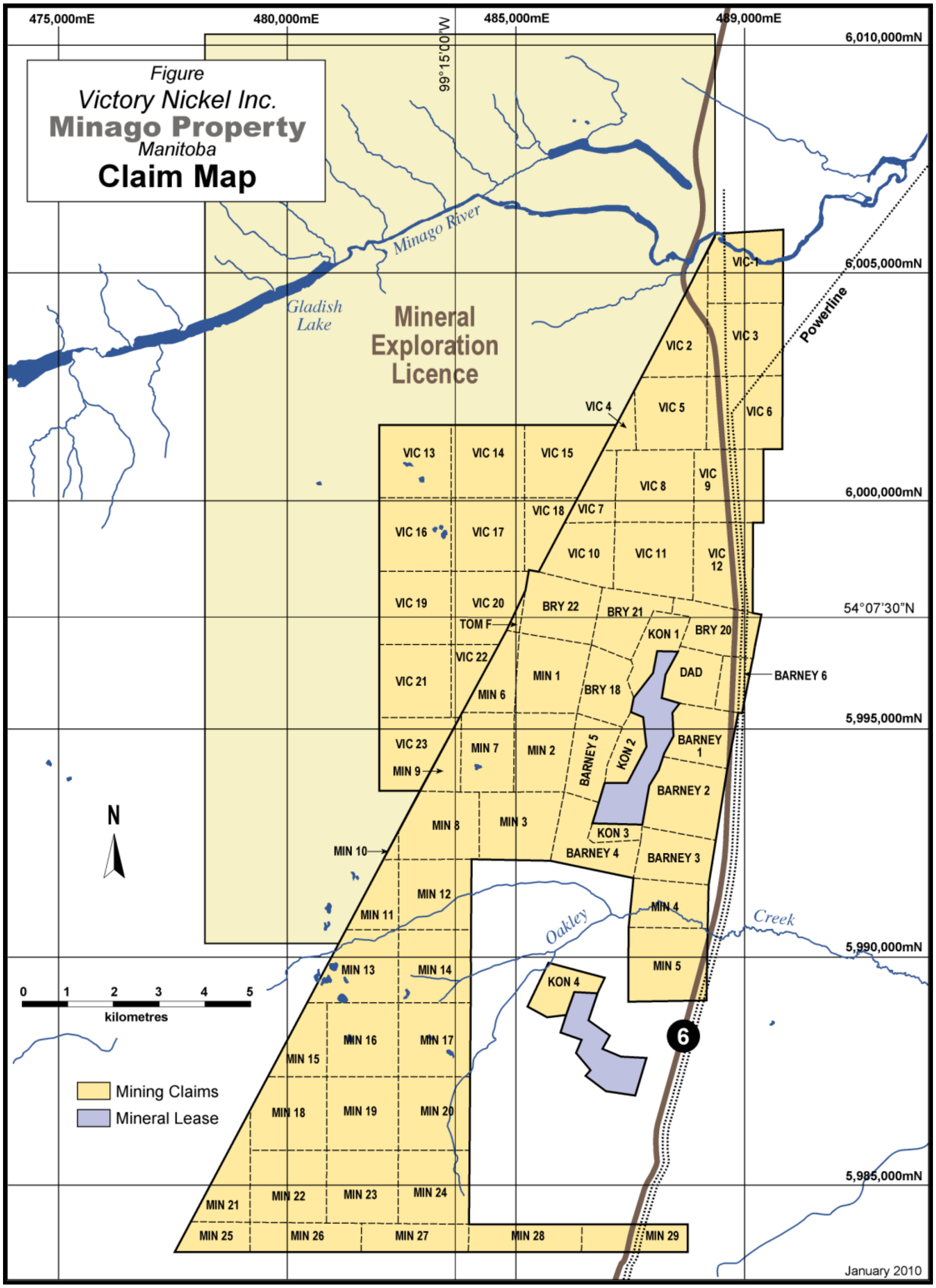
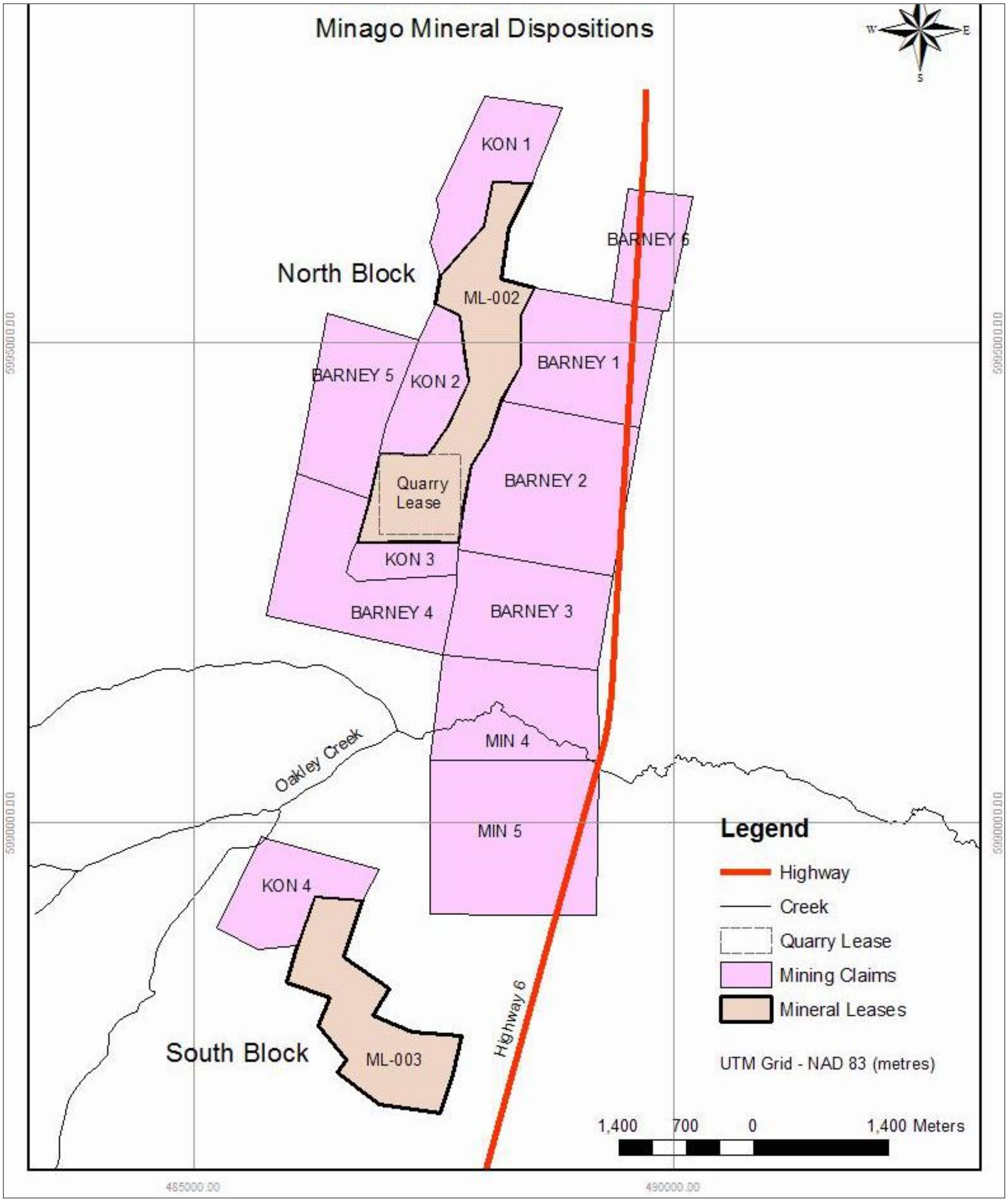


Figure 2.2-1 Minago Mineral Dispositions



Source: Wardrop, 2006

Figure 2.2-2 Minago's Historical Mineral Dispositions

Table 2.2-1 Minago Claim Group

Claim Name	Claim Number	Claim Holder	Date Staked	Date Recorded	Expiry Date	Area (ha)
KON 1	P2527F	Victory Nickel Inc.	1994/03/08 16:30	18/03/1994	17/05/2021	108
KON 3	P2529F	Victory Nickel Inc.	1994/03/10 16:05	18/03/1994	17/05/2021	43
KON 2	P2528F	Victory Nickel Inc.	1994/03/11 11:50	18/03/1994	17/05/2021	73
KON 4	P2530F	Victory Nickel Inc.	1994/03/13 11:00	18/03/1994	17/05/2021	105
BARN	MB5390	Victory Nickel Inc.	2004/07/04 15:45	26/07/2004	24/09/2022	168
BARN	MB5391	Victory Nickel Inc.	2004/07/05 16:00	26/07/2004	24/09/2022	242
BARN	MB5392	Victory Nickel Inc.	2004/07/06 16:00	26/07/2004	24/09/2022	170
BARN	MB5393	Victory Nickel Inc.	2004/07/07 16:15	26/07/2004	24/09/2022	184
BARN	MB5394	Victory Nickel Inc.	2004/07/08 15:45	26/07/2004	24/09/2022	155
BARN	MB5395	Victory Nickel Inc.	2004/07/17 13:30	26/07/2004	24/09/2022	76
MIN 1	MB7027	Victory Nickel Inc.	2006/11/06 19:20	27/11/2006	26/01/2009	235
MIN 2	MB7028	Victory Nickel Inc.	2006/11/07 19:30	27/11/2006	26/01/2009	214
MIN 3	MB7029	Victory Nickel Inc.	2006/11/08 18:30	27/11/2006	26/01/2009	252
MIN 4	W48594	Victory Nickel Inc.	2006/07/27 19:00	04/08/2006	03/10/2009	162
MIN 5	W48595	Victory Nickel Inc.	2006/07/27 19:30	04/08/2006	03/10/2009	256
MIN 6	MB7030	Victory Nickel Inc.	2006/11/06 19:05	27/11/2006	26/01/2009	135
MIN 7	MB7031	Victory Nickel Inc.	2006/11/07 19:15	27/11/2006	26/01/2009	204
MIN 8	MB7033	Victory Nickel Inc.	2006/11/10 18:20	27/11/2006	26/01/2009	205
MIN 9	MB7032	Victory Nickel Inc.	2006/11/10 16:00	27/11/2006	26/01/2009	78
MIN 10	MB7066	Victory Nickel Inc.	2007/01/09 14:20	23/01/2007	24/03/2009	57
MIN 11	MB7067	Victory Nickel Inc.	2007/01/09 13:40	23/01/2007	24/03/2009	121
MIN 12	MB7141	Victory Nickel Inc.	2007/01/10 15:22	23/01/2007	24/03/2009	250
MIN 13	MB7142	Victory Nickel Inc.	2007/01/11 16:51	23/01/2007	24/03/2009	256
MIN 14	MB7143	Victory Nickel Inc.	2007/01/10 16:51	23/01/2007	24/03/2009	256
MIN 15	MB7144	Victory Nickel Inc.	2007/01/12 14:37	23/01/2007	24/03/2009	138
MIN 16	MB7145	Victory Nickel Inc.	2007/01/12 16:0	23/01/2007	24/03/2009	256
MIN 17	MB7146	Victory Nickel Inc.	2007/01/11 15:1	23/01/2007	24/03/2009	247
MIN 18	MB7147	Victory Nickel Inc.	2007/01/13 16:0	23/01/2007	24/03/2009	247
MIN 19	MB7148	Victory Nickel Inc.	2007/01/14 16:0	23/01/2007	24/03/2009	256
MIN 20	MB7149	Victory Nickel Inc.	2007/01/13 15:2	23/01/2007	24/03/2009	243
MIN 21	MB7150	Victory Nickel Inc.	2007/01/15 13:4	23/01/2007	24/03/2009	181
MIN 22	MB7151	Victory Nickel Inc.	2007/01/14 16:1	23/01/2007	24/03/2009	256
MIN 23	MB7152	Victory Nickel Inc.	2007/01/15 15:4	23/01/2007	24/03/2009	256
MIN 24	MB7153	Victory Nickel Inc.	2007/01/08 16:2	23/01/2007	24/03/2009	241
MIN 25	MB7154	Victory Nickel Inc.	2007/01/16 13:0	23/01/2007	24/03/2009	88
MIN 26	MB7155	Victory Nickel Inc.	2007/01/16 15:4	23/01/2007	24/03/2009	145
MIN 27	MB7156	Victory Nickel Inc.	2007/01/07 16:2	23/01/2007	24/03/2009	145
MIN 28	MB7157	Victory Nickel Inc.	2007/01/08 15:4	23/01/2007	24/03/2009	153

MIN 29	MB7158	Victory Nickel Inc.	2007/01/07 15:51	23/01/2007	24/03/2009	153
TOM F	MB8549	Victory Nickel Inc.	2008/04/16 15:40	12/05/2008	11/07/2010	14
DAD	MB8497	Victory Nickel Inc.	2008/05/22 16:00	28/05/2008	27/07/2010	132
				Total		**7156**

Table 2.2-2 Minago Mineral Leases

Lease Name	Lease Number	Lease Holder	Expiry Date	Area (ha)
Mineral Lease 2	ML-002	Victory Nickel Inc.	01/04/2013	247.2
Mineral Lease 3	ML-003	Victory Nickel Inc.	01/04/2013	176.9

the MIN mineral claims in good standing is $25.00/ha per year in the form of work conducted and submitted for assessment or payment in lieu thereof.

Mineral claims VIC 1 through VIC 12 are in good standing until April 17, 2021 plus 60 days. Thereafter the cost to keep the KON mineral claims in good standing is $12.00/ha per year in the form of work conducted and submitted for assessment or payment in lieu there of.

Mineral claims VIC 13 through VIC 23 are in good standing until December 21 2011 plus 60 days. Thereafter the cost to keep the KON mineral claims in good standing is $12.00/ha per year in the form of work conducted and submitted for assessment or payment in lieu of.

As a result of an option agreement entered into with Xstrata Nickel on claims BRY 18, BRY 20, BRY 21, BRY 22, TOM F, and DAD and subsequently fully exercised at year- end 2008, a NSR is payable to Xstrata on any exploited mineralization found on the claims. The NSR consists of a 2% royalty when the London Metal Exchange (LME) three-month nickel price is greater than, or equal to, US$13,227.74/t, and a 1% NSR when the three-month price of nickel is less than US$13,227.74/t. All other metals will be subject to a 2% NSR.

Victory Nickel has obtained a quarry lease (QL-1853) with an area of 69.88 ha on a portion of the mineral lease ML-002. In addition, four quarry leases, surrounding and contiguous with QL-1853 have been applied for. These pending quarry leases over a total area of an additional 244 ha. Victory Nickel has also been issued the 10-year quarry lease QL-2067 that commenced in November 2009 (Figure 2.2-3).

Quarry lease QL-1853 has a term of 10 years and may be renewable for further terms of 10 years subject to the discretion of the Minister. The annual rental cost for the quarry lease is $1,677.12 payable on the anniversary date. The annual rental cost for QL-2067 is $1,680.00. A copy of lease QL-2067 is provided in Appendix 2.2.

Victory Nickel has made the initial payment of $150,000 and incurred expenditures of at least $500,000 on the claims prior to September 30, 2008. Payment of the remaining outstanding ‘cash in lieu’ is on the books of Manitoba Science, Technology, Energy, and Mines. The NSR consists of a 2% royalty, payable to Xstrata, when the London Metal Exchange (LME) three-month nickel price is greater than, or equal to, US$13,227.74/t, and a 1% NSR when the three-month price of nickel is less than US$13,227.74/t. All other metals will be subject to a 2% NSR.

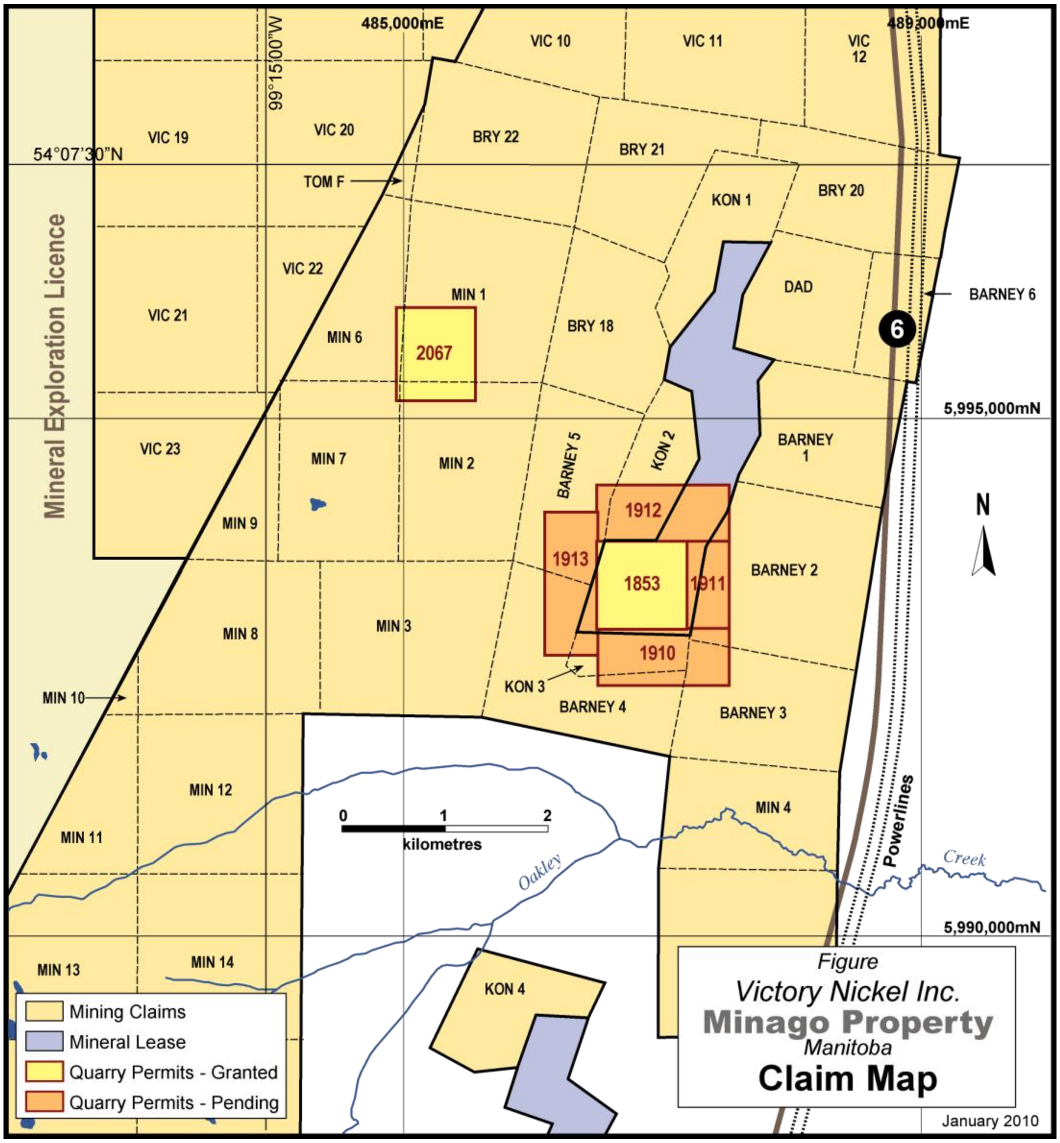


Figure 2.2-3 Minago Property Quarry Lease Status

2.2.3 Ownership

Victory Nickel has 100% ownership of the Minago Project and also the Mines and Minerals Act entitles mineral claims owners the rights as given below:

The holder (Victory Nickel) of a mineral claim has the exclusive right to explore for and develop the Crown minerals, other than the quarry minerals, found in place on, in, or under the lands covered by the claim (The Mines and Minerals Act, 73[1]).

The lessee (Victory Nickel) of a mineral lease has the exclusive right to the Crown minerals, other than quarry minerals, that are the property of the Crown and are found in place or under the land covered by the mineral lease. The lessee also has access rights to open and work a shaft or mine, and to erect buildings or structures upon the subject land (The Mines and Minerals Act, 108[a], [b], [i], [ii]).

With respect to the pending quarry lease, the lessee of a quarry lease has the exclusive right to the Crown quarry minerals specified in the lease (in this case limestone) that are found on or under the land covered by the lease and that are the property of the Crown (The Mines and Minerals Act ,140[1] [a]).

There are no instruments registered with the Mining Recorder at Manitoba Energy, Mines, Science and Technology Ministry on any of the mineral dispositions with respect to liens, judgments, debentures, royalties, back-in rights or other agreements.

2.2.3.1 Encumberances

Encumbrances on the mineral dispositions include:

- For Norway House District: Registered Trap Line (RTL) # 150-07 covering all mineral dispositions.
- For Forestry Branch, Forest Management Licence: (FORM REPAP W 0012 and FORM REPAP 2 0012 covering all mineral dispositions.
- For Manitoba Hydro, Transmission Line and Easement Agreement: Right of Way 319.735 m wide, plan number 5830 N.L.T.O for portions of BARNEY 1, BARNEY 2, BARNEY 6, and MIN 5.
- For Manitoba Department of Highways: Right of way 91.44 m wide that is split 65.532 m west of the centre line and 25.908 east of the centre line, plan number 6149 N.L.T.O for portions of BARNEY 1, BARNEY 2, BARNEY 3, BARNEY 6, MIN 4, and MIN 5.
- For Manitoba Department of Highways: Quarry Withdrawal, plan number 6148 N.L.T.0.for southeast corner of ML-003.

With respect to the pending quarry lease, royalties are applicable to quarry products such as limestone and frac sand at varying rates depending on their end use. Currently, a rehabilitation levy of $0.10/t will not apply to quarry production.

There is no mining-related infrastructure on the Property although the Minago River Nickel Deposit, previously referred to as the Nose Deposit, is located on mineral lease ML 002.

There are no environmental liabilities attached to the Property.

2.2.4 Tenure Rights

The holder of a mineral claim has the exclusive right to explore for and develop the Crown minerals, other than the quarry minerals, found in place on, in, or under the lands covered by the claim (The Mines and Minerals Act, 73[1]).

The lessee of a mineral lease has the exclusive right to the Crown minerals, other than quarry minerals, that are the property of the Crown and are found in place or under the land covered by the mineral lease. The lessee also has access rights to open and work a shaft or mine, and to erect buildings or structures upon the subject land (The Mines and Minerals Act, 108[a], [b], [i], [ii]).

The lessee of a quarry lease has the exclusive right to the Crown quarry minerals specified in the lease (in this case limestone) that are found on or under the land covered by the lease and that are the property of the Crown [The Mines and Minerals Act, 140 (1) (a)].

2.2.5 Option Agreement with Xstrata Nickel

As a result of an option agreement entered into with Xstrata Nickel on claims BRY 18, BRY 20, BRY 21, BRY 22, TOM F, and DAD and subsequently fully exercised at year- end 2008, a NSR is payable to Xstrata on any exploited mineralization found on the claims. The NSR, consists of a 2% royalty when the London Metal Exchange (LME) three-month nickel price is greater than, or equal to, US$13,227.74/t, and a 1% NSR when the three-month price of nickel is less than US$13,227.74/t. All other metals will be subject to a 2% NSR.

2.3 Existing Land Use

The project is located in the Norway House Resource Management Area. In addition, there is a Registered Trap Line (RTL) # 150-07 covering all mineral dispositions.

Resource Management Areas have been established by the Manitoba government. The RMA, in which the project area is located, is currently an inactive area so there are no current land use plans developed for the project area.

2.4 Minago Project – Economic Assessment

2.4.1 Feasibilty Study

In 2007, Victory Nickel retained Wardrop to undertake a Feasibility Study of the Minago Project following positive results of the Scoping Study completed in 2006. The Feasiblity Study was completed in the first quarter of 2010. The results of the Feasibility Study are discussed below.

The deposit has potential as a large tonnage, low-grade nickel sulphide deposit (25.2 Mt at 0.43% nickel (Ni), 0.20% cut-off grade) and contains 14.8 Mt million tons of marketable frac sand. The potential of the Property is supported by a recent metallurgical test program, where a very high grade nickel concentrate was produced. The excellent recoveries for the ore from the open pit mine are substantiated by historical and current metallurgical testing data.

The economic aspects of a deposit would be constrained by some 80 m of overburden, limestone, and sand resulting in a high open pit strip ratio. However, in the case of the Minago Project, the 10 m sand layer just above the ultramafic ore bearing rock contains marketable frac sand, which offsets the cost of the stripping.

In addition to the Nickel Ore Concentrating Plant, the installation of a Frac Sand Processing Plant will generate further revenues for the project. The financial analysis assumes that critical revenue streams will be developed from both the nickel and frac sand resources. Table 2.4-1 shows the proposed production schedule by year, for the waste, the nickel ore and the sand.

Table 2.4-1 Production Schedule by Year and Product

	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	Total
Dolomite (kt)	42,655	43,179	15,183	1,0015	0	0	0	0	0	0	111,032
Granite (kt)	0	1,744	20,890	20,440	35711	24459	9,784	4,944	3,832	199	122,005
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 Ore (kt)	0	112	3,000	3,600	3,600	3,600	3,600	3,600	3,600	453	25,166
% Ni(S), Grade Ore	0.000	0.374	0.419	0.429	0.430	0.413	0.436	0.431	0.447	0.468	0.430
Total Tonnage (kt)	42,655	51,186	49,105	47,045	44,979	33,792	17,766	11,570	9,728	881	

During the development of the Feasibitly Study, certain concepts were pursued in the interests of cost and efficiency. In place of the mechanical removal of the overburden, Wardrop has selected a dredging option to reduce costs significantly and create more favorable spoil areas. By co-depositing the potentially acid generating, metal leaching ultramafic rock and sealing these within the tailings, significant infrastructure and legacy costs are eliminated. Finally, by shortening the production life of the Frac Sand Plant to match that of the Ore Processing Plant, general and administrative and surface facility costs will be minimized.

The mine life is estimated to be seven full years and two partial years, with concentrate production mirroring ore production. The frac sand which is to be mined at the start of mining is produced throughout the life of the mine and beyond. The first partial year’s ore production (2013) will be stockpiled pending commissioning of the Ore Processing Plant in 2014.

The Project features an open pit bulk tonnage mining method, a 3.6 Mt/a Nickel Ore Processing Plant, and a 1.5 Mt/a Sand Processing Plant producing various sand products, including 20/40 and 40/70 frac sand, and other finer sized sands. The Project will be built over a three year period at a capital cost of $596.3 million. The Nickel Ore Processing Plant is scheduled to come online in the spring of 2014 and the Frac Sand Plant is scheduled to come online in the spring of 2013.

The work undertaken for the Feasibility Study and Environmental Baseline Studies form the basis of the EIS. A copy of the Feasibility Study for the Minago Project can be obtained at www.sedar.com.

2.4.2 2006 Scoping Study (Wardrop, 2006)

In 2006, VNI retained Wardrop to provide a preliminary economic assessment of the Minago property. Their engineering and financial analysis was done using NI 43-101 compliant resources within economic pit shells and underground designs. The resource estimates were completed by P.J. Chornoby with assistance from Mirarco and all remaining assessments of the property were done by Wardrop.

P. J. Chornoby, P. Geo conducted an independent review of the geology, exploration history, historical resource estimates, resource estimates, and the potential for discovery of additional nickel mineralization of the Minago property in central Manitoba. This independent review summarizes the results of exploration conducted during the period from 1966 to 1991 and work conducted by Nuinsco from 2004 to October 31, 2006. An independent report was deemed necessary for material disclosure of new diamond drill data, mineral resource estimates and technical studies.

2.5 Project Alternatives

Victory Nickel Inc. sees no feasible alternative to Minago Project. The project is the principle asset of VNI and although there are other mineral deposits in the Minago Area, VNI does not own any interest in them and therefore cannot effect the evaluation of the possible co-development with the Minago deposit. Similarly, currently it is not possible to consider the potential addition of other deposits that may be discovered through exploration. Given the current and future global market for Nickel, the proposed project is the best available option to achieve the business goals of the company.

VNI has assessed a number of alternatives in coming to the proposed design of the Minago Project. The alternatives considered include the various ways that the project could be implemented or carried out, including alternative locations in the project area, routes and methods of development, implementation, and mitigation.

Examining the main project alternatives involved answering the following three questions:

1. What alternatives are technically and economically feasible?
2. What are the environmental effects associated with the feasible alternatives?
3. What is the rationale for selecting the preferred alternative?

Throughout the Minago Project design process, various mining concepts were developed, analyzed, refined and eventually focused down to preferred alternatives. This section describes alternatives that were considered by VNI, and the rationale for selecting the preferred alternative.

The decisions made by VNI and its consultants for the purposes of project design and mine planning are based on feasibility level information. This information provides a reasonable basis for detailed design.

2.5.1 Mining Method

A conventional open pit with full seven and two partial years of ore production life is envisaged after dewatering the overburden and overlying limestone and sandstone. Twelve metre bench heights will be used. A contractor will be employed to remove the overburden and some limestone during the two pre-production years. Equipment will be purchased to utilize the favourable electric power costs in Manitoba. Electric hydraulic shovels will load ore and waste into 218 tonne haul trucks.

Underground operations have been considered but were deemed to be uneconomical due to poor ground control and low-grade aspects. Open pit mining is the only feasible means of extracting the Minago deposit. There will be two products mined from the open pit – frac sand and nickel ore. Frac sand will be mined after the overburden materials (peat and clay and dolomitic limestone) have been removed. The removal of the frac sand will expose the nickel ore. Open pit mining method is the most optimal extraction method to extract both frac sand and nickel ore.

2.5.2 Pit Location

The pit is located where the ore is and therefore, there is no viable alternative.

2.5.3 Ore and Waste Haulage

VNI will use 218 tonne trucks to move ore to the mill and waste rock to the waste rock dumps. The 218 tonne trucks are the most economical mode of transportation bearing in mind the waste-to-ore ratio of 6.7 to 1 for mining the nickel sulphide ore and the frac sand. Transportation of ore and waste rock using high capacity equipment is the most viable approach and therefore, there is no viable alternative.

2.5.4 Ore Processing

Conventional flotation will be employed by VNI to process the ore, as there is no viable alternative. The process flowsheet will consist of crushing plant, grinding circuit and a concentrator.

2.5.5 Waste Rock Disposal

The locations of the waste rock dumps and overburden stockpile are selected to optimize hauling costs and are located in the vicinity of the open pit. The waste rock dumps for Country Rock and Dolomite and overburden stockpile locations were selected based on geotechnical investigation results and for the following reasons:

- they are located near the pit to optimize haul distances;
- the overburden is largely clay;
- there will be large waste rock volumes; and
- the waste will be Non-Acid Generating (NAG).

The existing facilities have adequate storage capacities for the waste rock that will be generated from pit during development and operational phases and as such, no alternative to the existing infrastructure were examined. During the operations phase, waste rock will be disposed into the dumps. The Overburden, Dolomite and Country Waste Dumps with store approximately 11 Mt of overburden, 90 Mt of limestone waste and 122 Mt of granitic (country rock) waste, respectively. Approximately 35.67 Mt of ultramafic waste rock will be co-disposed with tailings in a Tailings and Ultramafic Waste Rock Management Facility (TWRMF). Co-disposal will minimize metal leaching and increase the stability of the tailings management area.

2.5.6 Tailings Disposal

Sub-aerial disposal of liquid tails (slurry) was selected for the property. An alternative method involving the on-land disposal of dry tailings in paste form was assessed. Advantages of paste tailings disposal are:

- A tailings dam does not have to be constructed, removing a significant capital cost item.
- Water does not have to be managed to prevent the oxidation of potentially acid generating materials.

The disadvantages of this option are:

- Dust can be generated from the tailings.
- Pumping is more difficult and expensive than for liquid tailings.
- Operating costs are higher due to the pumping and, potentially, the need to add minimal cement to the tails to retain its form as paste.

The most significant reason for selecting sub-aqueous disposal of liquid tailings is that VNI prefers to adopt proven technology rather than embark on a pioneer project. While numerous operations have elected to select paste tailings disposal in favour of sub-aqueous disposal, these are primarily gold operations with benign tailings.

2.5.7 Tailings Facility Location

There are numerous interdependencies among facilities that dictated the order in which they would be located. VNI located the tailings facility based on results of site surveys, test pits and reviews of past work. Wardrop Engineering Inc. conducted an assessment of potential tailings facility (TF) locations in 2007 and 2008. The Tailings and Ultramafic Waste Rock Management Facility (TWRMF) is located reasonably close to the mill.

The TWRMF location is the preferred location for the following reasons:

- The dam will be cost effective to construct as it is near the open mine, which is earmarked to be the source of the construction materials.
- Co-disposal of tailings and ultramafic waste rock will minimize metal leaching and will increase the stability of the facility.

VNI's closure objective is to design and manage the TWRMF to enable the site to be left without requirements for long-term water treatment.

2.5.8 Camp Location (Operational and Construction Camps)

The following two alternatives were considered for the camp location:

- Off site (South of the property near the existing William River Camp); and
- On site.

VNI selected the on site option as the preferred site for the camp. VNI assumes that the differences in the two locations, from an economic and technical perspective were significant so as other factors, such as health and safety aspects, were considered.

Locating a camp on site would be closer to the working area and will minimize travel time and eliminates the carbon footprint. The chosen site has the advantage that personnel can walk to or from the industrial complex to the camp and additional transportation will not be necessary.

The main disadvantage of locating a camp at the existing site in the vicinity of William River is that it is too far from the Minago site and VNI would have to provide transportation to the project site. This would increase the carbon footprint and may be a problem during winter storm events.

2.5.9 Power Supply

The Minago project will require a continuous power supply for the industrial complex, the camp and supporting facilities. The type of the energy sources used in the operation will have an immediate impact on the capital requirement and the on-going cost of the project. The three energy sources considered for the project and their limitations are as follows:

- Connection to the Main Grid - the connection to the existing Manitoba Hydro power grid will require a high voltage line located approximately 300 metres from the site access. Based on the proximity of the power grid, this option is considered viable.
- Natural gas power generation - previous studies of other mines have indicated that the natural gas and diesel based power generation systems have comparable reliability. However, the diesel generators seem to be 5% to 10% more efficient than natural gas. Diesel fuel is quite expensive and will result in significant operating costs and therefore, the genset option is not considered viable. Natural gas turbines are economical for processes that require high heat or where natural gas supplies, such as pipelines and wells, are nearby. Since there are no gas sources in the area of the project and the diesel-based system provides higher efficiency, the natural gas power generation is not considered viable.
- Hydropower generation - generally hydropower provides the environmentally cleanest operation with the lowest operating cost structure. There are disadvantages; however, such as very high initial capital cost investment, long payback period and complex regulatory requirements with a possible four to five year approval period. In addition, there are no water bodies in the immediate area that can be used for hydropower development. This option is not considered viable.

2.5.10 Site Access Road Location

The Minago Nickel Property (Property) is located 485 km north-northwest of Winnipeg, Manitoba, Canada and 225 km south of Thompson, Manitoba on NTS map sheet 63J/3. The property is approximately 100 km north of Grand Rapids off Provincial Highway 6 in Manitoba. Provincial Highway (PTH) 6 is a paved two-lane highway that serves as a major transportation route to northern Manitoba.

The Minago Project is located just off PTH6 and to access the proposed industrial area will require a maximum of 4 kilometres of road development. The road network to be constructed at the Minago Project will be located in the VNI Mineral Lease Parcel. VNI commissioned environmental baseline studies to determine current baseline conditions. The assessment included air photo and map reviews, and paper route projections. Helicopter reconnaissance and selective ground truthing was conducted. The key design and assessment requirements that were considered included:

- land tenure;

- the avoidance of environmentally sensitive areas such as streams, and wildlife critical habitat areas;
- alignment gradient and length; and
- the presence of bedrock and blasting requirements.

Based on these assessments, VNI optimized the design of the main access road to minimize environmental impacts and construction costs.

Grand Rapids, the closest community to the Property, is located where the Saskatchewan River flows into Lake Winnipeg. In 1996, Grand Rapids had 404 residents (1996 census). The economy of Grand Rapids is based on commercial fishing, hydroelectric generation, tourism, forestry, trapping.

Grand Rapids is served by an RCMP detachment, a nursing station, daily bus and truck transportation to Winnipeg and a 1.02 km grass/turf airstrip in addition to a number of small supply and service businesses.

Provincial Highway 6 crosses a portion of the Property and a network of diamond drill roads enables pickup truck travel on the Property in the winter and all terrain vehicle (Argo) travel in the summer.

The Omnitrax Canada railway line connecting the southern prairie region of western Canada to Churchill, Manitoba (a seasonal seaport) crosses Provincial Highway 6 approximately 60 km north of the Property.

2.6 Project Alternatives

Victory Nickel Inc. sees no feasible alternative to Minago Project. The project is the principle asset of VNI and although there are other mineral deposits in the Minago Area, VNI does not own any interest in them and therefore cannot effect the evaluation of the possible co-development with the Minago deposit. Similarly, currently it is not possible to consider the potential addition of other deposits that may be discovered through exploration. Given the current and future global market for Nickel, the proposed project is the best available option to achieve the business goals of the company.

VNI has assessed a number of alternatives in coming to the proposed design of the Minago Project. The alternatives considered include the various ways that the project could be implemented or carried out, including alternative locations in the project area, routes and methods of development, implementation, and mitigation.

Examining the main project alternatives involved answering the following three questions:

1. What alternatives are technically and economically feasible?
2. What are the environmental effects associated with the feasible alternatives?
3. What is the rationale for selecting the preferred alternative?

Throughout the Minago Project design process, various mining concepts were developed, analyzed, refined and eventually focused down to preferred alternatives. This section describes alternatives that were considered by VNI, and the rationale for selecting the preferred alternative.

The decisions made by VNI and its consultants for the purposes of project design and mine planning are based on feasibility level information. This information provides a reasonable basis for detailed design.

2.6.1 Mining Method

A conventional open pit with seven years of full production and two years of partial ore production life is envisaged after dewatering the overburden and overlying limestone and sandstone. Twelve metre bench heights will be used. A contractor will be employed to remove the overburden and some limestone during the two pre-production years. Equipment will be purchased to utilize the favourable electric power costs in Manitoba. Electric hydraulic shovels will load ore and waste into 218 tonne haul trucks.

Underground operations have been considered but were deemed to be uneconomical due to poor ground control and low-grade aspects. Open pit mining is the only feasible means of extracting the Minago deposit. There will be two products mined from the open pit – frac sand and nickel ore. Frac sand will be mined after the overburden materials (peat and clay and dolomitic limestone) have been removed. The removal of the Frac sand will expose the nickel ore. Open pit mining method is the most optimal extraction method to extract both Frac sand and nickel ore.

2.6.2 Pit Location

The pit is located where the ore is and therefore, there is no viable alternative.

2.6.3 Ore and Waste Haulage

VNI will use 218 tonne trucks to move ore to the mill and waste rock to the waste rock dumps. The 218 tonne trucks are the most economical mode of transportation bearing in mind the waste-to-ore ratio of 6.7 to 1 for mining the nickel sulphide ore and the frac sand. Transportation of ore and waste rock using high capacity equipment is the most viable approach and therefore, there is no viable alternative.

2.6.4 Ore Processing

Conventional flotation will be employed by VNI to process the ore, as there is no viable alternative. The process flowsheet will consist of crushing plant, grinding circuit and a concentrator.

2.6.5 Waste Rock Disposal

The locations of the waste rock dumps and overburden stockpile are selected to optimize hauling costs and are located in the vicinity of the open pit. The waste rock dumps (Dumps #1, 2 and 3) and overburden stockpile locations were selected based on geotechnical investigation results and for the following reasons:

- they are located near the pit to optimize haul distances;
- the overburden is largely clay;
- there will be large waste rock volumes;
- the waste will be Non-Acid Generating (NAG).

The existing facilities have adequate storage capacities for the waste rock that will be generated from pit during development and operational phases and as such, no alternative to the existing infrastructure were examined. During the operations phase, waste rock will be disposed into the dumps. The Overburden, Dolomite and Country Waste Dumps with store approximately 11 Mt of overburden, 111.1 Mt of limestone waste and 122 Mt of granitic (country rock) waste, respectively. Approximately 35.67 Mt of ultramafic waste rock will be co-disposed with tailings in a Tailings and Ultramafic Waste Rock Management Facility (TWRMF). Co-disposal will minimize metal leaching and increase the stability of the tailings management area.

2.6.6 Tailings Disposal

Sub-aerial disposal of liquid tails (slurry) was selected for the property. An alternative method involving the on-land disposal of dry tailings in paste form was assessed. Advantages of paste tailings disposal are:

- A tailings dam does not have to be constructed, removing a significant capital cost item.
- Water does not have to be managed to prevent the oxidation of potentially acid generating materials.

The disadvantages of this option are:

- Dust can be generated from the tailings.
- Pumping is more difficult and expensive than for liquid tailings.
- Operating costs are higher due to the pumping and, potentially, the need to add minimal cement to the tails to retain its form as paste.

The most significant reason for selecting sub-aerial disposal of liquid tailings is that VNI prefers to adopt proven technology rather than embark on a pioneer project. While numerous operations have elected to select paste tailings disposal in favour of sub-aerial disposal, these are primarily gold operations with benign tailings.

2.6.7 Tailings Facility Location

There are numerous interdependencies among facilities that dictated the order in which they would be located. VNI located the tailings facility based on results of site surveys, test pits and reviews of past work. Wardrop Engineering Inc. conducted an assessment of potential tailings facility (TF) locations in 2007 and 2008. The Tailings and Ultramafic Waste Rock Management Facility (TWRMF) is located reasonably close to the mill.

The TWRMF location is the preferred location for the following reasons:

- The dam will be cost effective to construct as it is near the open mine, which is earmarked to be the source of the construction materials.
- Co-disposal of tailings and ultramafic waste rock will minimize the potential for ARD and metal leaching and will increase the stability of the facility.

VNI’s closure objective is to design and manage the TWRMF to enable the site to be left without requirements for long-term water treatment.

2.6.8 Camp Location (Operational and Construction Camps)

The following two alternatives were considered for the camp location:

- Off site (South of the property near the existing William River Camp); and
- On site.

VNI selected the on site option as the preferred site for the camp. VNI assumes that the differences in the two locations, from an economic and technical perspective were significant so as other factors, such as health and safety aspects, were considered.
Locating a camp on site would be closer to the working area and will minimize travel time and eliminates the carbon footprint. The chosen site has the advantage that personnel can walk to or from the industrial complex to the camp and additional transportation will not be necessary.

The main disadvantage of locating a camp at the existing site in the vicinity of William River is that it is too far from the Minago site and VNI would have to provide transportation to the project site. This would increase the carbon footprint and may be a problem during winter storm events.

2.6.9 Power Supply

The Minago project will require a continuous power supply for the industrial complex, the camp and supporting facilities. The type of the energy sources used in the operation will have an immediate impact on the capital requirement and the on-going cost of the project. The three energy sources considered for the project and their limitations are as follows:

- Connection to the Main Grid - the connection to the existing Manitoba Hydro power grid will require a high voltage line located approximately 300 metres from the site access. Based on the proximity of the power grid, this option is considered viable.
- Natural gas power generation - previous studies of other mines have indicated that the natural gas and diesel based power generation systems have comparable reliability. However, the diesel generators seem to be 5% to 10% more efficient than natural gas. Diesel fuel is quite expensive and will result in significant operating costs and therefore, the genset option is not considered viable. Natural gas turbines are economical for processes that require high heat or where natural gas supplies, such as pipelines and wells, are nearby. Since there are no gas sources in the area of the project and the diesel-based system provides higher efficiency, the natural gas power generation is not considered viable.
- Hydropower generation - generally hydropower provides the environmentally cleanest operation with the lowest operating cost structure. There are disadvantages; however, such as very high initial capital cost investment, long payback period and complex regulatory requirements with a possible four to five year approval period. In addition, there are no water bodies in the immediate area that can be used for hydropower development. This option is not considered viable.

Therefore, power required for the operations will come from Manitoba Hydro.

2.6.10 Site Access Road Location

The Minago Nickel Property (Property) is located 485 km north-northwest of Winnipeg, Manitoba, Canada and 225 km south of Thompson, Manitoba on NTS map sheet 63J/3. The property is approximately 100 km north of Grand Rapids off Provincial Highway 6 in Manitoba. Provincial Highway (PTH) 6 is a paved two-lane highway that serves as a major transportation route to northern Manitoba (Figure 2.6-1).

The Minago Project is located just off PTH6 and to access the proposed industrial area will require a maximum of 4 kilometres of road development. The road network to be constructed at the Minago Project will be located in the VNI Mineral Lease Parcel. VNI commissioned environmental baseline studies to determine current baseline conditions. The assessment included air photo and map reviews, and paper route projections. Helicopter reconnaissance and selective ground truthing was conducted. The key design and assessment requirements that were considered included:

- land tenure;
- the avoidance of environmentally sensitive areas such as streams, and wildlife critical habitat areas;
- alignment gradient and length; and
- the presence of bedrock and blasting requirements.

Based on these assessments, VNI optimized the design of the main access road to minimize environmental impacts and construction costs.

Grand Rapids, the closest community to the Property, is located where the Saskatchewan River flows into Lake Winnipeg. In 1996, Grand Rapids had 404 residents (1996 census). The economy of Grand Rapids is based on commercial fishing, hydroelectric generation, tourism, forestry, trapping.

Grand Rapids is served by an RCMP detachment, a nursing station, daily bus and truck transportation to Winnipeg and a 1.02 km grass/turf airstrip in addition to a number of small supply and service businesses.

Provincial Highway 6 crosses a portion of the Property and a network of diamond drill roads enables pickup truck travel on the Property in the winter and all terrain vehicle (Argo) travel in the summer.

The Omnitrax Canada railway line connecting the southern prairie region of western Canada to Churchill, Manitoba (a seasonal seaport) crosses Provincial Highway 6 approximately 60 km north of the Property.

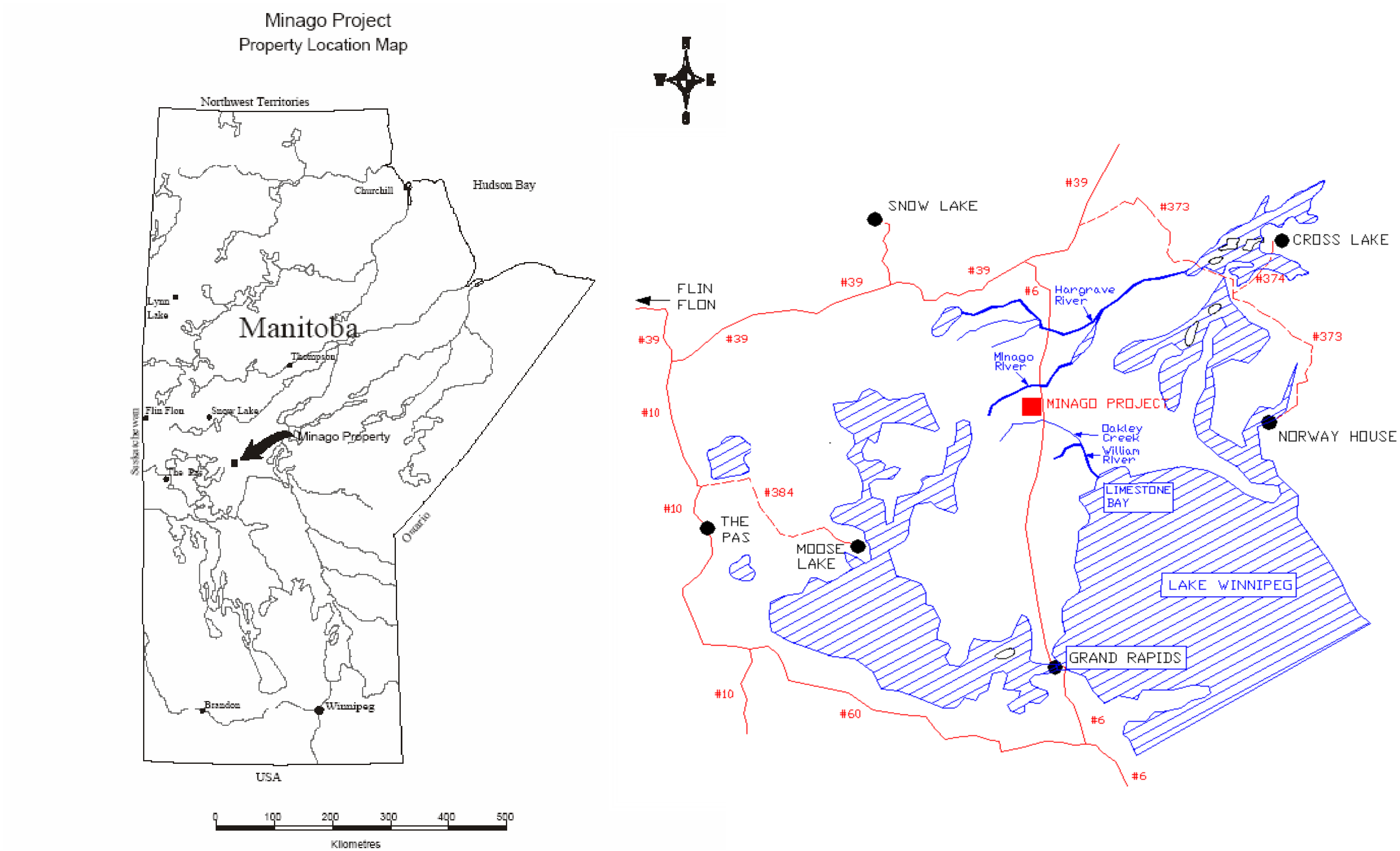


Figure 2.6-1 Site Location Map

2.7 Project Geology

2.7.1 Introduction

Wardrop (2009b) assembled the historic project geological data for the Minago Project to establish a resource estimate that conforms to the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Mineral Resource and Mineral Reserves definitions, referred to in NI 43-101, Standards of Disclosure of Mineral Projects.

Wardrop conducted a mineral resource estimate of the sedimentary and intrusive rocks hosted nickel sulphide mineralization, and the Paleozoic Winnipeg Sandstone Formation immediately above. The estimation was completed for total nickel (Ni%), nickel sulphide (NiS%) and Frac Sand using data from historic and recent drilling.

Wardrop (2009b) estimated that the Minago deposit contains a measured resource of 11 Mt, grading 0.56% Ni, above a cutoff grade of 0.25% Ni. In addition, the deposit contains 43 Mt of Indicated Resource at 0.51% Ni above a 0.25% Ni cutoff grade. An Inferred Resource of 15 Mt at 0.53% Ni above a 0.25% Ni cutoff has also been estimated.

In order to better define the recovery potential on the nickel, Wardrop also estimated the nickel sulphide resource within the total nickel resource. The nickel sulphide resource is contained within the total nickel resource, and is not an additional resource. Nickel in sulphide is considered a more reliable method of determining the nickel content as only a single stage assessment is required. Wardrop (2009b) estimated that the Minago deposit contains a measured resource of 9.1 Mt, grading 0.47% NiS, above a cutoff grade of 0.2% NiS. In addition, the deposit contains 35 Mt of Indicated Resource at 0.42% NiS above a 0.2% NiS cutoff grade. An Inferred Resource of 12 Mt at 0.44% NiS above a 0.2% NiS cutoff has also been estimated (Wardrop, 2009b).

An indicated resource of 15 Mt of Frac Sand within the Winnipeg Sandstone Formation has also been identified. Approximately 10% to 20% of the Frac Sand will report to the 20/40 size fraction, while approximately 68% to 83% will report to the 40/140 size fraction (Wardrop, 2009b).

The Minago deposit has demonstrated potential as a large tonnage low-grade nickel sulphide deposit amenable to open pit, and possibly to underground bulk tonnage mining methods. Significant parts of the deposit below a depth of 400 m require additional drilling to upgrade the resource class from inferred to indicated (Wardrop, 2009b).

The sandstone layer must be removed to access the mineralization within the proposed open pit mine.

2.7.2 Regional Geology

The regional geology comprises the eastern edge of the Phanerozoic sediments of the Western Canada Sedimentary Basin that unconformably overlie Precambrian crystalline basement rocks including the Thompson Nickel Belt. The Western Canada Sedimentary Basin tapers from a maximum thickness of about 6,000 m in Alberta to zero to the north and east where it is bounded by the Canadian Shield. The Property is located near the northeast corner of the Western Canada Sedimentary Basin. At the Minago site, Phanerozoic sediments are comprised of approximately 53 m of Ordovician dolomite underlain by approximately 7.5 m of Ordovician sandstone (Wardrop, 2009b).

The Precambrian basement rocks of the Thompson Nickel Belt form a northeast southwest trending 10 to 35 km wide belt of variably reworked Archean age basement gneisses and Early Proterozoic age cover rocks along the northwest margin of the Superior Province. Lithotectonically the Thompson Nickel Belt is part of the Superior Boundary zone. The Archean age rocks to the southeast of the Thompson Nickel Belt include low to medium grade metamorphosed granite greenstone and gneiss terranes and the high grade metamorphosed Pikwitonei Granulite Belt. The Pikwitonei Granulite Belt is interpreted to represent exposed portions of deeper level equivalents of the low to medium grade metamorphosed granite greenstone and gneiss terranes. The Superior Province Archean age rocks are cut by mafic to ultramafic dikes of the Molson swarm dated at 1883 Ma. Dikes of the Molson swarm occur in the Thompson Nickel Belt, but not to the northwest in the Kisseynew domain. The early Proterozoic rocks to the northwest of the Thompson Nickel Belt comprise the Kisseynew domain that is interpreted to represent the metamorphosed remnants of a back arc or inter arc basin (Wardrop, 2009b).

The variably reworked Archean age basement gneisses constitute the dominant portion (volumetrically) of the Thompson Nickel Belt. The Early Proterozoic rocks that occur along the western margin of the Thompson Nickel Belt are a geologically distinguishable stratigraphic sequence of rocks termed the Opswagan Group (Wardrop, 2009b).

2.7.3 Property Geology

There is no outcrop on the Property. Bedrock geology is interpreted from geophysical data, diamond drill hole core, and regional structural and isopach trends.

2.7.4 Surficial Geology

The surface cover typically comprises 1.0 to 2.1 m of muskeg and peat that is underlain by 1.5 to 10.7 m of impermeable compacted glacial lacustrine clays. The clays are dark brown to grey and carbonate rich (Wardrop, 2009b).

2.7.5 Ordovician Stratigraphy

The Phanerozoic geology comprises the north-eastern edge of the sediments of the Western Canada Sedimentary Basin that unconformably overlie Precambrian crystalline basement rocks, which includes the Thompson Nickel Belt. The Western Canada Sedimentary Basin tapers from a maximum thickness of about 6,000 m in Alberta to zero to the north and east where it is bounded by the Canadian Shield. The Williston Basin strata, in Manitoba, form a basinward-thickening, southwesterly-sloping wedge, with the strata reaching a thickness of 2.3 km in the extreme southwestern corner of the province (Wardrop, 2009b).

Underlying the surficial cover are flat lying Ordovician dolomite and sandstone. The dolomite is fine grained, massive to stratified and varies in color from creamy white to tan brown to bluish grey. Dolomite thickness ranges from 42 to 62 m with thickness increasing southward. The upper 24 m of the formation is stratified with horizontal clay/organic beds 1 to 5 mm in thickness, spaced at intervals ranging from millimetres to one metre. A stratified zone of dolomite breccia and microfracturing characterized by dolomite clasts in a carbonate clay matrix and varying in thickness from 0.3 to 3.0 m is located 15 m to 21 m below the surface of the formation. Scattered throughout the dolomite are occasional soft clay seams ranging from 1 to 2 centimetres (cm) in thickness. The seams may contain dolomite fragments and sand grains and vary in orientation from semi horizontal to semi vertical (Wardrop, 2009b).

The Ordovician sandstone (Winnipeg Formation) occurs stratigraphically below the dolomite approximately 46 to 73 m below surface. The sandstone ranges in thickness from 5.1 to 15.9 m. Cohesiveness varies from consolidated and carbonate cemented to semi consolidated, friable and clay/silt rich to unconsolidated sand. Clay/silt rich zones are brown grey in color while white zones are carbonate cemented (Wardrop, 2009b).

The deposition of the Winnipeg sand in the Williston Basin is thought to be controlled by tectonics in the Williston Basin to the south and the ancestral Sweetgrass Arch (in Saskatchewan) to the west. The bulk of the sediments were derived from the erosion of the Cambrian Deadwood Formation sediments (present in the extreme southwestern portion of Manitoba and into Saskatchewan) and deposition occurred in marine beach to offshore bar environments. The sandstone is distinguished from all other sediments in the basin on the basis of being quartzose and well rounded with variable cementation. The quartz grains are thought to have undergone both fluvial and aeolian transport. They show distinctive frosting caused by wind transport. It has been suggested that these sediments may have been partially derived from the Upper Proterozoic Athabasca Group in northern Saskatchewan (Paterson, 1971; Gent, 1993).

The Ordovician clastic and carbonate sequence in the Minago area was part of a large cratonic depositional platform that extended from the Hudson Platform in the northeast to New Mexico to the south (Norford et al., 1994). The lowermost Paleozoic unit on the Property is the Ordovician Winnipeg Formation (Figure 2.7-1) which is composed of Lower and Upper units in the southern portion of the basin in Manitoba (a lower continuous, poorly consolidated, quartz-rich sandstone sheet overlain by an upper unit of shale with interbedded sandstone). The Lower Unit was deposited in a marine beach to off-shore bar environment. Near-shore, high-energy, shallow-marine to shoreline conditions, possibly at times terrestrial, prevailed in the northern margin of the basin. The northern edge probably approximates the average shoreline position during early Winnipeg time. The Lower Unit rapidly thins to a sandstone sheet to the northern portion of the basin, at the sacrifice of the upper shale unit. The shale is not present in the Minago Sandstone Deposit area.

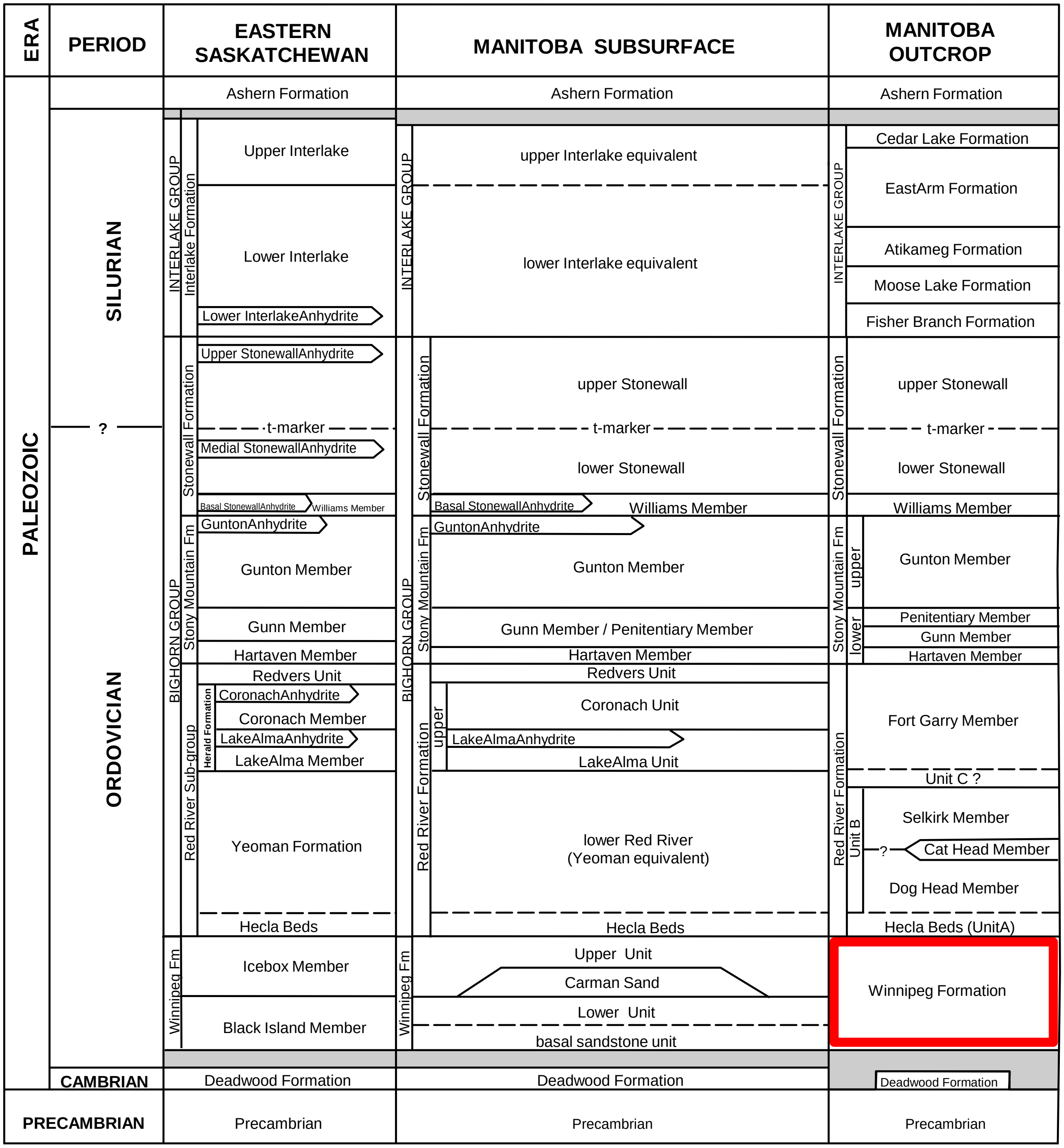
The Winnipeg Formation varies in content from 90% sand to 90% shale (Wardrop, 2009b). The formation has a maximum thickness of 68.6 m in southwestern Manitoba and thins to zero metres to the north, at a rate of least 17% per 100 km, with the sandstone content increasing relative to shale from south to north (Figure 2.7-2). The Winnipeg Formation sandstone that overlays the Minago deposit averages 8.9 m vertical thickness in the proposed pit area, occurring as highly cemented competent rock to loose, and unconsolidated sand size grains (Wardrop, 2009b).

The Ordovician Red River Formation dolomite conformably overlies the Winnipeg Formation in the Project area. There is some debate whether the contact between the Winnipeg and Red River formations is erosional (Norford et al., 1994).

2.7.6 Precambrian Lithologies

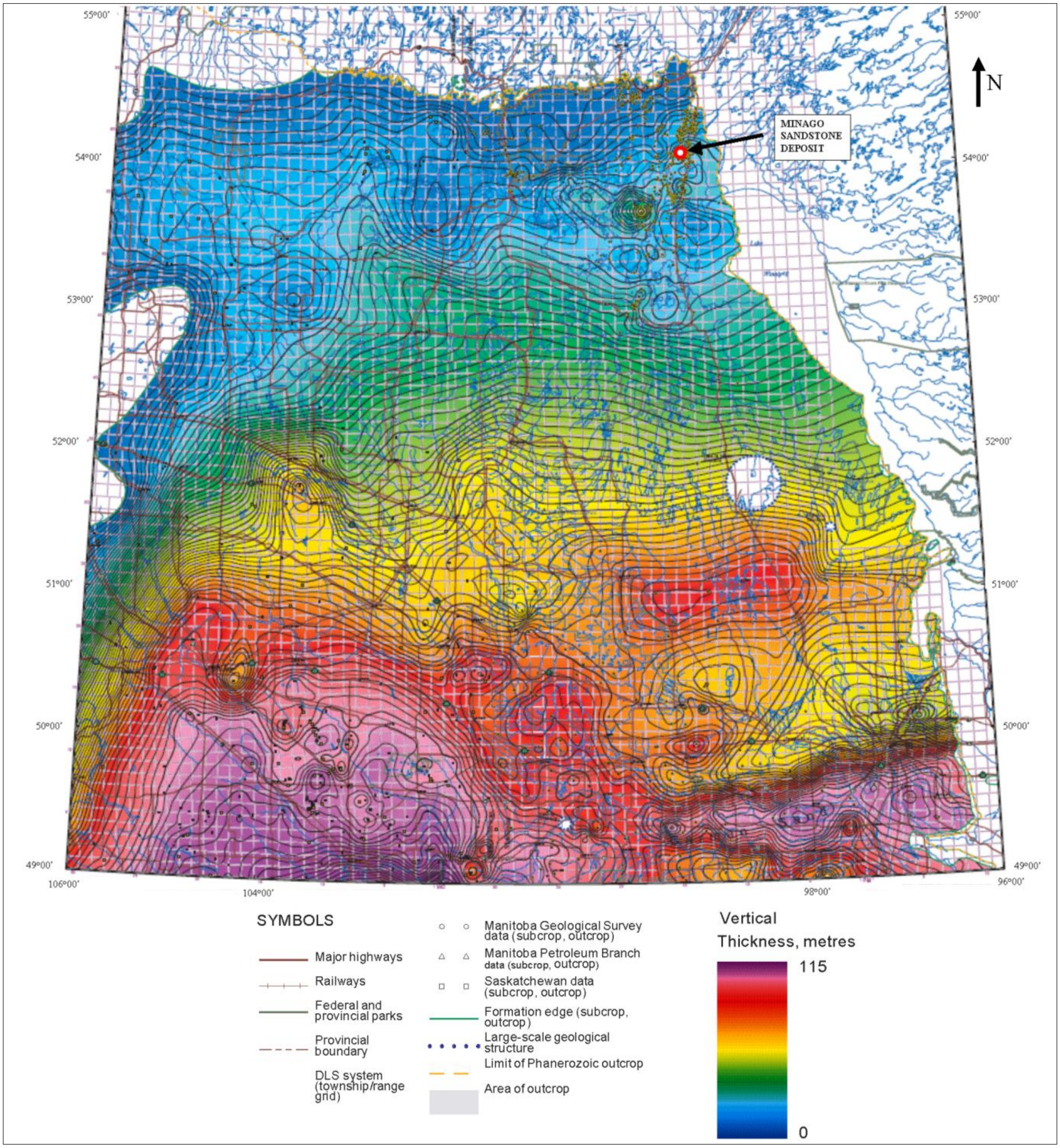
Below the Paleozoic sandstone the Precambrian rocks are intensely weathered typically over distances ranging from 0.6 to 32.8 m sometimes with complete obliteration of original textures. Alteration minerals include kaolin, sericite, chlorite, biotite and carbonate. The alteration is whitish green to bluish green in color, soft, and can be semi consolidated, friable and/or unconsolidated. Weathering persists along zones of intense fracturing down to depths of 60 m below the Paleozoic-Precambrian interface. At depth the weathering is most apparent in granitic rocks where fracture cleavage is prominent resulting in alternating zones of altered fractured rock, and unaltered rock that vary in width from 0.15 m to greater than 3 m. The alteration varies from weak to intense with intensely altered rock being poorly consolidated (Wardrop, 2009b).

Below the Paleozoic sandstone the Precambrian rocks are intensely weathered typically over distances ranging from 0.6 to 32.8 m sometimes with complete obliteration of original textures. Alteration minerals include kaolin, sericite, chlorite, biotite and carbonate (Wardrop, 2009b). The alteration is whitish green to bluish green in colour, soft, and can be semi consolidated, friable and/or unconsolidated. Weathering persists along zones of intense fracturing down to depths of 60 m below the Paleozoic-Precambrian interface. At depth the weathering is most apparent in granitic rocks where fracture cleavage is prominent resulting in alternating zones of altered fractured rock, and unaltered rock that vary in width from 0.15 m to greater than 3 m. The alteration varies from weak to intense with intensely altered rock being poorly consolidated (Wardrop, 2009b).



Source: Nicolas and Barchyn, 2008

Figure 2.7-1 Stratigraphic Column for Manitoba and Eastern Saskatchewan



Basemap Source: Manitoba Science, Technology, Energy and Mines, Manitoba Geological Survey. 2008. Targeted Geoscience Initiative (TGI). Williston Basin Architecture and Hydrocarbon Potential. Ordovician Winnipeg Formation: Isopach. Stratigraphic Map SM2008-OW-I.

Figure 2.7-2 Ordovician Winnipeg Formation Isopach for Manitoba and Eastern Saskatchewan

The Precambrian basement comprises a variety of lithologies briefly described and listed below, in decreasing order of abundance (Wardrop, 2009b):

1. Granitic rocks include granite, granitic gneiss (foliated granite) and pegmatite sills and dikes. Typically grey to pink, the granitic rocks range from almost white to almost red in colour. Grain size ranges from fine to coarse with medium to coarse grain size predominating. Textures vary from massive to strongly foliated. The granitic rocks are mostly potassium (K) feldspar rich, may contain up to 15% biotite and appear to intrude all other rock types.

2. **The fine to coarse grained ultramafic rocks that host the nickel deposit include serpentinized dunite, peridotite (harzburgite, lherzolite, wehrlite) and pyroxenite (orthopyroxenite, websterite, clinopyroxenite)**. The ultramafic rocks dip vertical to near vertical with individual bodies having strike lengths up to 1,525 m and widths up to 457.2 m. Serpentinization varies from intense to weak and appears to decrease with depth, most markedly a change is observed at approximately 400 m below surface. Scoates (2008) attributes the change in serpentinization to a change from retrograde metamorphism (serpentine-talc-tremolite-calcite) in the upper part of the ultramafic to prograde metamorphism (tremolite-hornblende-phlogopite) at depth. Zoned contact alteration on a centimetre to metre scale occurs adjacent to granite and some fractures. From most intense (adjacent to granite or fracture) to least intense (furthest from granite or fracture) the alteration typically comprises biotite/phlogopite-chlorite-tremolite. Varying abundances (<1% to >50%) of fine to coarse grain pseudomorphs of olivine, orthopyroxene and clinopyroxene occur over core intervals ranging from several centimetres to several tens of metres. Magnetite concentrations up to 50% occur locally. Sulphide tenor is usually <15%.

3. Metavolcanic rocks, interpreted to be Bah Lake Formation, include chlorite-biotite schist and amphibolite. Amphibolite is dark green to black, fine to medium grained, foliated and lineated.

4. Metasedimentary rocks, interpreted to be Pipe Formation, comprise sillimanite paragneiss, siliceous sediments, skarn, iron formation, graphitic sediments, semi pelite and calc silicate. Distinctive minerals include graphite, sillimanite, garnet, diopside, carbonate, muscovite and very fine grain quartz. Sulphide facies iron formation comprises semi massive to massive pyrite and pyrrhotite, sometimes nodular, and associated with detrital metasediments often containing siliceous fragments and includes sulphide breccia in zones of cataclastic deformation.

5. Molson dikes and sills that are olivene rich.

The Precambrian lithologies have undergone complex multiphase ductile and brittle deformation. Interpretations of magnetic data suggest that the ultramafic rocks containing the Minago deposit have undergone dextral strike slip fault movement which resulted in a large Z shaped drag fold and that the deposit flanks the axial plane of an eastern limb. Vertical longitudinals of the mineralized zones indicate that the folded limb plunges steeply towards the southeast (Wardrop, 2009b).

Observations of the mineralized lenses indicate lateral/vertical displacement resulting in the development of drag folds and boudins. In some cases, the mineralization appears to have been folded creating ore zones with true widths over 24.4 m or has been folded and pulled apart creating two parallel zones of the same lense (Wardrop, 2009b).

Cataclastic deformation with lateral and vertical displacement is indicated by fault gouge and fault breccia zones in both ultramafic rocks and granitic rocks. These zones range in width from 1 mm to 10 cm, are subvertical to vertical, and parallel the trend of the ultramafic rocks. Fault gouge is characterized by clay rich seams with or without fragments. Fault breccia is characterized by angular fragments in a matrix of serpentine, carbonate and clay minerals (Wardrop, 2009b).

Cataclastic zones in serpentinitized ultramafic rocks are grey in color, soft, and associated with massive and fine grained units, whereas in granitic rocks they are red to brown in color and associated with fracture cleavage. Cataclastic deformation confined to relatively fresh ultramafic rocks has a ground appearance, is brittle and poorly consolidated. Mylonite has an aphanitic to vitreous texture and is light to dark in color. Mylonitization in granitic rocks is proximal to contacts between the granitic rocks and serpentinized ultramafic rocks (Wardrop, 2009b).

Fracture cleavage occurs adjacent to zones of cataclastic deformation and folding. More readily observed in granitic rocks, the fractures also occur in serpentinites as open fractures and minor shears that are schistose and contain talc, chlorite, phlogopite and biotite. Two fracture cleavage orientations are indicated: parallel to foliation; and acute to approximately perpendicular to foliation. Fractures filled with carbonate and serpentine are cohesive. Fractures filled with sericite and clay minerals lack cohesion and possess slickensides (Wardrop, 2009b).

Sedimentary and intrusive rock hosted nickel sulphide mineralization are recognized as two distinct and economically important deposit types in the Thompson Nickel Belt. Often intimately spatially related due to interaction of sedimentary, magmatic, metamorphic and deformational processes, the deposit types can be distinguished on the basis of field observations, structural, textural, mineralogical and chemical criteria (Wardrop, 2009b).

Sulphide enrichment also occurs in pegmatites and breccias derived from existing sedimentary or magmatic sulphides (Wardrop, 2009b).

The Ospwagan Group hosts the nickel deposits of the Thompson Nickel Belt. Within the Ospwagan Group almost all of the nickel deposits of the Thompson Nickel Belt are found within lower Pipe Formation (Wardrop, 2009b).

Bleeker and Macek proposed a stratigraphic nomenclature for the Proterozoic rocks within the Thompson Nickel Belt that is summarized in the stratigraphic column shown in Figure 2.7-3.

The rocks of the Thompson Nickel Belt have been complexly folded. Three major periods of folding are commonly recognized. The earliest structures due to compressional tectonism are isoclinal F1 folds that may be of regional extent. F1 preceded the emplacement of Molson dikes. The metamorphic regime during F1 is unknown. F1 is overprinted by F2 isoclinal folds that developed under high temperature and caused folding of the Molson dikes. The thermal peak of regional metamorphism overprinted F2. At least 30 million years later and at much lower temperatures intense sinistral transpression produced high amplitude, nearly upright, doubly plunging F3 folds that transposed the pre-existing recumbent fold pile into a steep gneiss and schist belt (Wardrop, 2009b).

The main phase of mylonitization occurred late during or overprints F3 and is confined to shear zones that tend to be parallel to the steeply dipping limbs of the upright F3 folds (Wardrop, 2009b).

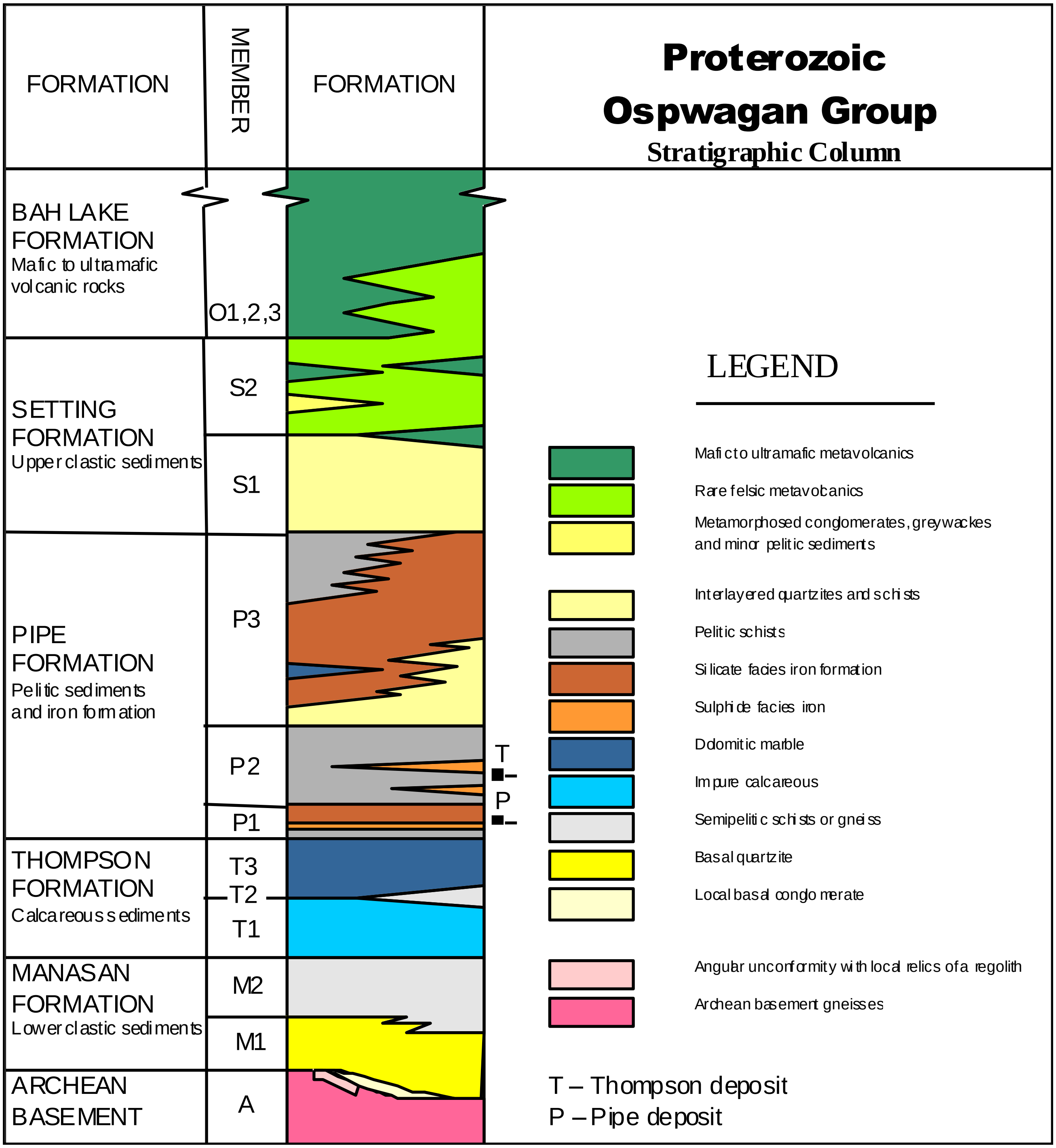
2.7.7 Sedimentary Sulphide Mineralization

Sedimentary sulphides may be barren or enriched in nickel. Barren sulphides characterized by nickel concentrations below 500 parts per million (ppm) occur beyond the immediate vicinity of significant nickel enriched zones. Sedimentary sulphides enriched in nickel by later magmatic processes are visually indistinguishable from barren sedimentary sulphides but occur in close proximity to more significant nickel enriched zones (Wardrop, 2009b).

The dominant geological feature of economic interest underlying the Property is a series of boudinaged nickeliferous ultramafic bodies folded in a large Z shaped pattern. The ultramafic bodies contain intraparental magmatic nickel sulphide mineralization, and intrude mafic metavolcanic and metasedimentary rocks interpreted to be lower Pipe Formation stratigraphy (Wardrop, 2009b).

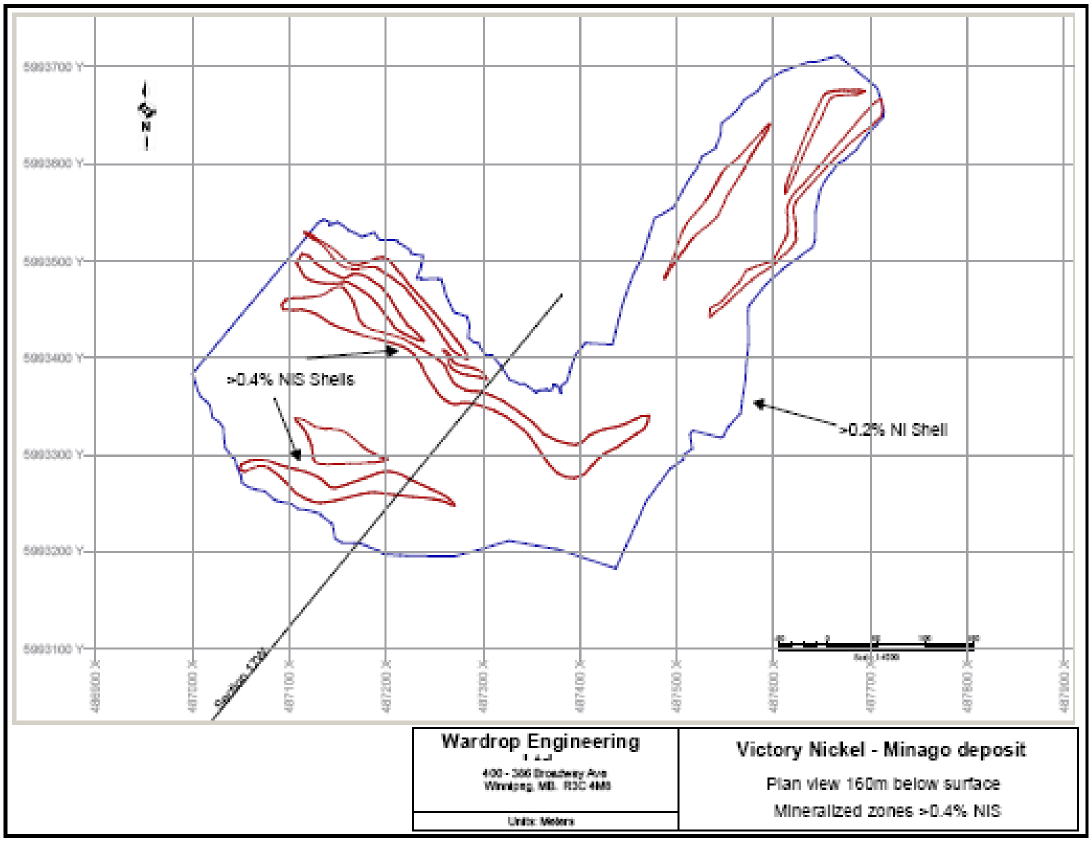
Within the ultramafic rocks, the nickel sulphides are concentrated in several tabular lenses that parallel the trend of the ultramafic bodies (Figures 2.7-4 and 2.7-5). Lower grade nickel occurs between and adjacent to the higher grade lenses. Typically sulphides are fine grained varying in size from <0.5 to 4 mm (generally 1 to 2 mm) and range in volume from 2 to 15% (generally 2 to 7%). The sulphides predominantly occur as disseminated crystals, small aggregates (<5 mm) and occasionally are net textured. The dominant sulphide species are nickel bearing pentlandite with lessor violarite and millerite. Minor amounts of pyrite, pyrrhotite and chalcopyrite are present (Wardrop, 2009b).

Graphitic, coarse grained and sometimes nodular sedimentary and extraparental nickeliferous sulphide mineralization occurs sporadically along the southeast margin of the Minago deposit.



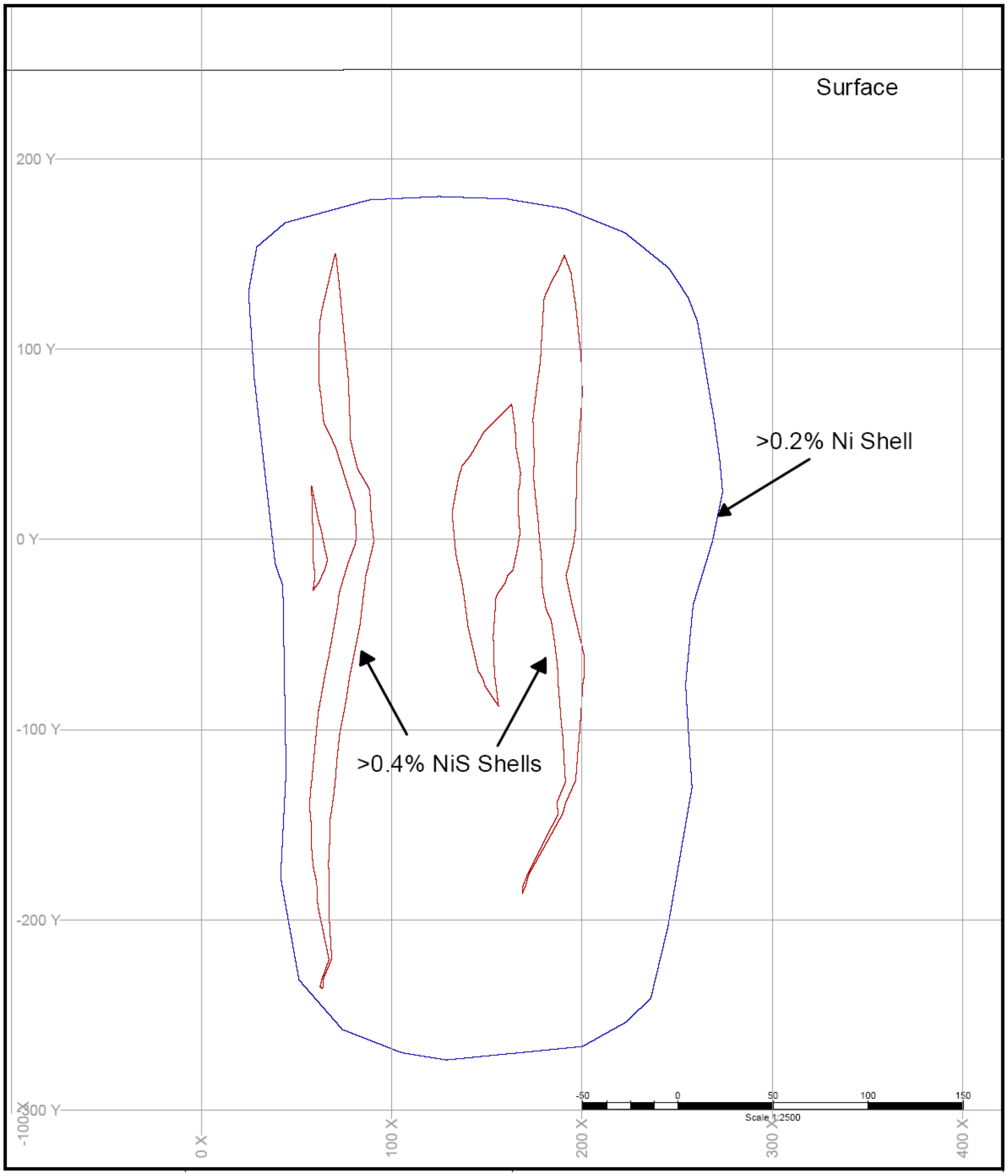
Source: Wardrop, 2009b (Secondary source: Bleeker and Macek, 1990)

Figure 2.7-3 Proterozoic Ospwagan Group - Stratigraphic Column



Source: Wardrop, 2009b

Figure 2.7-4 Minago Deposit at 160m Below Surface Showing Mineralization >0.4% NiS (red) in Lower Grade Envelope



Source: Wardrop, 2009b

Figure 2.7-5 Section 17W Showing Mineralization >0.4% NiS (red) in Lower Grade Envelope

Limited shallower diamond drilling in the North Limb has intersected a number of boudinaged ultramafic bodies that contain nickel mineralization similar to that at the Minago deposit (Figure 2.7-6).

The southern part of the claim block has not had any work done on it since 1972. A number of intersections of nickel bearing ultramafic rock have been encountered (Figure 2.7-7). Based on the limited available information, the nickeliferous ultramafic rocks appear to be irregularly distributed.

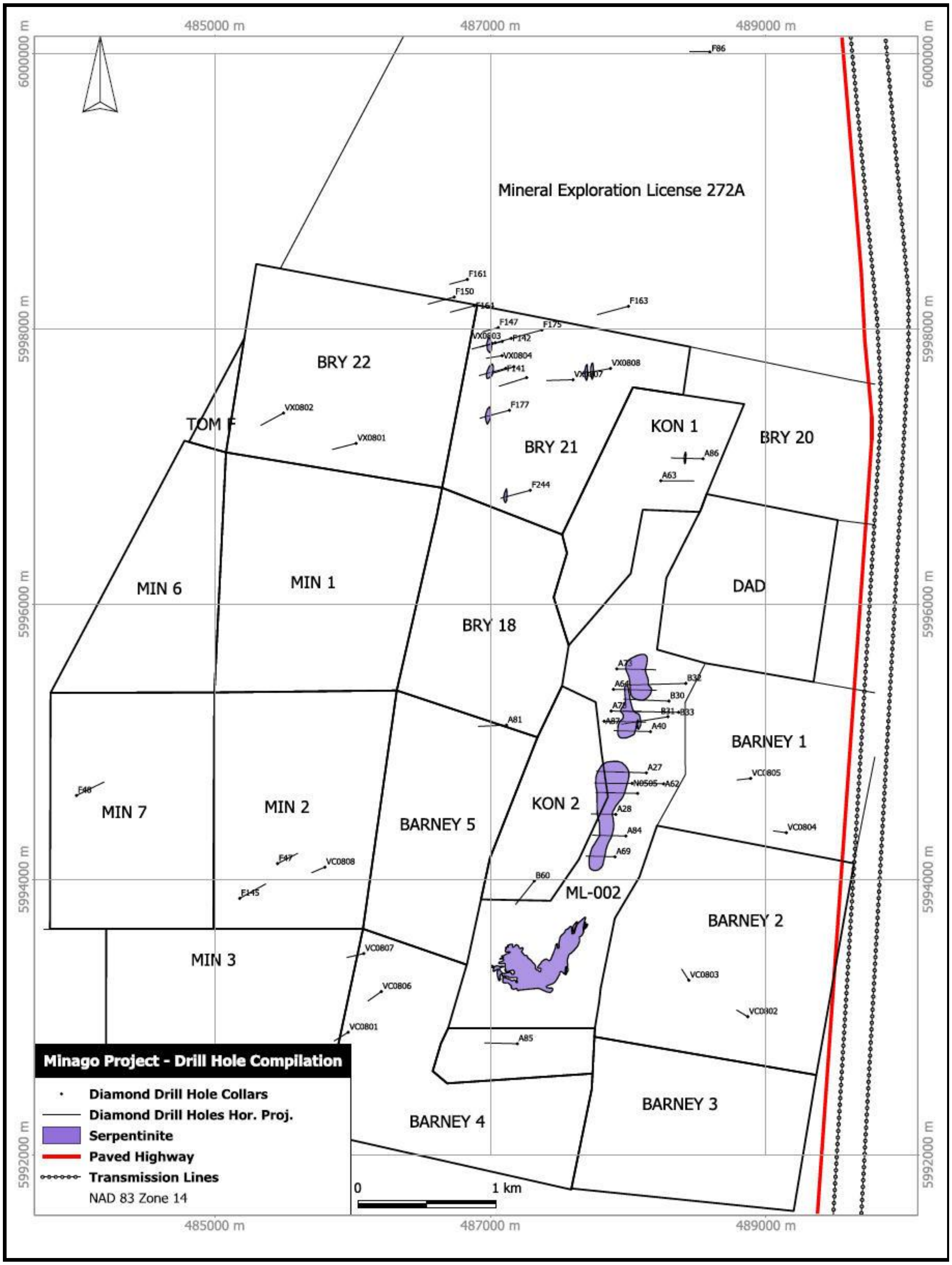
2.7.8 Magmatic Sulphide Mineralization

Magmatic nickel sulphide mineralization can be intraparental or extraparental based on whether it occurs within or external to the ultramafic parent rocks. Typically massive, extraparental mineralization occurs as pods and lenses of variable size within host pelitic schist adjacent to ultramafic boudins. The interpretation of the magmatic affinity of the extraparental mineralization is based on certain shared chemical characteristics with the intraparental mineralization. Intraparental mineralization occurs as lower abundances of interstitial sulphide and semi massive to massive concentrations of sulphide in veins and breccias all within ultramafic rocks (Wardrop, 2009b).

2.7.9 Exploration History of the Minago Deposit

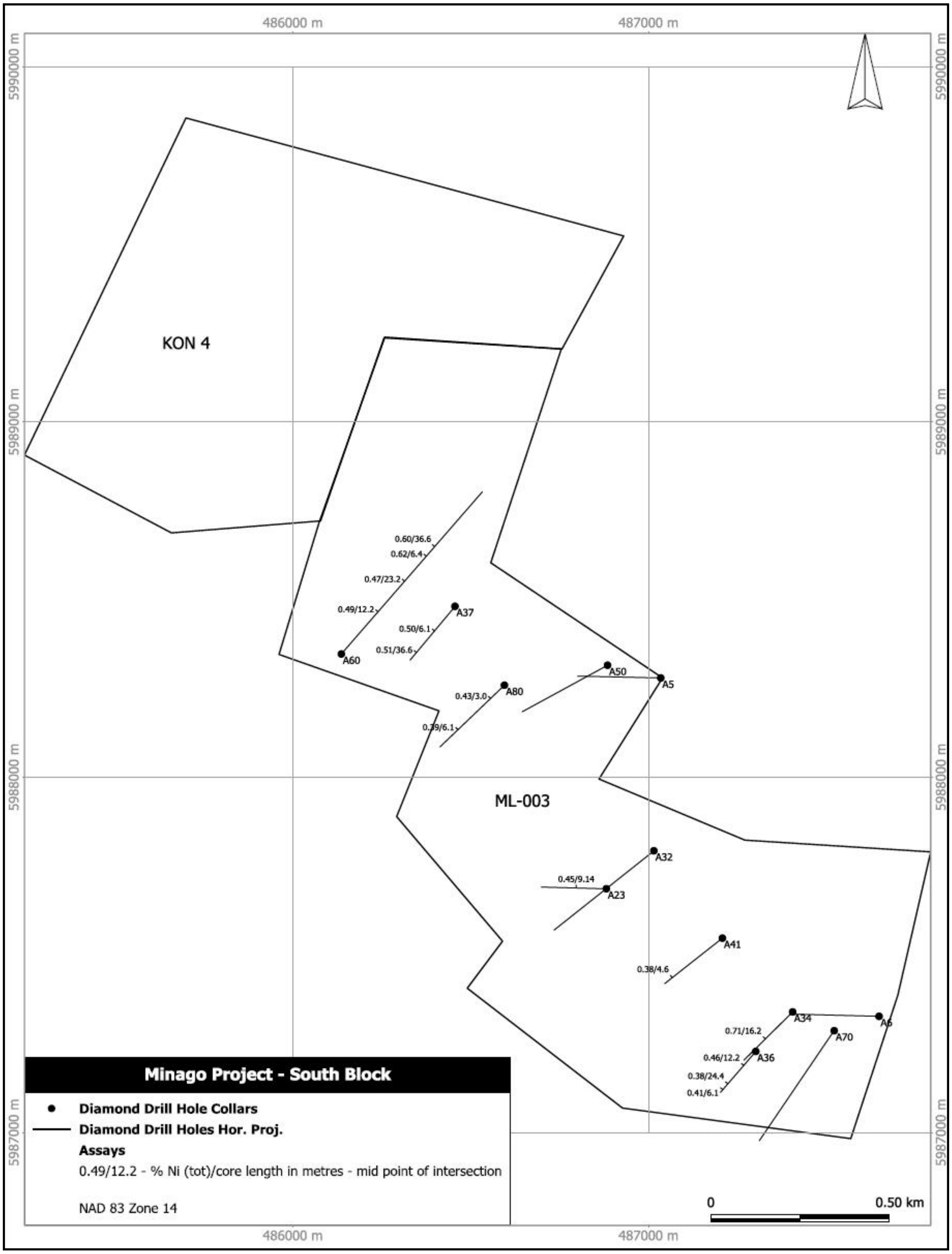
Geophysical Reservation 34 (GR 34) covering an area of 19.2 km by 38.4 km was granted to Amax Potash Ltd. (Amax) on November 1, 1966 for a period of two years and extended in 1968 to April 30, 1969 (reference to Amax in this report includes the subsidiaries and successor companies of Amax Potash Limited, namely Amax of Canada Limited, 121991 Canada Limited and Canamax Resources Inc.).

In March 1969, Amax converted the most prospective area of GR 34 to 844 contiguous claims; in April of 1969, an additional 18 claims were staked. In 1973, the claims covering ground deemed to have the most potential for economically viable nickel mineralization were taken to lease status as Explored Area Lease 3 (North Block) and Explored Area Lease 4 (South Block). In an agreement dated December 12, 1973, Granges Exploration Aktiebolag (Granges) was granted an option on the Explored Area Leases. Reference to Granges in this report includes the subsidiaries and successor companies of Granges Exploration Aktiebolag namely Granges



Source: Wardrop, 2009b

Figure 2.7-6 North Limb (from Victory Nickel)



Source: Wardrop, 2009b

Figure 2.7-7 South Block Nickel Intersections (from Victory Nickel)

Exploration Ltd. and Granges International Ltd. In 1977, Granges became a passive partner with a 25% interest and a 0.5% NSR royalty in the leases. On May 18, 1989 Black Hawk Mining Inc. (Black Hawk) purchased the Amax interest in the explored area leases. On August 2, 1989 Black Hawk purchased the Granges interest and NSR royalty in the explored area leases. On April 1, 1992 Explored Area Lease 3 and Explored Area Lease 4 were converted to Mineral Lease 002 and Mineral Lease 003 respectively. On March 18, 1994 a portion of Mineral Lease 002 was converted to mineral claims KON 1, KON 2 and KON 3, also on March 18, 1994 a portion of Mineral Lease 003 was converted to mineral claim KON 4. On November 3, 1999 Nuinsco purchased the Black Hawk interest in the Property subject to a graduated NSR royalty based on nickel prices. In 2008, Victory Nickel acquired Independent Nickel and effectively eliminated the royalty (Wardrop, 2009b).

2.7.9.1 Amax Exploration Work from 1966 to 1972

Amax conducted a regional scale exploration program on the southern extension of the Thompson Nickel Belt, and concluded that the corporate threshold for deposit size justifying production would not be achieved on the Property. A brief summary of work conducted on the Property by Amax follows (Wardrop, 2009b):

- an AFMAG airborne survey with nominal 1,609 m line spacing;
- a helicopter airborne magnetic survey with nominal 402 m line spacing;
- a Turair electromagnetic survey;
- linecutting at 305 m line spacing with ground geophysical surveys including; magnetic (Askania magnetometer), electromagnetic (Radem VLEM), dipole-dipole induced polarization (McPhar) and gravity surveys;
- drilling of eighteen holes plus one wedged hole were diamond drilled at the deposit;
- drilling of fourteen holes elsewhere on the North Block (Figure 2.7-6); and
- diamond drilling of twelve holes on the South Block (Figure 2.7-7).

2.7.9.2 Granges Exploration Work from 1973 to 1976

Granges focused their efforts on the Minago Nickel deposit conducting resource estimates, mining, metallurgical and milling studies. Eight holes were diamond-drilled at the Minago Nickel Deposit; limited in-hole surveys were also conducted (Wardrop, 2009b).

Granges concluded that the deposit was sufficiently confirmed and that further delineation and exploration should be conducted from underground workings (Wardrop, 2009b).

2.7.9.3 Black Hawk Exploration Work from 1989 to 1991

Black Hawk conducted a deep penetrating ground electromagnetic survey, resource estimates, mining, metallurgical and milling studies. A helicopter-borne electromagnetic and magnetic survey covering the Property was obtained from Falconbridge Limited and interpreted (Wardrop, 2009b).

Forty holes were drilled in the vicinity of the deposit. Collars were surveyed for location and in-hole orientation surveys were conducted on most holes using an ABEM fotobor. Five holes were diamond drilled elsewhere on the North Block (Wardrop, 2009b).

2.7.9.4 Nuinsco Work in 2005

Between mid-January, early April 2005, 3,027.78 m were drilled in 6 holes (N-1 to N-6). All holes, except N-5, were drilled to verify earlier diamond drill results, provide infill data and extend previously intersected mineralizaton. Hole N-5 was drilled 900 m northeast of the Minago deposit to explore the North Limb (Wardrop, 2009b).

In-hole surveys were performed by Major Drilling and Reflex Instrument North America. During the drilling of each hole, Major Drilling collected Reflex EZ-Shot data approximately every 50 m down the hole. Reflex EZ-Shot measures the following six parameters in one single shot: azimuth, inclination, magnetic tool face angle, gravity roll angle, magnetic field strength and temperature. However, the azimuth data is not reliable due to the magnetic properties of the rocks (Wardrop, 2009b).

Reflex Instrument North America personnel traveled to the property on three occasions to conduct surveys using the Reflex Maxibor. The Reflex Maxibor calculates the spatial coordinates every three metres along the drill hole path based on optical measurements of dip and direction changes. All holes except N-3 were surveyed. Holes were not surveyed in their entirety due to considerable difficulty in getting the instrument down the hole inside the BQ rods (Wardrop, 2009b).

Drill hole collars from N-1 to N-6 were surveyed for location, azimuth and dip by Pollock and Wright, Land Surveyors utilizing a Trimble RTK5700 dual frequency GPS survey instrument (Wardrop, 2009b).

Each hole was logged for rock quality designation (RQD). Samples were shipped by commercial trucking to the ALS Chemex laboratory to Thunder Bay, Ontario for sample preparation; thereafter the pulps were shipped by ALS Chemex to their laboratory for analysis (Wardrop, 2009b).

2.7.9.5 Nuinsco Work in 2006

Two holes totalling 1,533.57 m were drilled from March 4 to April 21, 2006. The drilling was undertaken in order to confirm and upgrade the resource estimates of deposit, enable geotechnical observations and measurements required to revise preliminary open pit shell designs, and provide additional material for metallurgical testing (Wardrop, 2009b).

It was necessary during the drilling of the Minago Deposit to employ NQ size rods to drill through the Ordovician strata and into the upper Precambrian basement. The remainder of the drill holes were reduced to BQ rod size. Due to the drill hole lengths required to cut a section through the sand deposit, changing the drill bit midway through each hole was necessary. If it were not for the reduction in rod size shortly below the unconformity, removal of the NQ rods from the hole to change a bit would have invariably resulted in collapse of unconsolidated Winnipeg Formation sand into the hole, and the near certain loss of the hole below the unconformity (Wagg, 2006).

In-hole surveys were performed by Major Drilling personnel utilizing a Reflex EZ-Shot instrument. During the drilling of each hole the drill crew collected Reflex EZ-Shot data approximately every 50 m down the hole (Wardrop, 2009b).

Drill hole collars were surveyed for location, azimuth and dip by Pollock and Wright, Land Surveyors, with a Trimble RTK5700 dual frequency GPS survey instrument. Dip values for the drill holes are not valid due to droop in the survey rod however location co-ordinates and azimuths are considered reliable (Wardrop, 2009b).

Drill project supervision and core logging and assay interval selection were conducted by the project geologist. Geotechnical parameters recorded included: RQD values, core recovery, fracture pattern orientations, abundance, nature (open or filled), type of fill, marginal alteration, cohesiveness, wetness, and strength estimates utilizing the R0-R6 scale wherein R0 represents extremely weak rock, and R6 represents extremely strong rock (Wardrop, 2009b).

An industry-standard point load test apparatus manufactured by Rokworth Corporation was employed for unconfined compressive strength testing that was routinely undertaken every 3.0 m for all drill core recovered. The lithology tested was recorded as well as the failure point in pounds per square inch. Diametral and less frequent axial tests were recorded for each lithology (Wardrop, 2009b).

In 2006, an NI 43-101 compliant mineral resource estimate for the Minago deposit was conducted by P. Vasak, P.Eng. of Mirarco. The Mirarco procedures and results are contained in a report titled “Resource Modelling of the Mineralized Zone of the Minago Nickel Deposit”, December 24, 2004 (Vasak, 2004). The Mirarco resource estimates were undertaken on behalf of and supervised by independent Qualified Person P.J. Chornoby, P.Geo., and P. Jones, P. Geo., Vice President of Exploration, Victory Nickel. This resource estimate summarizes the results of exploration conducted during the period from 1966 to 1991 and the work conducted by Nuinsco from 2004 to October 31, 2006. The resource model is for all mineralized zones in the Minago Deposit to a depth of 944.88 m below surface and provides resource classification and block models for deposit evaluation purposes. The primary scope was to build a resource block model based on a 0.2% Ni cut-off resource wireframe. The mineral resource estimates were optimised to evaluate resources mineable using open pit techniques to a depth of 411.5 m below surface based on the analysis of a qualified mining engineer (Wardrop, 2009b). The model utilizes a block size of 7.6 m x 7.6 m x 7.6 m.

2.7.9.6 Victory Nickel Work in 2007

Victory Nickel carried out a diamond drill program on the property commenced by Nuinsco in January and completed by Victory Nickel in May 2007. Major Drilling was contracted to perform the drilling.

The 2007 drill program was designed to upgrade inferred resource estimates above the pit bottom used in the PEA study (Wardrop, 2006). Mirarco and Victory designed 29 holes for this purpose. Five holes were drilled to provide material for metallurgical testing. These five holes were logged for geology, sample intervals were selected and tagged but the core was not split (Wardrop, 2009b). Wardrop designed ten of the holes specifically to examine final pit wall stability and logged a total of 24 holes for comprehensive geotechnical data complete with point load testing. Orientated core measurements were also performed on portions of two holes (Wardrop, 2009b).

As per industry norms each hole was logged, with sample intervals based on the following hierarchy (Wardrop, 2009b):

- rock type;
- alteration (style and intensity); and
- sulphide content (type and abundance).

A total of 7,260 nickel samples representing 13,217 m of NQ core were selected from the holes drilled in 2007. Five sandstone samples were also submitted for frac sand quality analysis and an additional 25 sandstone samples were identified and submitted for density measurement (Wardrop, 2009b).

All of the ultramafic rock intersected in each drill hole was sampled as was the immediately adjacent barren and included barren rock. Nickel samples varied in length from 0.13 m to a maximum of 3.45 m with a mean sample length of 1.21 m. Core recoveries in the Precambrian were generally 95% to 100% for each 3.0 m run with only rare intervals of lost core. Sandstone quality samples varied in length from 7.86m to 13.41m, and were limited to drill holes with >90% recovery (Wardrop, 2009b).

All ultramafic lithologies encountered were sampled and assayed except for some composited material required for crushing and grinding testwork (Wardrop, 2009b).

Drill hole collars were surveyed for location by Pollock and Wright of Winnipeg. Ongoing in-hole surveys were performed by Major Drilling personnel every 50 metres in all holes using a Reflex Easy Shot instrument. In addition, holes greater than 200 m long were in-hole surveyed by Victory contractors/employees using a Reflex Maxibor II instrument. A total of 3,752 measurements were taken (Wardrop, 2009b).

2.7.9.7 Victory Nickel Work in 2008

Victory Nickel conducted a diamond drill program including 26 holes on the property between January and May 2008. The 2008 drill program was designed to upgrade inferred resource estimates below the pit bottom used in the PEA study (Wardrop, 2006). Ten holes were designed by Wardrop for this purpose. Victory Nickel planned eight holes to explore the property where Wardrop proposed future mine construction, and eight holes to satisfy the expenditure requirement of at least $500,000 for the Xstrata option on claims BRY 18, BRY 20, BRY 21, BRY 22, TOM F and DAD, illustrated in Figure 2.7-6 (Wardrop, 2009b).

Twenty six holes totalling 11,748 metres were drilled in 2008, 10 holes totalling 7,505 m were targeted on the known Minago mineralized zone, 2,517 m were drilled to satisfy the requirements of the Xstrata option, and the remaining 1,726 m were drilled for property exploration (Wardrop, 2009b).

A total of 2,106 nickel samples representing 2,783.6 m of NQ core were selected from 10 holes drilled in 2008. The sample intervals were determined by the geologist during core logging. Twenty one sandstone samples were collected for density measurements.

As per industry norms each hole was logged, with sample intervals based on the following hierarchy (Wardrop, 2009b):

- rock type;
- alteration (style and intensity); and
- sulphide content (type and abundance).

All of the ultramafic rock intersected in each drill hole was sampled as was the immediately adjacent barren and included barren rock. Core recoveries in the Precambrian were generally 95% to 100% for each 3.0 m run with only rare intervals of lost core. Samples varied in length from 0.14 m to a maximum of 4.4 m, with a mean sample length of 1.32 m (Wardrop, 2009b).

The program was also designed to provide material for metallurgical testing, especially as the serpentinization appears to decrease with depth and there may be an accompanying change in the metallurgical response (Wardrop, 2009b).

Drill hole collars were surveyed for location by Pollock and Wright of Winnipeg. Ongoing in-hole surveys were performed by Major Drilling personnel every 50 m in all holes using a Reflex Easy Shot instrument. In addition, holes greater than 200 metres long were in-hole surveyed by Victory contractors/employees using a Reflex Maxibor II instrument. A total of 2,109 measurements were taken (Wardrop, 2009b).

As the holes were being drilled, the core was transported to Victory’s core room in Grand Rapids, Manitoba and securely stored indoors for processing. The core was photographed and logged initially for geotechnical data, then the core was subsequently logged for lithology, alteration and mineralization (Wardrop, 2009b).

2.7.10 Data and Tools Used in the Mineral Resource Estimate

A detailed mineral resource estimate of the frac sand and the nickel sulphide mineralization at the Property was prepared by Wardrop (2009b). The estimation was completed for total Ni%, NiS% and frac sand quality using data from historic and recent drilling.

Gemcom version 6.1.3 was used for the resource estimate (wireframing and block modeling) in combination with Sage 2001 for the variography. WinStat software was used to identify the regression curves for grade-density and NiS-Ni relationships. Historically, total nickel analysis was carried out on the core. Additional analysis for nickel sulphide was requested by Wardrop, so that the recoverable portion of the nickel could be estimated.

Traces of Cu, Pt and Pd are also present at Minago. The Cu model was estimated using the same parameters as the NiS model. Cu grades are very low, but may become part of a smelter credit.

The frac sand resource was estimated using all the available data. The size and continuity of the Winnipeg sandstone were well established by 141 drill holes in and around the proposed pit shell. Frac sand quality data and metallurgical testing data were available for 5 of the holes.

The metallurgical test program at Minago established that the Minago deposit contains a significant amount of nickel in the form of nickel silicates, which are not recoverable by froth flotation (Wardrop, 2008a). Thus, the deposit defined using the total nickel assay is not reliable in terms of determination of mineable sections, pit design, and economic analysis of the mining and mill operation. Therefore, Minago's head grade-recovery curve was based on the grade of nickel sulphide (Wardrop, 2008a).

2.7.10.1 Total Nickel

There were 78 drillholes from the historical dataset that were used for the resource estimation (Wardrop, 2009b). An additional 44 drillholes from the 2006-07 winter drill campaign, and 10 holes from the 2008 program, were added. A total of 132 drillholes in the vicinity of the sulphide mineralization were used for the resource estimation (Wardrop, 2009b). The drillhole database comprised of collar, survey, lithology and assay information as summarized in Table 2.7-1.

Table 2.7-1 Total Records in Database

	Drillholes	Coordinates	Survey	Lithology	Assays
Records	132	132	8,334	3,620	19,875

Source: Wardrop, 2009b

2.7.10.1.1 Total Nickel Assays

A total of 19,875 assay intervals from 132 drillholes were selected and defined the zone of mineralization on the deposit (Table 2.7-2). Data analysis was conducted by creating probability and histogram plots of the data (Figure 2.7-8). The probability plot seems to exhibit a non-lognormal population. Probably two populations are seen in the plot as this may tie in with the serpentinite/granite mix (Wardrop, 2009b). Figure 2.7-9 shows a boxplot of nickel assays by rock type inside the selected mineralized zone (wireframe).

Non-assayed intervals were assigned a value of zero and are included with the assayed values.

Table 2.7-2 Ni% Assay Statistics

	Ni%
Minimum	0.000
Mean	0.280
Median	0.130
Maximum	5.860
N	19,875

Source: Wardrop, 2009b

2.7.10.2 Nickel Sulphide

The NiS% database consists of 4,557 assays and is summarized in Table 2.7-3.

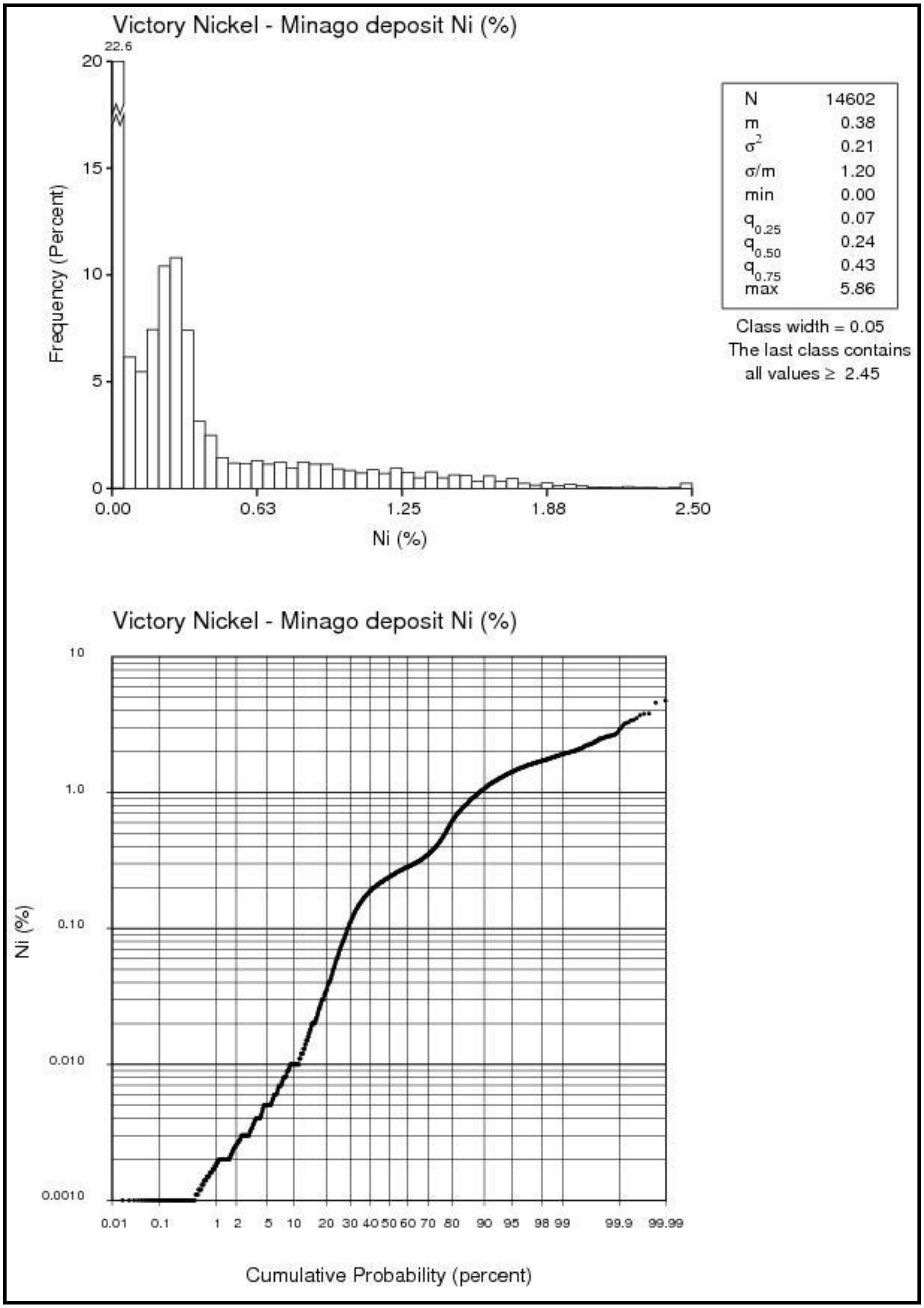
2.7.10.2.1 Nickel Sulphide Assays

A total of 4,557 NiS% assays were available compared to 14,829 Ni% assays. Data analysis was conducted by creating probability and histogram plots of the data. The probability plot seems to exhibit a non-lognormal population (Figure 2.7-10). Probably two populations are seen in the plot, as this may tie in with the serpentinite/granite mix (Wardrop, 2009b). Figure 2.7-11 shows a boxplot of NiS% assays by rock type, where 3,071 out of 3,298 NiS% assays were in the serpentinite unit.

2.7.10.2.2 NiS/Ni Ratios

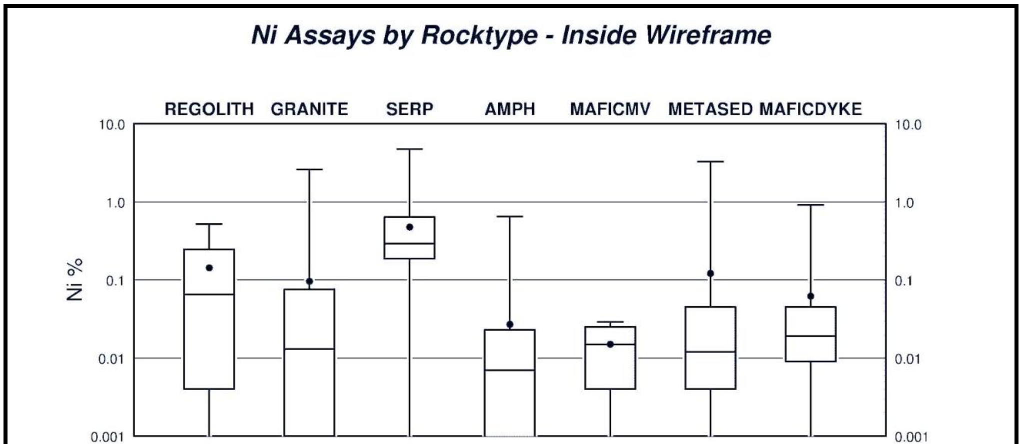
Wardrop examined in detail the relationship between the individual NiS to Ni assays. Ni to NiS scattered plots were created. Using regression information, the NiS/Ni ratio was subdivided into three groups:

- low NiS/Ni ratio <0.25;
- middle NiS/Ni ratio >0.25 and <0.5; and
- high NiS/Ni ratio >0.5.



Source: Wardrop, 2009b

Figure 2.7-8 Histogram and Probability Plot of Ni% Assay Data



	REGOLITH	GRANITE	SERP	AMPH	MAFICMV	METASED	MAFICDYKE	
Number of data	8	2838	8933	81	3	171	183	Number of data
Mean	0.143	0.096	0.475	0.027	0.015	0.121	0.062	Mean
Maximum	0.52	2.6	4.74	0.65	0.029	3.28	0.91	Maximum
Upper quartile	0.245	0.076	0.64	0.023	0.025	0.045	0.045	Upper quartile
Median	0.065	0.013	0.29	0.007	0.015	0.012	0.019	Median
Lower quartile	0.004	0.001	0.187	0.0	0.004	0.004	0.009	Lower quartile
Minimum	0.0	0.0	0.0	0.0	0.0	0.0	0.0	Minimum
Variance	0.0309	0.0522	0.2279	0.0059	0.0001	0.1709	0.0138	Variance
CV	1.226	2.375	1.005	2.86	0.807	3.427	1.89	CV
Skewness	1.1	4.554	1.768	6.812	-0.042	5.909	3.651	Skewness

Source: Wardrop, 2009b

Notes:
- SERP = Serpentinite
- AMPH = Amphibolite
- MAFICMV = Mafic Metavolcanic
- METASED = Metasediment
- MAFICDYKE = Mafic Dyke

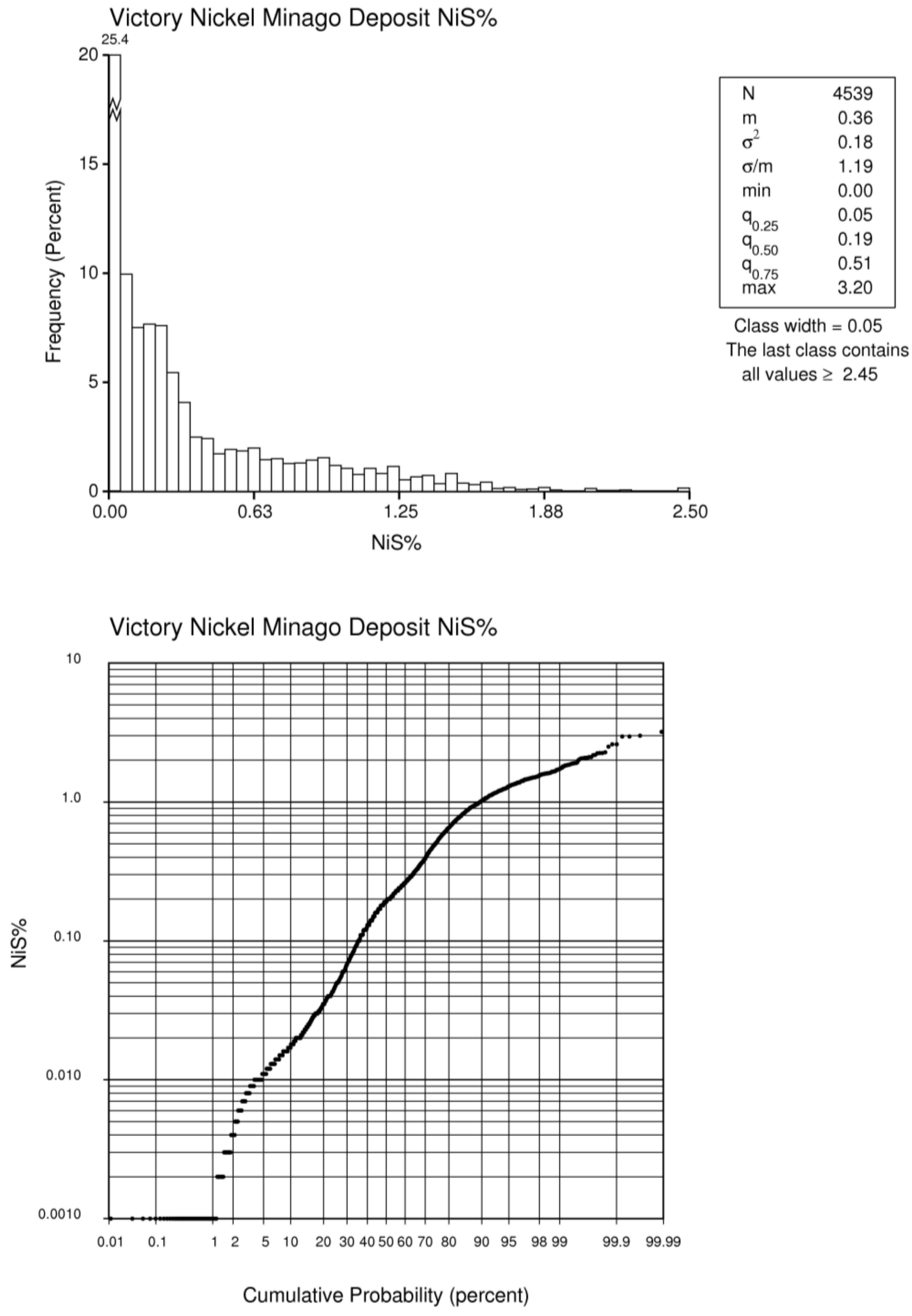
Figure 2.7-9 Boxplot of Ni Assay Data by Rock Type

The results from the ratio analysis among other information were used to identify spatially where the low versus high NiS/Ni ratio lies. Figure 2.7-12 displays a vertical section where drillholes are plotted with their NiS/Ni ratio using the three groups. The low NiS/Ni ratio is at the top of the deposit. It is unclear what kind of geological controls govern this, however moving from west to east, the low NiS/Ni ratio can be traced (Wardrop, 2009b). On the west limb, the low NiS/Ni ratio is on the south side, whereas on the east limb the low ratio is in the middle (Wardrop, 2009b). Around the fold nose, the low NiS/Ni ratio is almost non-existent. This is probably due to the fact that higher Ni samples with high NiS content exist around the fold nose, therefore having a higher NiS/Ni ratio (Wardrop, 2009b).

Table 2.7-3 NiS% Database

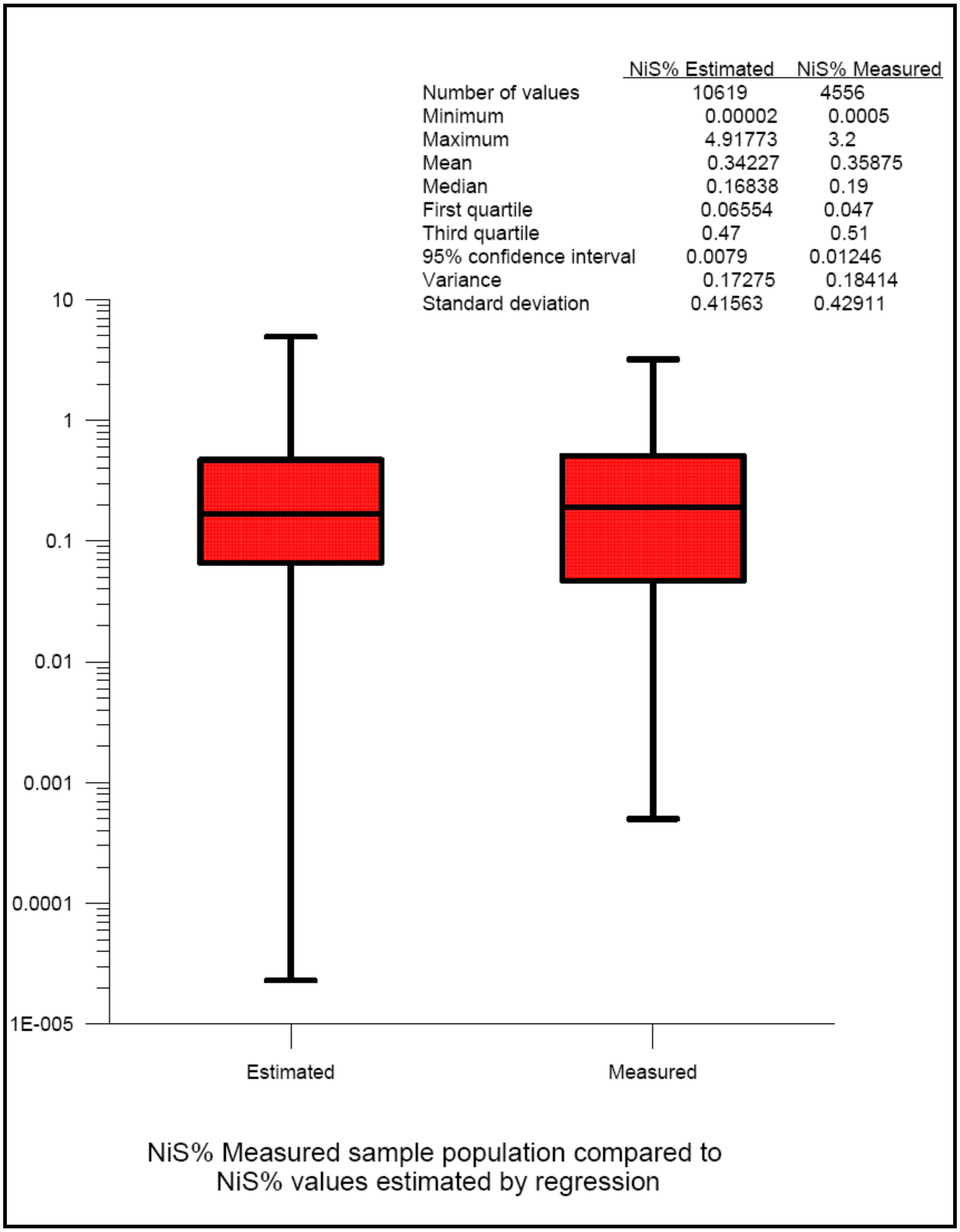
Company	Drillhole Number	Number of Assays
Amax Exploration	MXB-70-48	657
	MXB-70-54 TO MXB-70-58	
	MXB-71-88 TO MXB-71-99	
Thompson Core	B-16-89	273
	B-7-89	
	G-1-74	
	G-2-75	
	MXB-71-93	
Victory Nickel	N-07-01 TO N-07-04	2,368
	N-07-06 TO N-07-07	
	N-07-09 TO N-07-23	
	N-07-25 TO N-07-28	
	N-07-30 TO N-07-39	
	N-07-41 TO N-07-44	
Victory Nickel	V-08-01 TO V-08-10	1,259

Source: Wardrop, 2009b



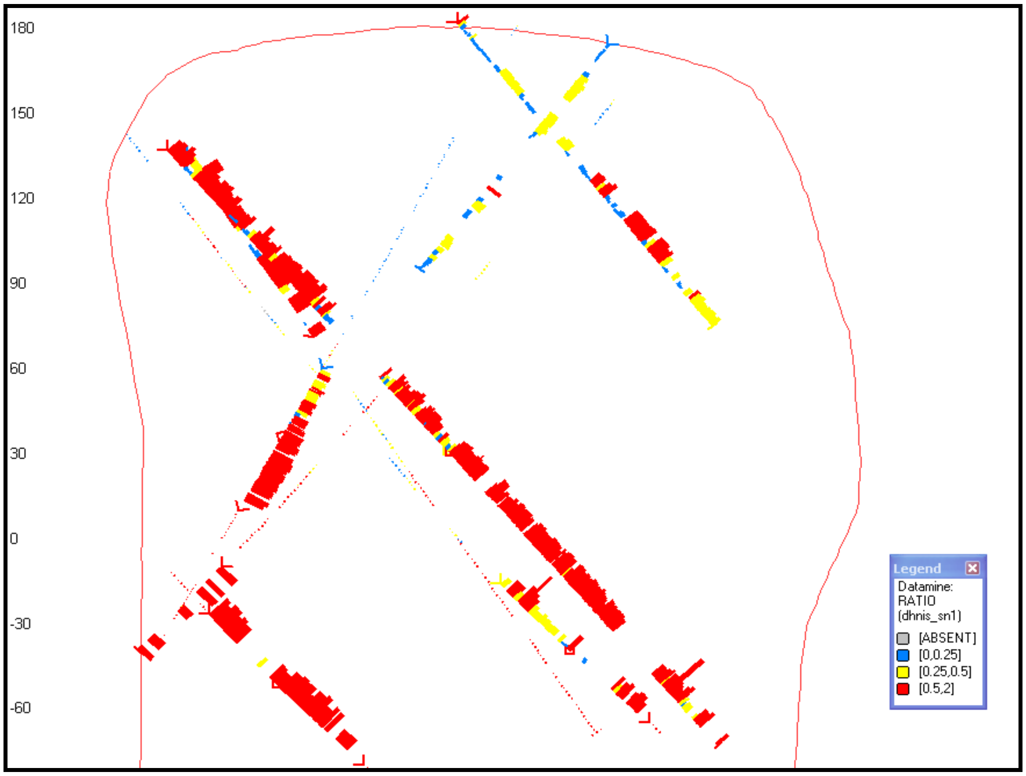
Source: Wardrop, 2009b

Figure 2.7-10 Histogram and Probability Plots of NiS% Assay Data



Source: Wardrop, 2009b

Figure 2.7-11 Boxplot of NiS% Assays by Rock Type



Source: Wardrop, 2009b

Figure 2.7-12 Vertical Section of the Wireframe and Drill holes with NiS/Ni Ratios

2.7.10.3 Frac Sand

Assay data used to evaluate the frac sand deposit was from test results performed by Loring, SRC, and TSL Laboratories Inc. (TSL). Comparison of the sand particle size distribution between the different samples submitted to Loring shows little variation across the area of the proposed pit. The size fraction data was combined into a coarse fraction (20/40) and fine fraction (40/140) for resource estimation purposes. Quality of the sand was shown to be affected by the testing method, so actual quality values will ultimately be determined by the recovery process (Wardrop, 2009b).

2.7.10.4 Solids

Mr. Chornoby standardized all the lithology codes from different drill campaigns. Lithological data from Amax, Granges, Blackhawk, Nuinsco, and Victory Nickel 2007 drilling now uses a common standardized codification system for all lithological units. Table 2.7-4 summarizes lithologies and their codes used in the model.

Table 2.7-4 Lithology Units and Rock Codes

Lithocode	Rock Code	
Overburden	OVB	10
Dolomite	DOL	20
Sandstone	SS	30
Serpentinite	SPT	40
Granite	GT	50
Amphibolite	AMP	60
Mafic Dyke	MD	70
Metasediment	MSD	80
Mafic Metavolcanic	MMV	90
Lost Core	LC	100
Regolith	R	110
Dunite/Peridotite/Pyroxenite	DPP	130

Source: Wardrop, 2009b

2.7.10.5 Bulk Density

During the 2006-07 and 2008 drill programs, TSL Laboratories (TSL) of Saskatoon conducted bulk density determinations as instructed by Nuinsco/Victory Nickel personnel. Table 2.7-5 is a compilation of the 2,050 samples that were used for density test work, out of which 779 samples were serpentinite (Wardrop, 2009b).

Table 2.7-5 Summary of 2007 Density Data

Lithology code	Number of Samples	Mean	Minimum	Maximum
Amphibolite (60)	493	3.01	2.41	3.57
Dolomite (20)	9	2.64	2.61	2.68
Granite (50)	361	2.67	2.32	3.26
Mafic Metavolcanic (90)	44	2.89	2.60	3.11
Metasediment (80)	57	2.86	2.63	3.43
Serpentinite (40)	779	2.58	2.16	3.86

Source: Wardrop, 2009b

Historically, the density for serpentinite, reported by Danley in a 1972 report, indicated a mean of 2.40 with a minimum value of 2.18 and a maximum value of 2.48 grams per cubic centimetre (g/cm^3) (Wardrop, 2009b). In the tests conducted by TSL, serpentenite had a minimum, mean, and maximum density of 2.16 g/cm^3, and 2.58 g/cm^3, 3.86 g/cm^3, respectively.

2.7.11 Geological Interpretation

The geological interpretation of the Minago deposit was conducted by Mr. Jim Chornoby, P.Geo. and Shahé Naccashian, P.Geo. The model was subsequently updated by Cliff Duke, P. Eng. Gemcom 6.1.3 software was used to build the surfaces and the solids.

The mineralization at Minago was considered as west and east domains based on the shape of the deposit. The deposit appears to consist of two limbs of a folded structure, with the apparent fold nose roughly located at UTM 487,350 East (NAD 83). The west domain was coded as 4010 and the east domain was coded as 4030. Figure 2.7-13 shows the 3D solid by domains split at the fold nose.

Figure 2.7-14 displays the geological solid at Minago. Using all drilling information, the current geological interpretation was completed and the overall mineralization continuity was maintained (Wardrop, 2009b).

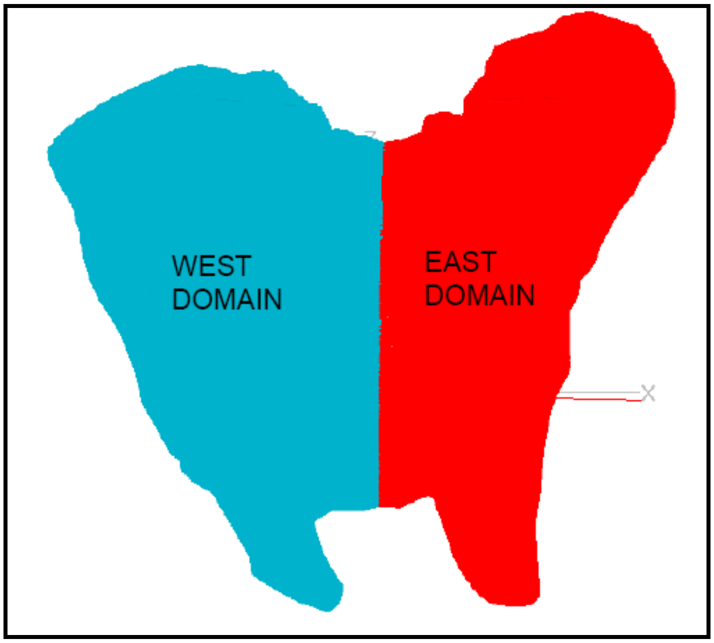
Figure 2.7-15 shows two solids: a yellow inside a large red solid. The yellow indicates areas with a low NiS/Ni ratio, whereas the red is the orebody wireframe. Transparent wireframes were plotted so that both can be visible (Wardrop, 2009b).

2.7.12 Conclusion – Resource Estimate and Geological Interpretations

The resource estimate and geological interpretations indicate that the upper portion of the Minago deposit (down to 400 m below surface) may be reasonably perceived as a large tonnage low grade nickel deposit (Wardrop, 2009b). A significant part of the lower portion of the deposit remains incompletely delineated as evidenced by the considerable tonnage of resource estimates in the Inferred category. Wardrop estimates that the Minago deposit contains a measured resource of 9.1 Mt, grading 0.47% NiS, above a cutoff grade of 0.2% NiS. In addition, the deposit contains 35 Mt of indicated resource at 0.42% NiS above a 0.2% NiS cutoff grade. An Inferred Resource of 12 Mt at 0.44% NiS above a 0.2% NiS cutoff has also been estimated (Wardrop, 2009b).

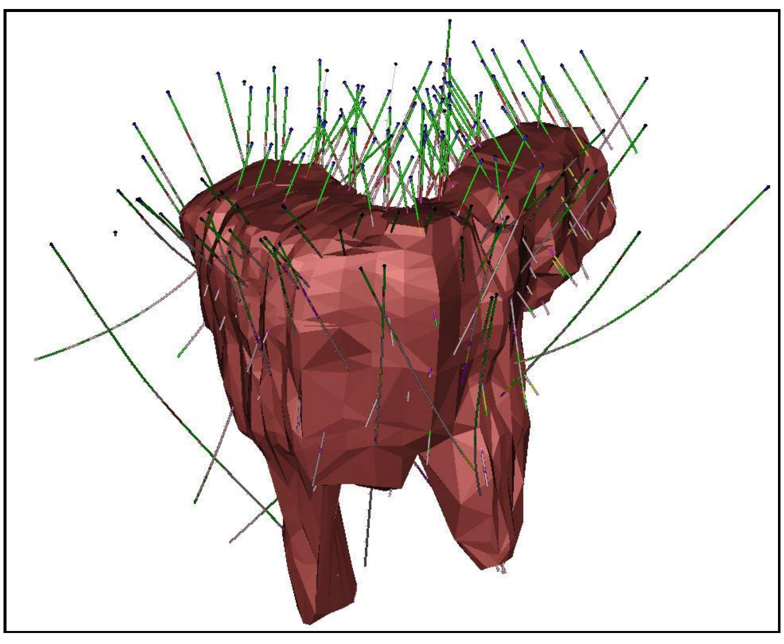
NiS% represents the recoverable part of the nickel. Unfortunately, all the samples were not analyzed for NiS, and grades for the missing assay values had to be interpolated using regression analysis from total nickel assays. For a significant part of the database, Wardrop's estimate of the missing NiS% assays was based on the relationship of NiS to Ni. The population distribution of the calculated NiS values was similar to that of the assayed NiS values (Wardrop, 2009b).

Rock density values appear to be more dependent on rock type than on grade (Wardrop, 2009b).



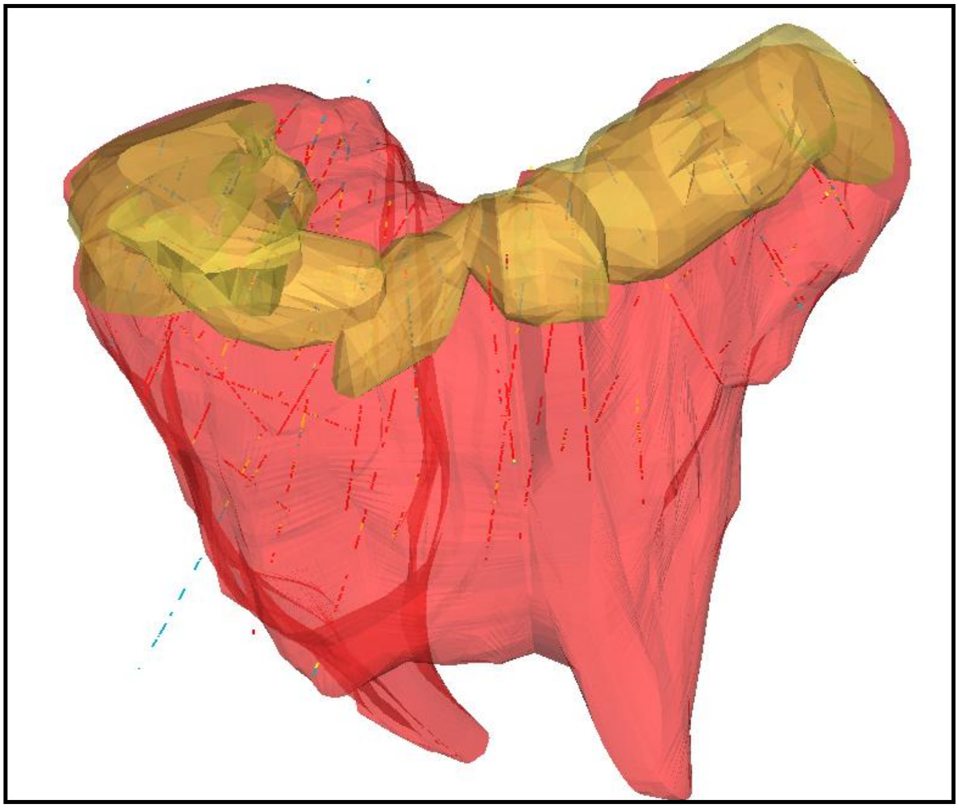
Source: Wardrop, 2009b

Figure 2.7-13 Three Dimensional Solid by Domain



Source: Wardrop, 2009b

Figure 2.7-14 Minago Deposit 3D Wireframe with Drill Holes



Source: Wardrop, 2009b

Figure 2.7-15 Low NiS/Ni Ratio Solid Inside Orebody Wireframe

Widely spaced single tier diamond drilling north of the Minago deposit and in the south block has intersected nickel mineralization similar to that found in the Minago deposit indicating exploration potential. This additional resource is not included in this study (Wardrop, 2009b).

A pit shell proposed to recover the nickel at Minago has been used to laterally constrain the frac sand resource. Wardrop estimates that the proposed pit shell contains 15 Mt of sand. Of this, 13% is expected to report to the 20/40 size fraction, and 71% is expected to report to the 40/140 size fraction (Wardrop, 2009b).

Based on the depositional model of the Winnipeg Formation sandstone, Wardrop expects the frac sand quality of the sandstone within the confines of the proposed pit shell to be fairly uniform. The size distribution of the sand particles that have been sampled is consistent across the area of the proposed pit shell. Initial testing of a sample composite indicates that a viable frac sand product can be produced from the resource. Drill hole intersections and density measurements have been sufficient to establish the tonnage of the deposit with a reasonable degree of accuracy (Wardrop, 2009b).

2.7.13 Mineralogy

Typical sulphide mineralization of the Minago nickel deposit consist of very fine grained (<0.5 to 4 mm), disseminated (2 to 7%) and occasionally net-textured pentlandite ($(Fe,Ni)_9S_8$) with lesser violarite (Fe,Ni_2S_4), millerite (NiS) and heazlewoodite (Ni_3S_2) (URS, 2009i). Minor amounts of pyrite (FeS_2), pyrrhotite ($Fe_{1-x}S$) and chalcopyrite ($CuFeS_2$) are also present (URS, 2009i).

The dominate minerals in the deposit are serpentinite (serpentine mineral) and peridotite (olivine mineral) and both are silicates. The serpentinite ore is closer to the surface and thus more oxidized (Wardrop, 2006).

Typically sulphides on the Property are very fine-grained varying in size from < 0.5 to 4 mm (generally 1 to 2 mm) and range in volume from 2 to 15% (generally 2 to 7%). Sulphides are predominantly disseminated and occasionally net textured. The dominant sulphide species are nickel bearing pentlandite with lessor violarite and millerite. Minor amounts of pyrite, pyrrhotite and chalcopyrite are present (Wardrop, 2006).

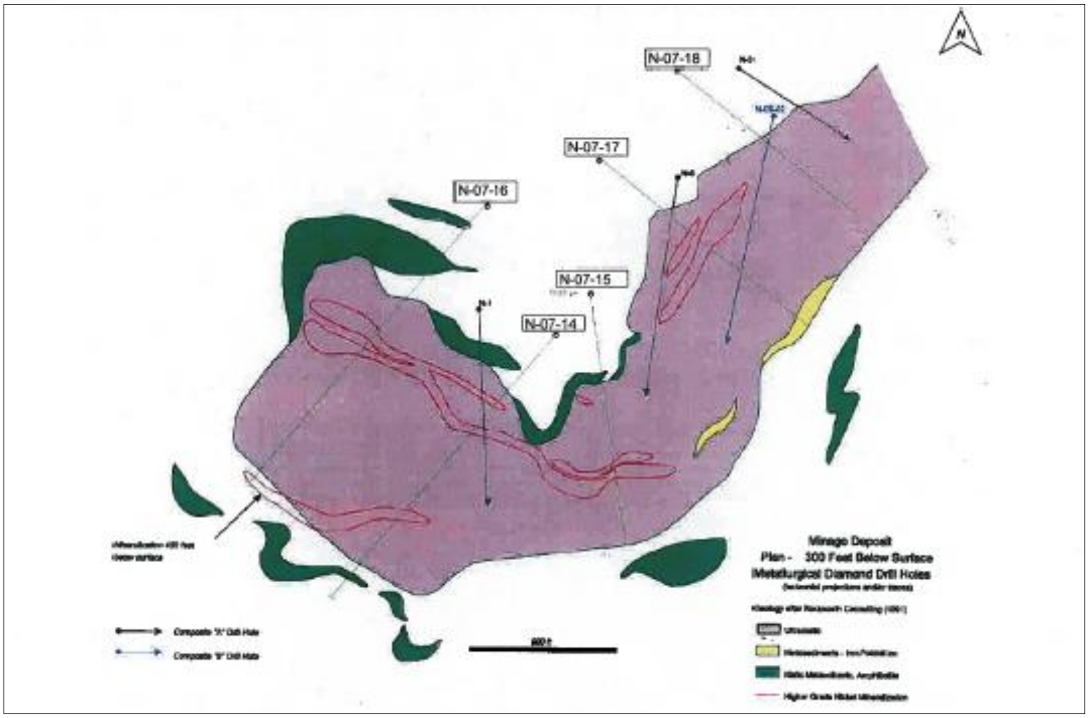
Due to the relatively high talc content in the ore, metallurgical test work focussed on processes that would result in a high nickel grade concentrate (>25% Ni), with a low talc content (<10% MgO), at the highest possible nickel recovery. Concentrates containing high talc content can be detrimental to the smelting operation and potentially unmarketable (Wardrop, 2006). Contract penalties escalate when talc values in the nickel concentrate rise above levels acceptable to smelters.

2.7.14 Metallurgical Testing

The metallurgical test program at Minago established that the Minago deposit contains a significant amount of nickel in the form of nickel silicates, which are not recoverable by froth flotation (Wardrop, 2008a). Thus, the deposit defined using the total nickel assay is not reliable in terms of determination of mineable sections, pit design, and economic analysis of the mining and mill operation. Therefore, Minago’s head grade-recovery curve was based on the grade of nickel sulphide (Wardrop, 2008a).

2.7.14.1 Drill Holes used for Metallurgical Testing

Five dedicated drill holes, identified as N-07-14, N-07-15, N-07-16, N-07-17 and N-07-18, were selected by geologists from Wardrop and Victory Nickel to generate samples for metallurgical testing for the bankable Feasibility Study on the Minago project. The five holes were located roughly even along the strike of the deposit (as shown in Figure 2.7-16) in order to represent ores from the whole open pit (Wardrop, 2008a).



Source: adapted from Wardrop, 2008

Figure 2.7-16 Location of the Five Metallurgical Drill Holes

All 1,117 drill core intervals obtained from the five metallurgical drill holes were split at SGS Lakefield Research (SGS or Lakefield) and sent for total nickel assay. The total weight of these drill cores was 4,174.2 kilograms (kg). The intervals with a total nickel assay of higher than 0.2% nickel were later assayed for sulphidic nickel (Wardrop, 2008a).

The uncrushed core splits and coarsely crushed core samples were stored in a cold room at the Lakefield site. The metallurgical testing samples were formed from these core samples based on a sample recipe designed to suite the objectives of the tests (Wardrop, 2008a).

Two Master Composite Samples for the Open Pit were prepared for the metallurgical testing. The Open Pit Master Composite No. 1 was based on the total nickel assays of the intervals of the five metallurgical holes, as the nickel sulphide assay of the intervals was not available, and the significant variation of the ratio of sulphidic nickel to total nickel was not understood at the time. The Open Pit Master Composite No. 2 was a composite sample to represent the ore from the open pit operation based on nickel sulphide. The Open Pit Master Composite No. 2 sample was generated from the metallurgical drill core intervals, based on volume of influence with consideration of the average nickel sulphide grade of the ore contained in the open pit. The Open Pit Master Composite No. 2 was used to develop the design criteria for the Minago project. The Open Pit Master Composite No. 2, which was designed to represent the overall open pit ore, contained 0.53% total nickel and 0.36% sulphidic nickel (Wardrop, 2008a).

The sulphidic nickel assays of the drill core intervals of the five dedicated holes were provided by SGS. The sulphidic nickel assays of the intervals of other 2007 drill holes were provided by SGS and ACME Labs in Vancouver (Wardrop, 2008a).

2.7.14.2 Summary of Results of the Metallurgical Test Program

The complete metallurgical program at Minago is presented and discussed in Wardrop (2008a). Major conclusions from the metallurgical testing program are as follows (adpated from Wardrop, 2008a):

1. Geological Model: Part of the nickel (Ni) in the Minago deposit is in the form of nickel silicates; the ratio of nickel content in silicate to the nickel content in sulphides varies over the deposit. It is therefore impossible to assess the nickel recovery and project economics based on total nickel assays.

2. A locked cycle test on the Open Pit Master Composite No. 2 achieved a nickel concentrate containing 22.23% nickel and 10.43% magnesium oxide (MgO) with a nickel sulphide equivalent recovery of 77.2%. Multiplying this recovery by the average sulphidic nickel to total nickel ratio of 75.4% yields an average total nickel recovery of 58.2%.

3. The grindability testing samples had a median SPI (SAG Power Index) of 27.4 minutes; a median RWI (Rod Mill Work Index) of 9.6 kilowatt hours per tonne (kWh/t) and a median BWI (Ball Mill Work Index) of 14.9 kWh/t. These data indicate that the grinding hardness of the samples is intermediate on average.

4. The optimum grind size for the Minago sample was determined to be at P80 = 68 micrometres (µm).

5. Assays of the sulphidic nickel indicated that there is a significant portion of the nickel sulphide lost to the flotation tails. Mineralogy work (optical and QEMSCAN) indicated that the nickel sulphides lost to flotation tails were fine particles liberated or attached to silicates.

6. Three samples from Hole N-O7-14 and three samples from Hole N-O7-17 (at total nickel assay of ~0.3%, ~0.4% and ~0.5%) were selected for flotation tests to investigate the relationship between the nickel head grade and rougher tail nickel grade. The results indicated that nickel rougher recovery is lower for lower head grades, especially for samples with low nickel sulphide content. Results from these tests further confirmed that the nickel recovery of mining blocks has to be predicted from its sulphidic nickel grade.

7. Control of the magnesium oxide content in the final concentrate will likely be a challenge for the flotation of Minago samples. A series of depressants/dispersants regimes were tested. Carboxmethyl Cellulose (CMC) or CMC in combination with Calgon proved to be most effective in controlling the magnesium oxide content in the final concentrate. The

overall magnesium oxide rejection achieved in the locked cycle test on the Open Pit Master Composite No. 2 was 99.61% (0.39% recovered to the final concentrate). It will be difficult to further reduce the amount of magnesium oxide in the concentrate without significant loss of nickel.

8. The Minago slurries are viscous and pulp density has to be kept low to improve the selectivity of flotation. The effect of dispersants, such as sodium silicate and sodium hexametaphosphate (SHMP), were tested and proved to be effective in improving the flotation selectivity for the Minago samples.

9. Flotation tests and sulphidic nickel assays finished by 2008 indicated that the mineralogy and the floatability of the ore in the different locations of the deposit were quite different.

10. During the test program, it was found that Hole N-O7-18 contained such a small concentration of nickel sulphide; and therefore, should not considered to be ore.

2.7.14.3 Key Results for the Open Pit Master Composite No. 2 Sample

The Open Pit Master Composite No. 2 sample is a composite sample to represent the ore from the open pit operation based on nickel sulphide. The Open Pit Master Composite No. 2 sample was generated from the metallurgical drill core intervals based on volume of influence with consideration of the average nickel sulphide grade of the ore contained in the open pit (Wardrop, 2008a).

2.7.14.3.1 Flotation Test Results for the Open Pit Master Composite No. 2

The main objective of the flotation test work was to develop the design criteria for the plant design of the Minago project based on a composite that represents the ores produced from the open pit based on the sulphidic nickel deposit block model. The test work consisted of cleaner tests to confirm and further optimize the flotation parameters and locked cycle tests to generate the mass and water balance of the flow sheet. The effect of recycled water was also tested.

The cleaner tests indicated the following:

- Addition of Potassium Amyl Xanthate (PAX) in the cleaning stage resulted in lower nickel and higher magnesium oxide assays in final cleaner concentrate.
- Carboxmethyl Cellulose (CMC) addition in rougher flotation did not have a remarkable impact on the flotation of Open Pit Master Composite No. 2.
- Addition of Calgon in grinding produced the best flotation results with the concentrate assaying 22.2% total nickel and 11.8% magnesium oxide at a nickel sulphide equivalent recovery of 68%.

- Addition of Dep. C in grinding did not show any positive effect on the flotation of the composite sample.
- Acid wash did not improve the selectivity of the flotation process.

2.7.14.3.2 Locked Cycle Results for the Open Pit Master Composite No. 2

One locked cycle test was run based on the flotation parameters of one of the cleaner tests (SMC-6; Wardrop, 2008a). Based on this test, it was inferred that a nickel concentrate containing 22.27% nickel and 10.43% magnesium oxide may be produced with a total nickel recovery of 52.28% and a nickel sulphide recovery of 77.23% (Wardrop, 2008a). Based on this test, 126 tonnes of nickel concentrate will be produced at a mill throughput of 10,000 t/d and mill feed grade of 0.364% sulphidic nickel (8 pounds (lbs) sulphidic nickel per tonne of feed) (Wardrop, 2008a).

A second locked cycle test was completed to assess the effect of recycled water on the flotation behavior of the Open Pit Master Composite No. 2. Testing indicated that recycled water did not have a significant effect on the flotation behavior of Open Pit Composite No. 2 (Wardrop, 2008a).

2.7.14.3.3 Settling Tests of Flotation Tails

Based on the results of flocculant screening, five bench scale tests were conducted to evaluate the effect of flocculant dosage and initial pulp density. It was found that at an initial pulp density of 16.38% solids, 1.732 square metres (m^2) per tonnes per day of thickener settling area is required to achieve a pulp density of 40.7% solids for the thickener underflow. Thus, a 150 m diameter conventional tailings thickener is required for a mill with a throughput of 10,000 t/d (Wardrop, 2008.

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.

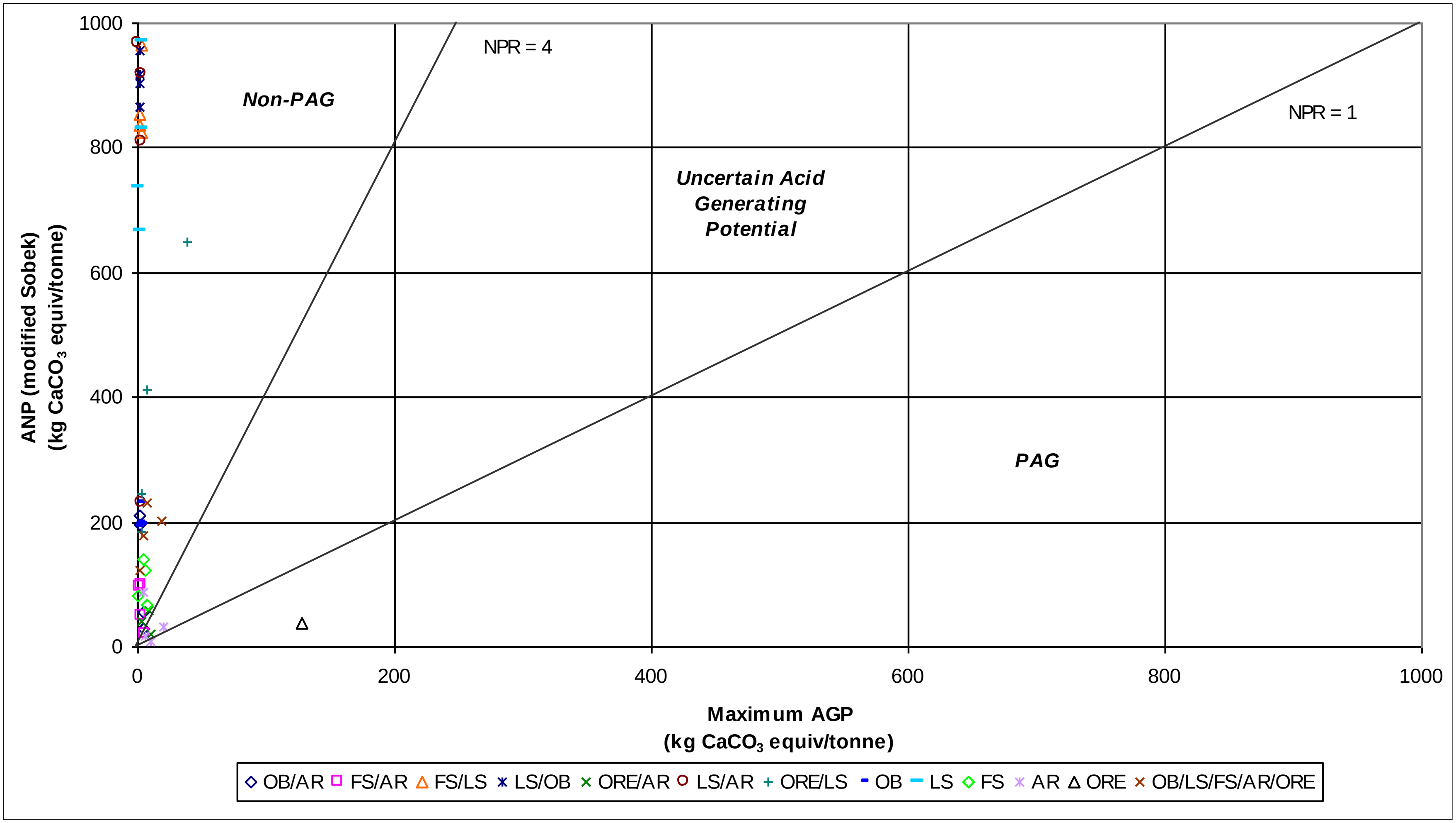
NPR = NP / MPA

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

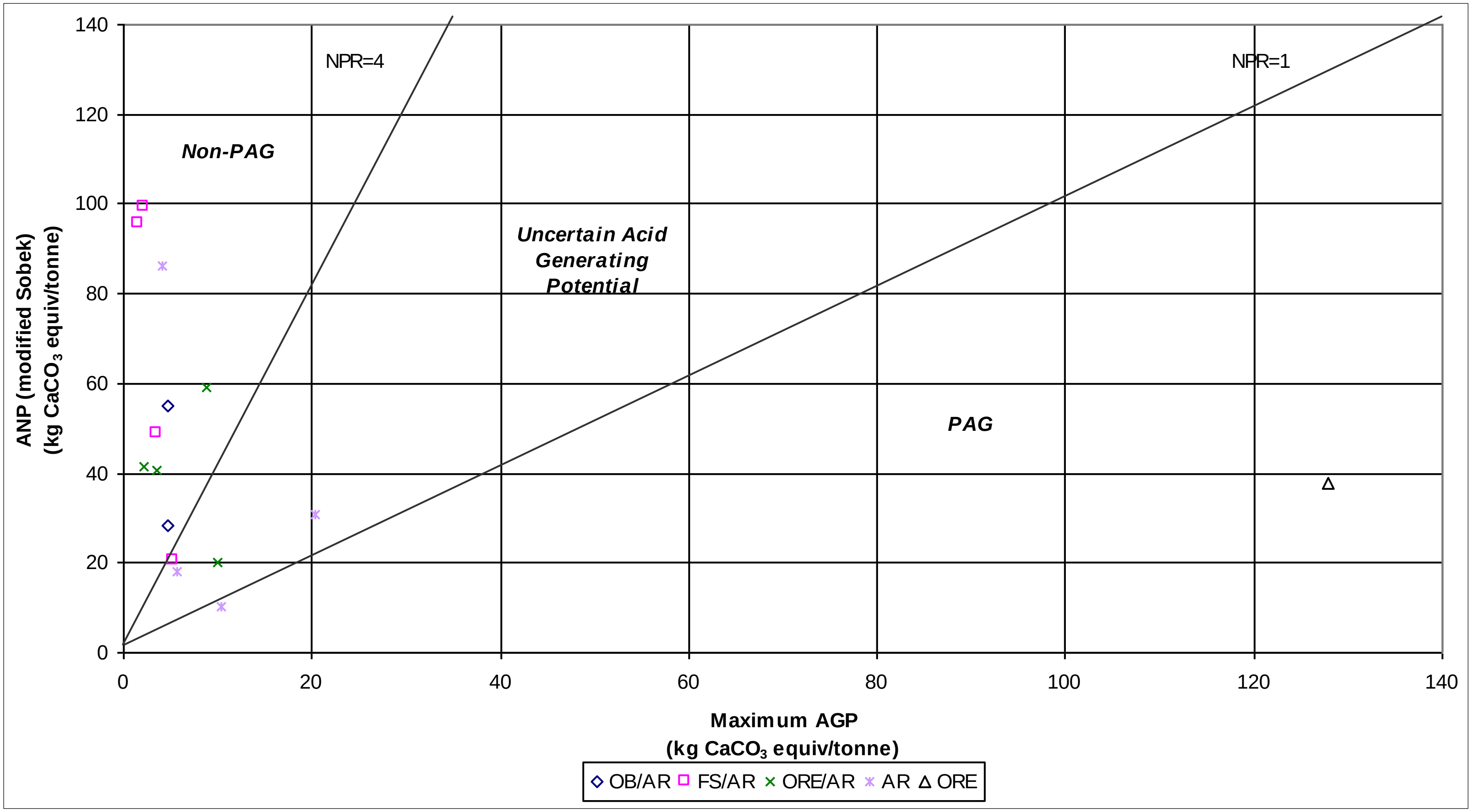
Rock Types: 1 = glacial lacustrine clay, 2 = dolomite, 3 = sandstone, 4 = serpentinite, 5 = granite

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.

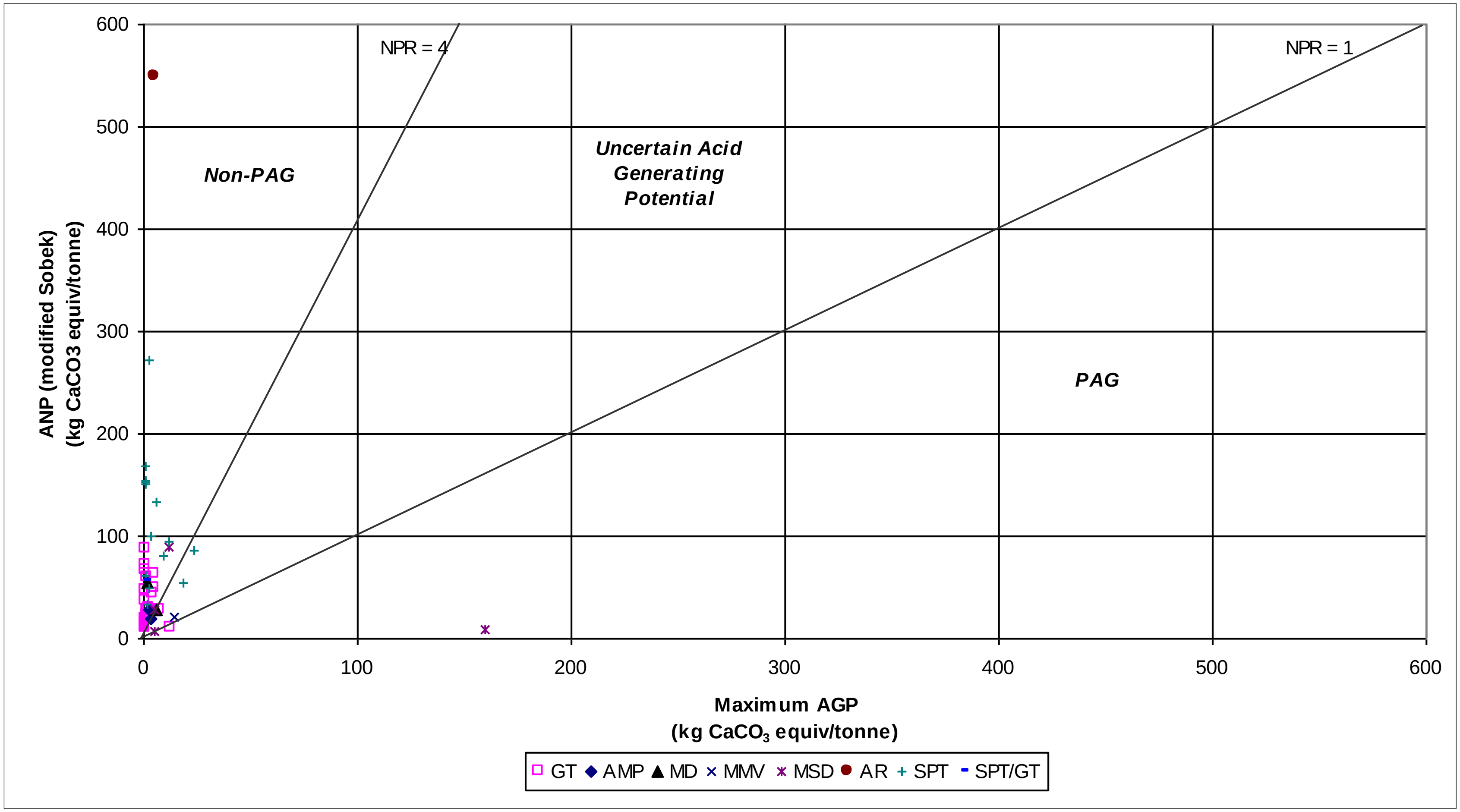
ANP = Modified Sobek Bulk Neutralization Potential in kilograms $CaCO_3$ equivalent per tonne of material.

NPR = ANP / AGP

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

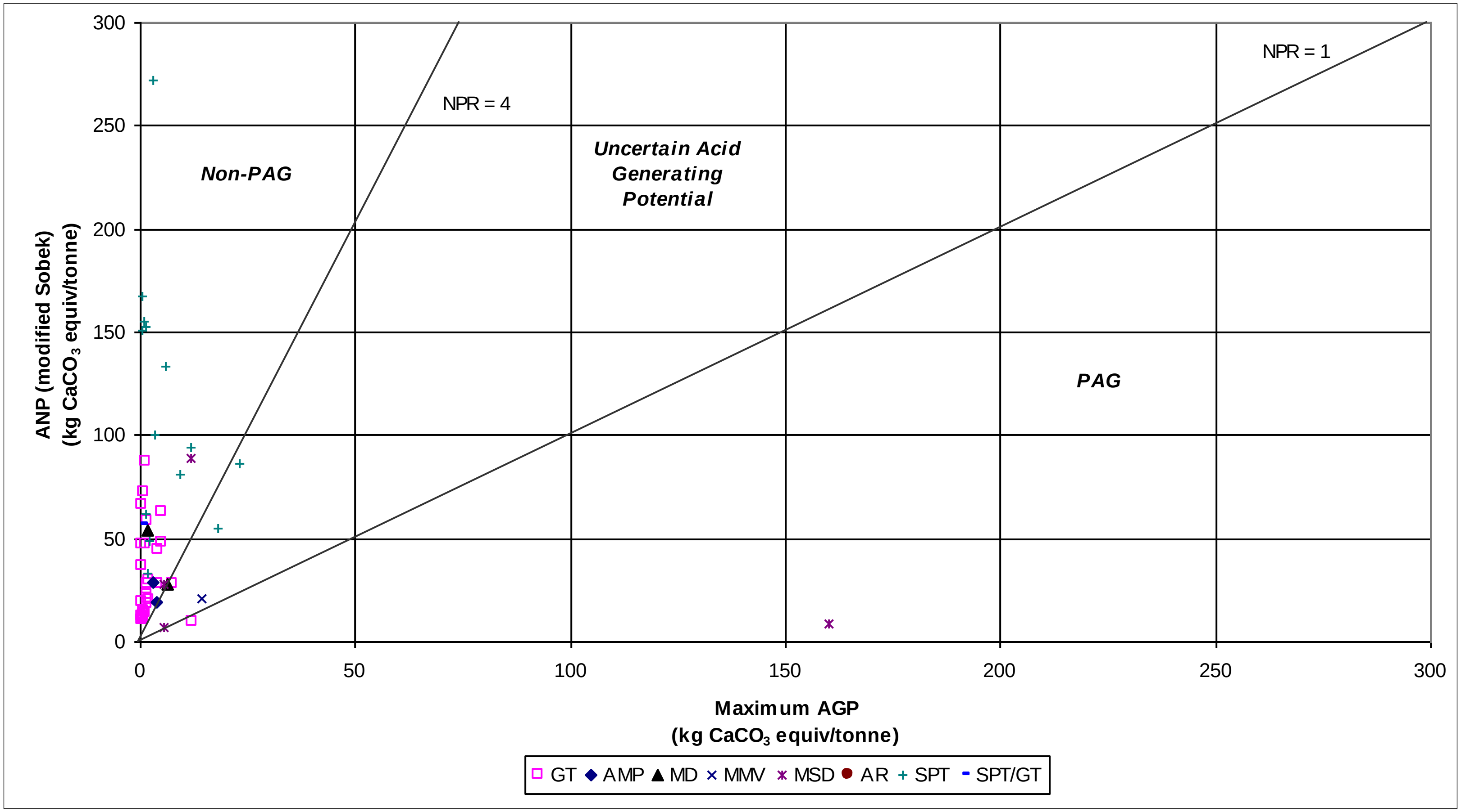
If the concentation was below the dectecion limit, half the detection limit was used to calculate the average.

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.

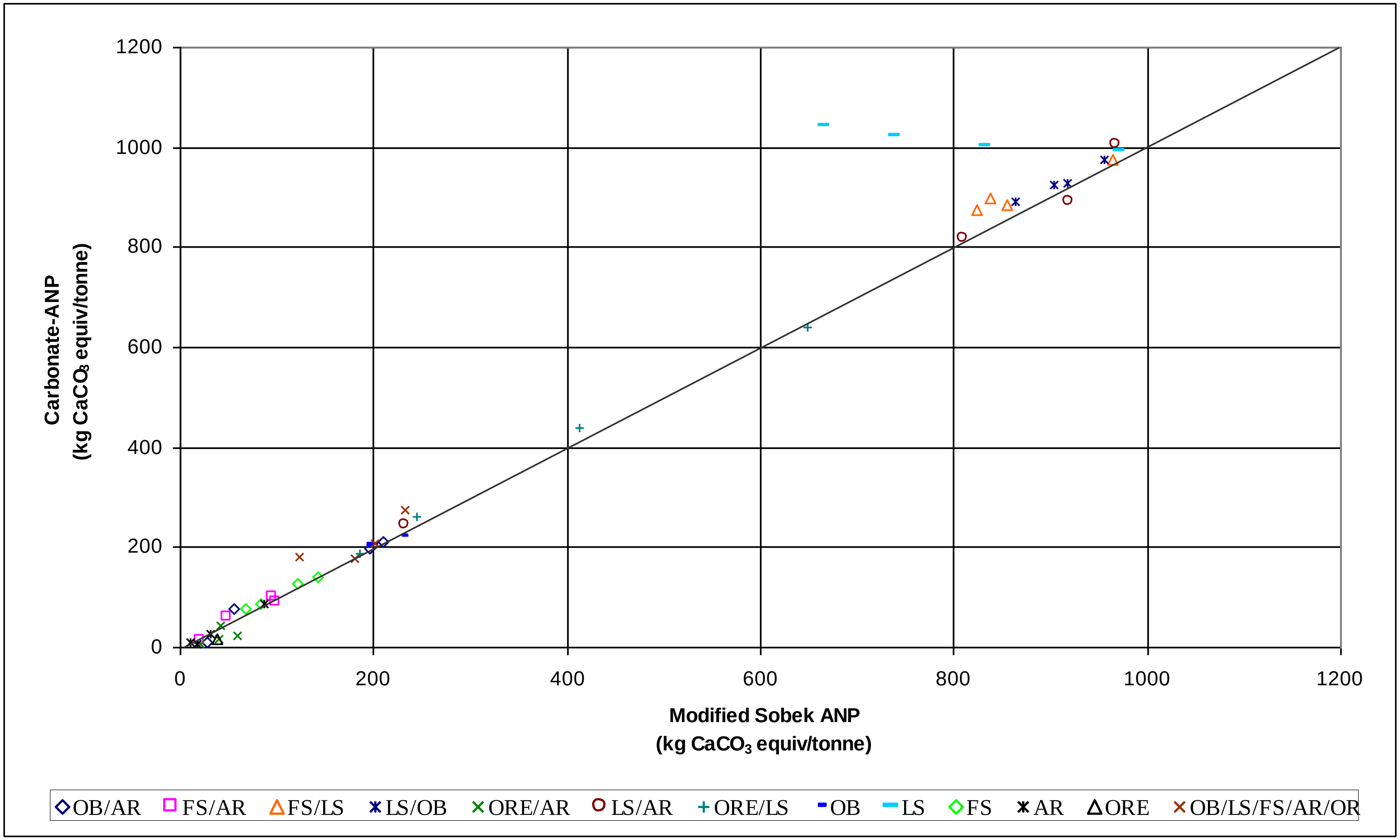
ANP = Modified Sobek Bulk Neutralization Potential in kilograms $CaCO_3$ equivalent per tonne of material.

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).

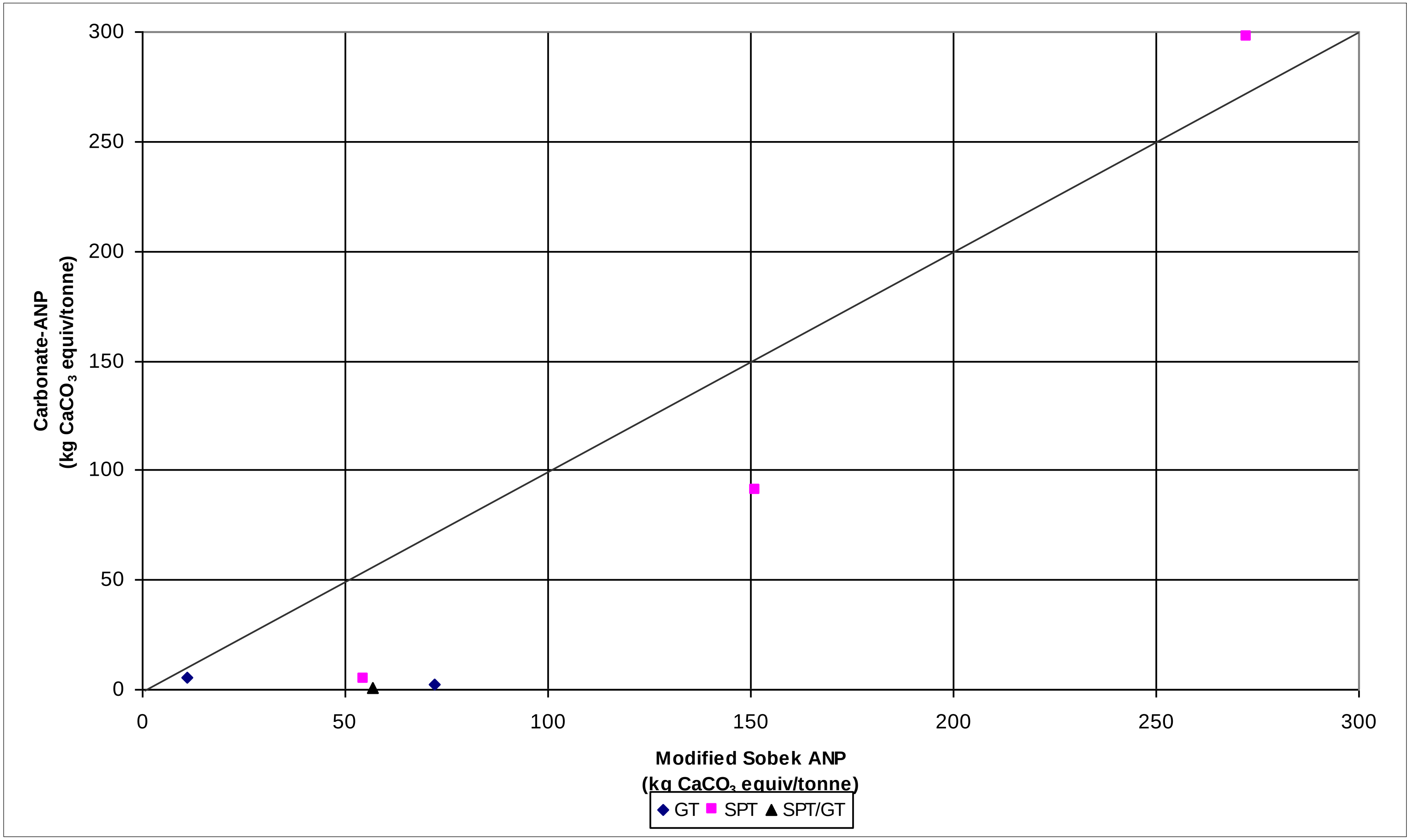
Rock Types: GT=granite, SPT=serpentinite, MD=mafic dike, MMV=mafic metavolcanic, R=regolith, AMP=amphibolite, MSD=metasediment

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)

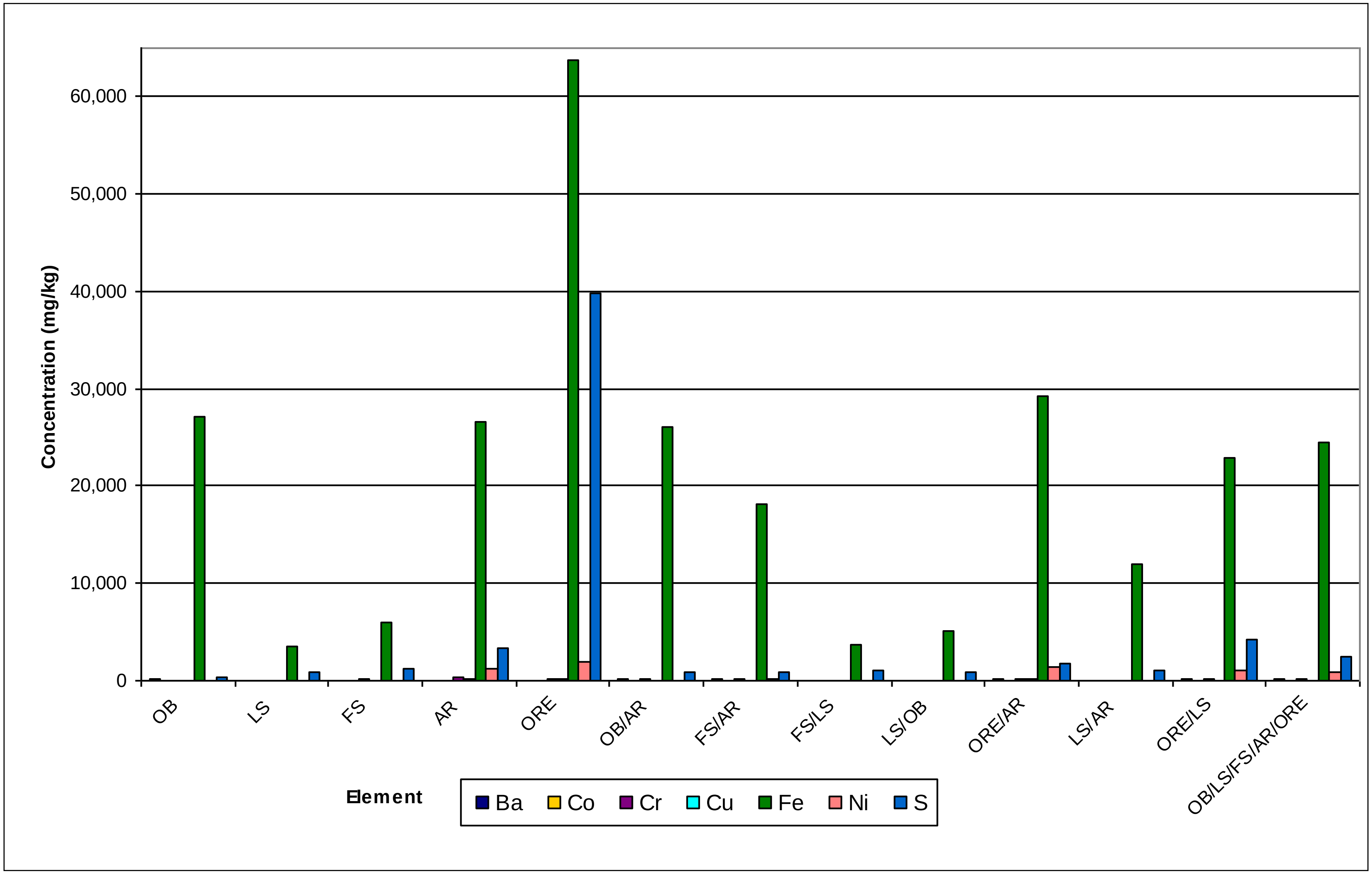


Figure 2.8-7 Phase I Static Test Results - Elemental Concentrations in Major Lithogies

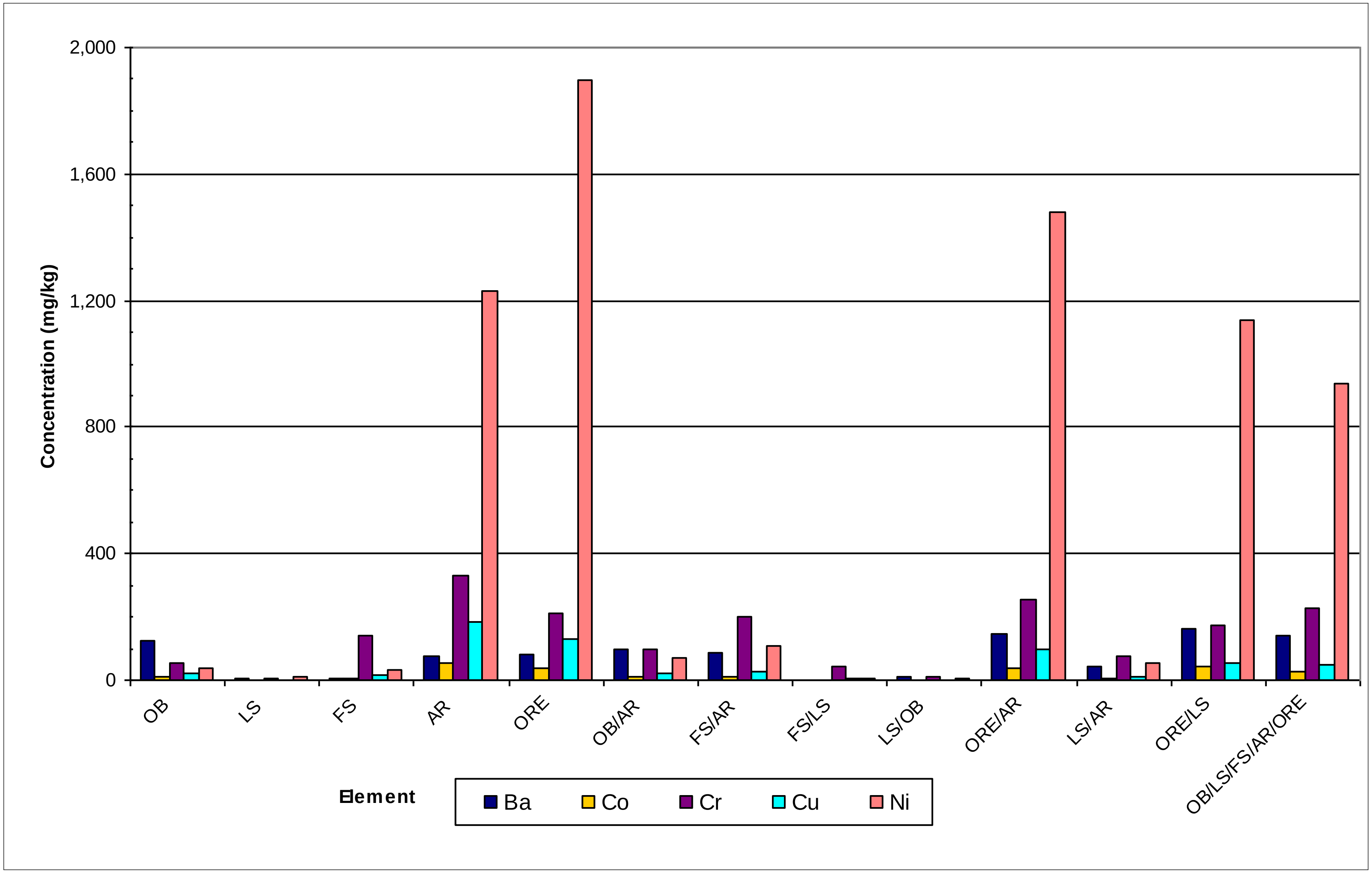


Figure 2.8-8 Phase I Static Test Results – Elemental Concentrations in Major Lithogies (Zoomed in)

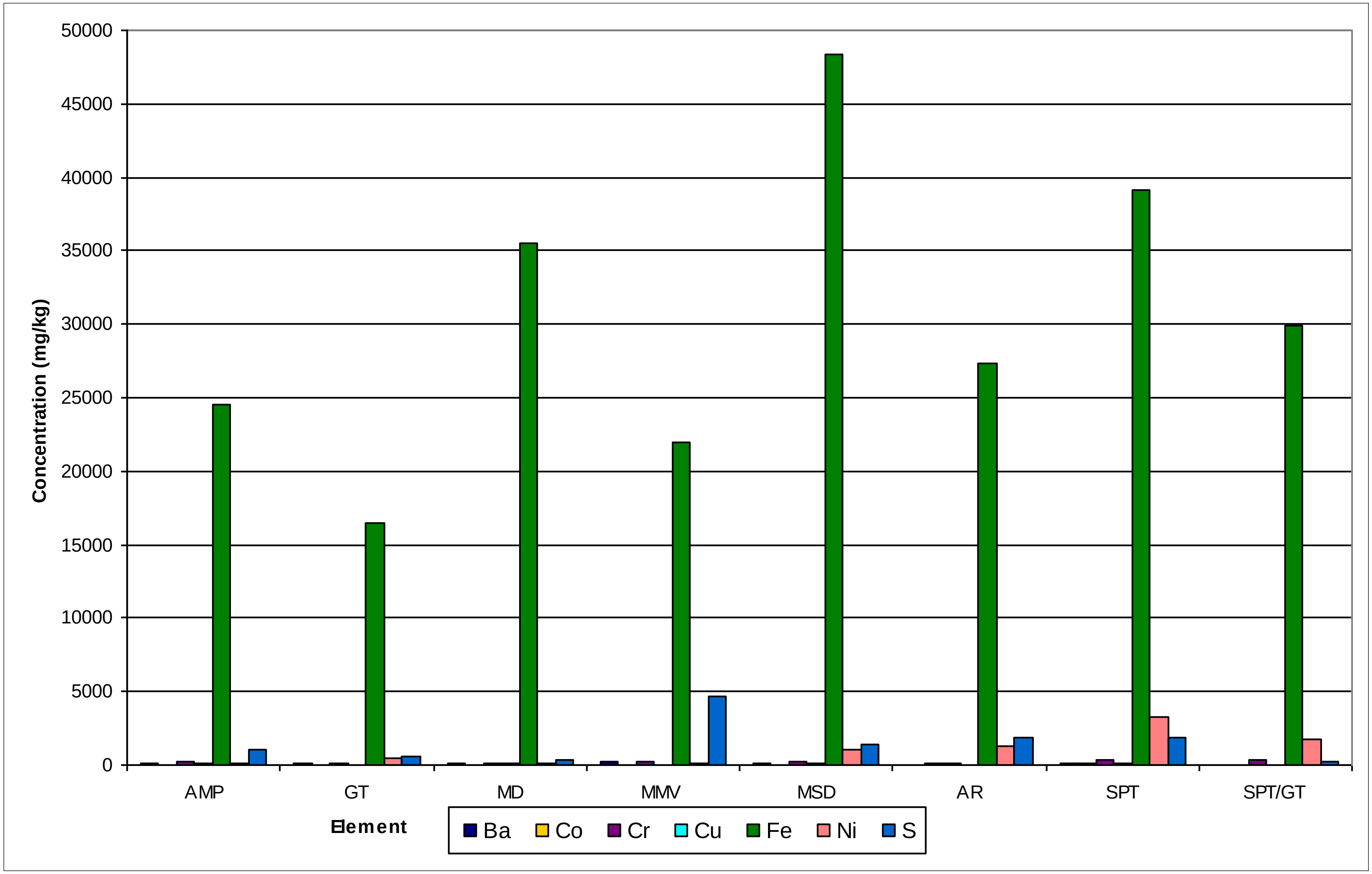


Figure 2.8-9 Phase II Static Test Results – Close-up of Elemental Concentrations in Major Lithogies

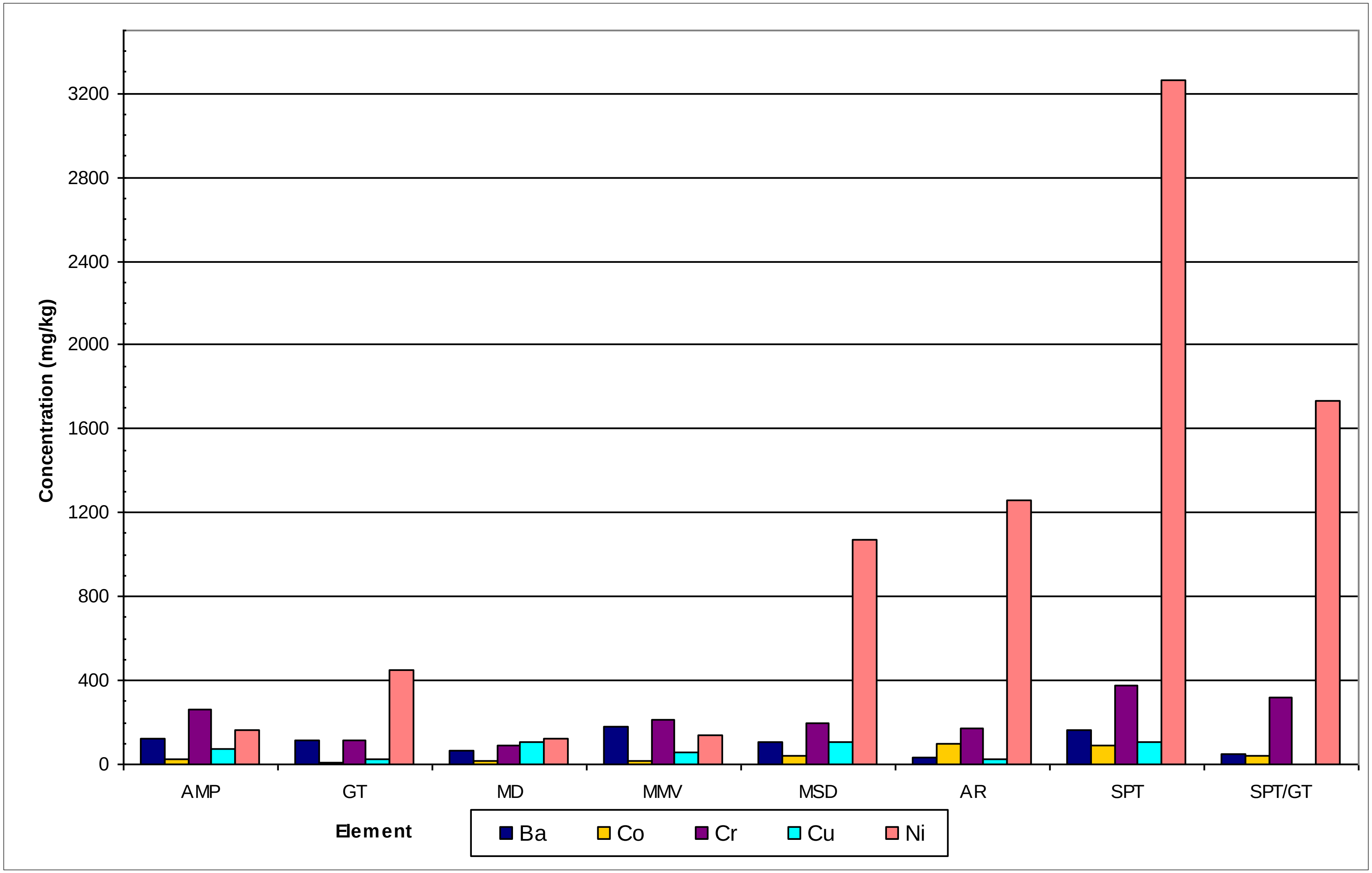


Figure 2.8-10 Phase II Static Test Results – Elemental Concentrations in Major Lithogies (Zoomed in)

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.

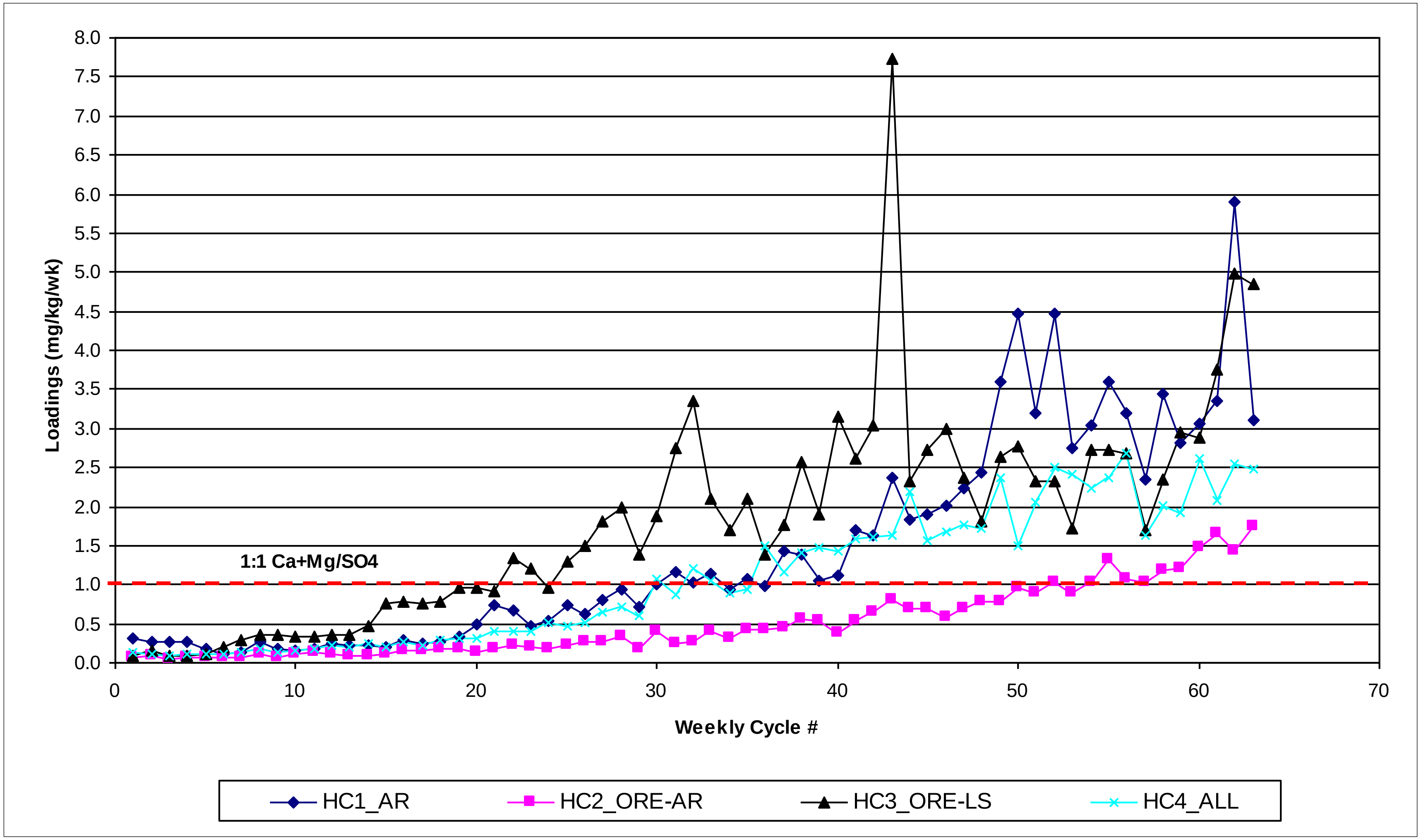


Figure 2.8-11 Ca+Mg/SO_4 Ratios for Minago Phase I Waste Rock Kinetic Tests

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	Aluminum	Antimony	Arsenic	Cadmium,	Chromium	Copper	Iron	Lead	Molybdenum	Nickel	Selenium	Zinc
		mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk
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	Aluminum	Antimony	Arsenic	Cadmium,	Chromium	Copper	Iron	Lead	Molybdenum	Nickel	Selenium	Zinc
		mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk	mg/kg/wk
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

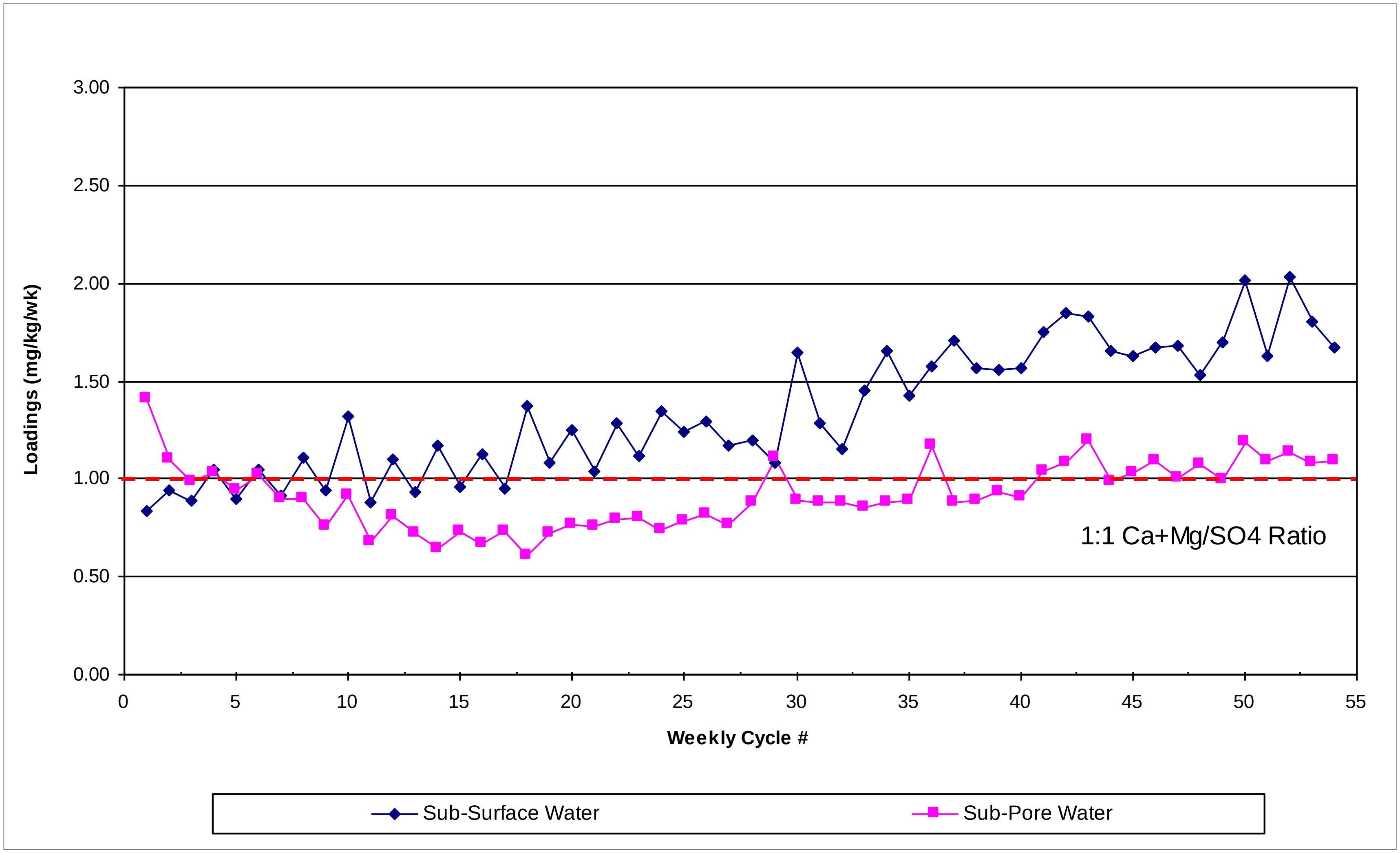
Carbonate molar ratios (the molar ratio of calcium and magnesium to sulphate in the leachates; $(Ca+Mg)/SO_4$) for the subaqueous column test SAC-1 are shown in Figure 2.8-12. This unitless ratio provides an estimate of the proportion of carbonate material that is released (dissolved) in response to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

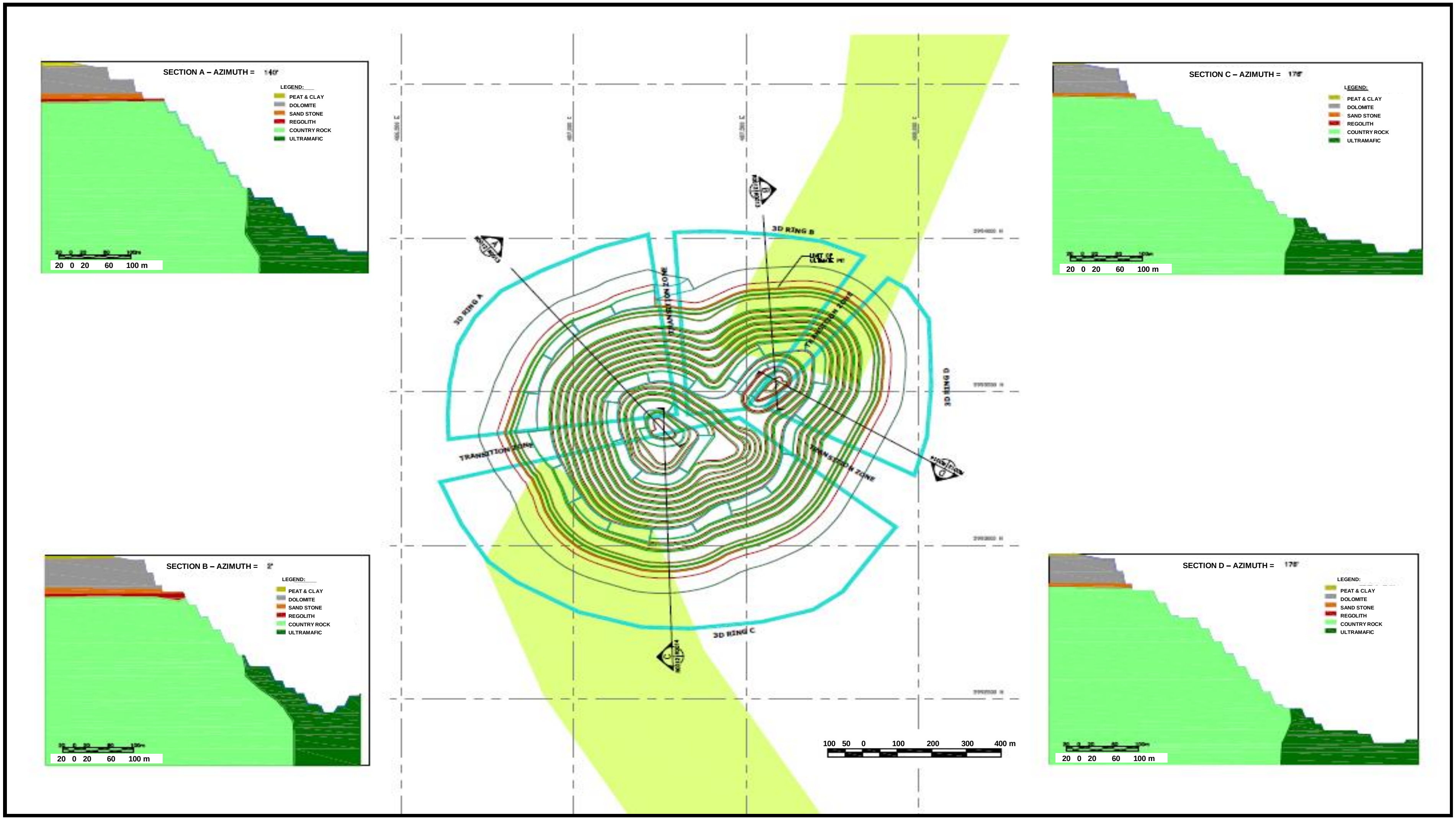
- Condition 1: The pit is dewatered and dry.
- Condition 2: The pit is dewatered, but the sandstone remains saturated.
- Condition 3: The pit has a perched water table above the basement rocks.
- Condition 4: The pit is completely saturated through the basement rocks.

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

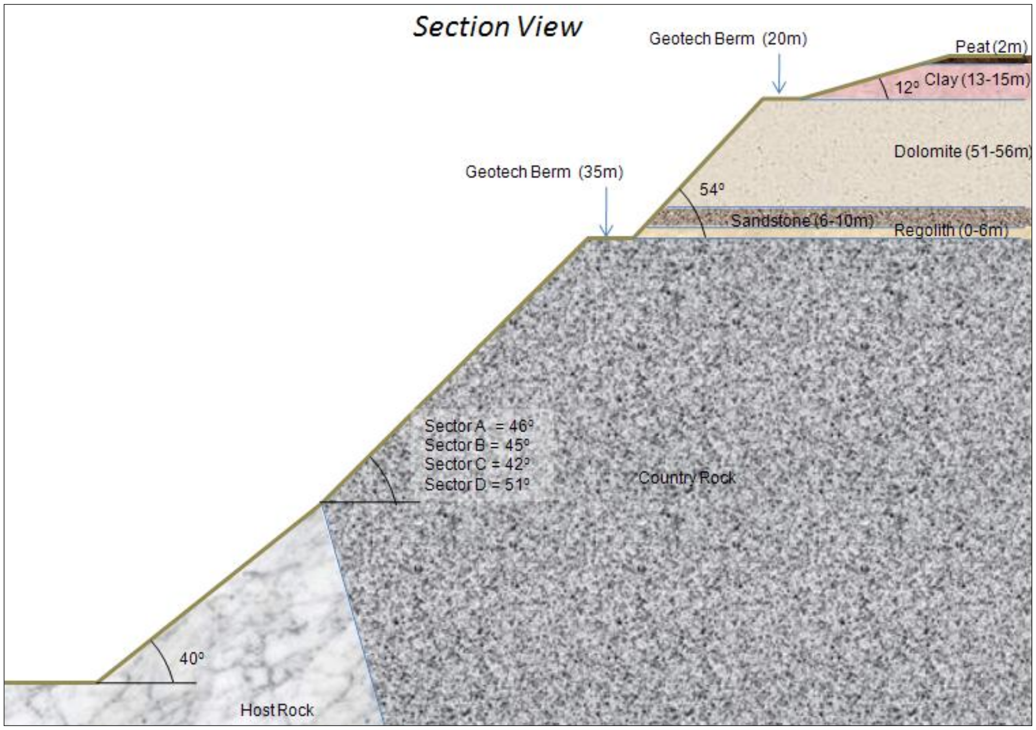
Design Sector	Geotechnical Domain	Interamp Angle (°)	Bench Face Angle (°)	Bench Height (m)	Catch Bench Width (m)
A	I, II	8.9	12.0	12	20
A	III	54.0	80.0	24	10
A	IV,V	12.9	80.0	12	35
A	VI	40.0	70.0	24	12
A	VII	46.0	70.0	24	12
B	I, II	8.9	12.0	12	20
B	III	54.0	80.0	24	10
B	IV,V	12.9	80.0	12	35
B	VI	40.0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

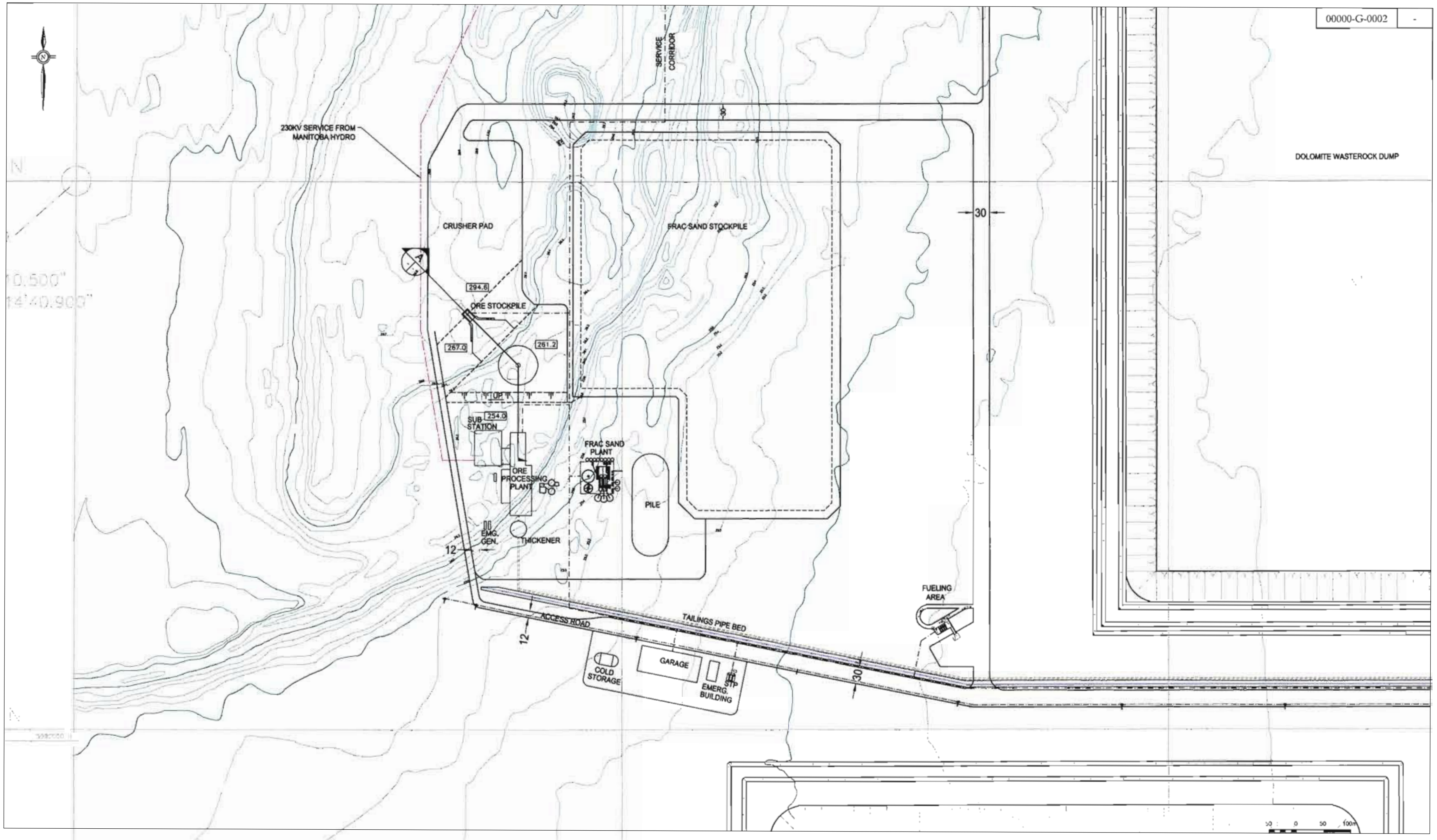
In Wardrop's work, the final pit was sub-divided into four main design sectors based on similar rock types and geotechnical data. The block dimensions for the Lerchs-Grossman algorithm were determined for each sector of the geotechnical domains. A 3D geological block model and other required economic 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).

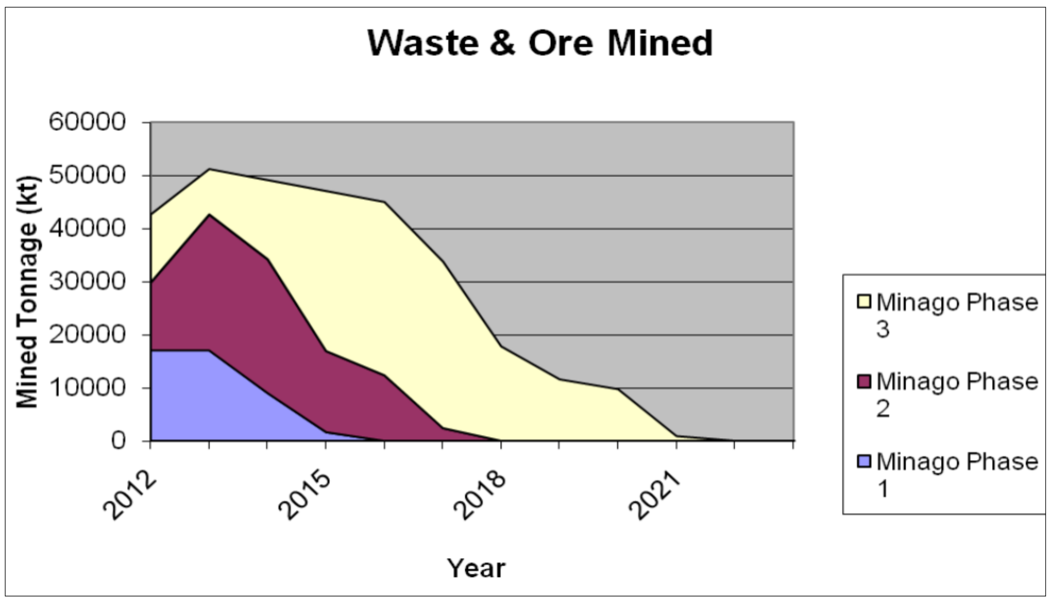
The Phase II geometry expands in all directions from the Phase I geometry to mine the next high grade blocks of the orebody. The final highwalls will be reached in the west and southwest of the ultimate pit shell to achieve the required minimum mining width. Phase II will mine 4.91 Mt of sand, 9.4 Mt 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.

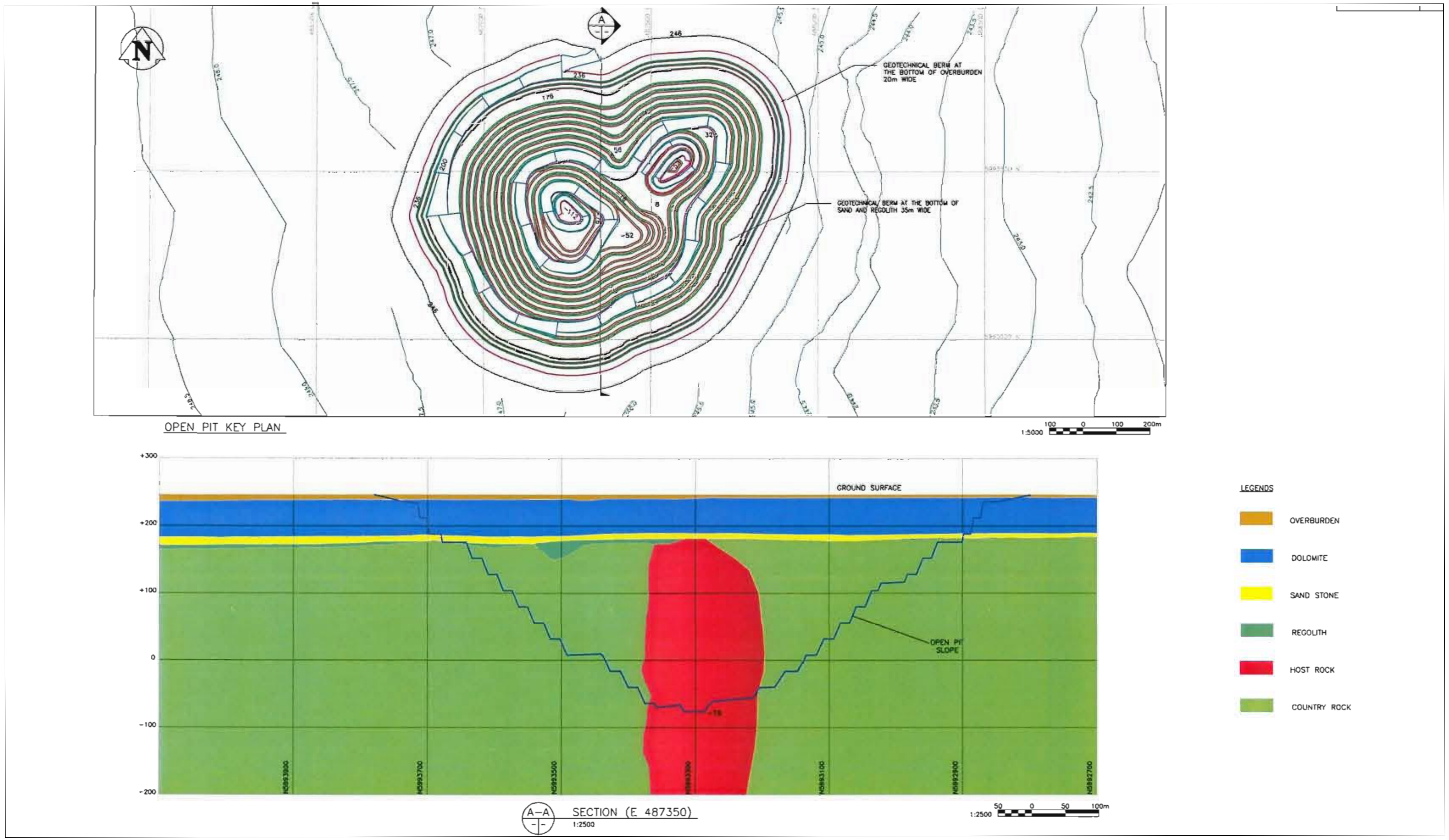
The mine will start delivering frac sand ore in the year just prior to Frac Sand production at the start of 2013. The delivery of nickel sulphide ore is scheduled to begin in late 2013 in preparation for Ore Processing at the start of “Year 1” (2014) and will continue until “Year 8” (2021).

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.

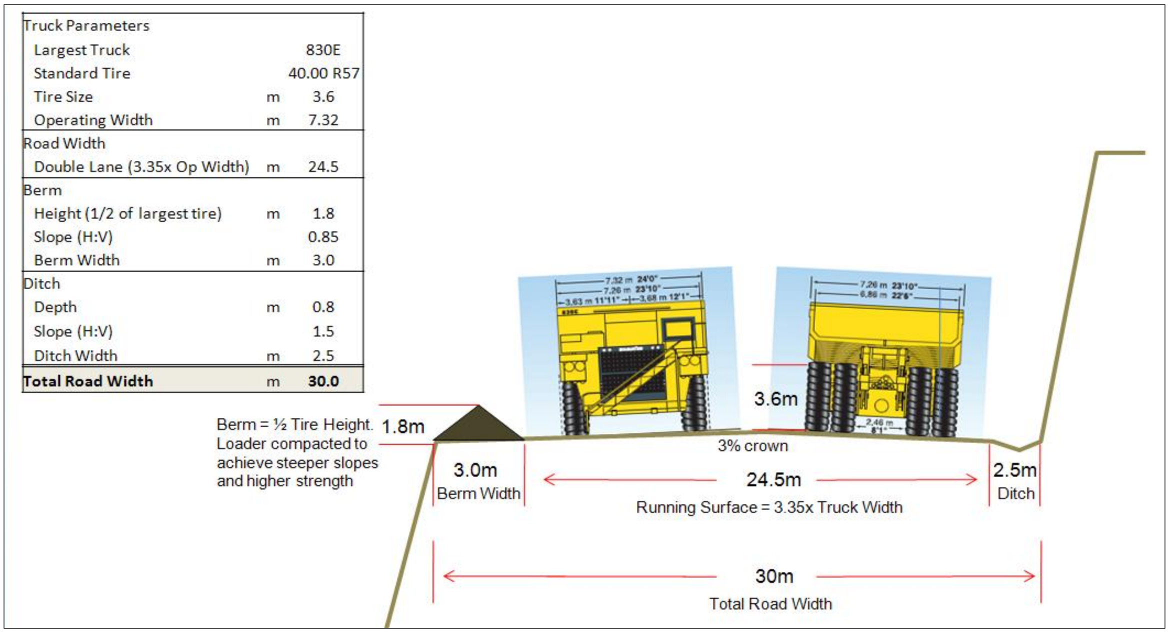
The sand will be mined over a period of 3 years and stockpiled. A Frac Sand Process Plant is projected to be commissioned during 2013 and it is anticipated that delivery of frac sand ore will begin in 2013.

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%.



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

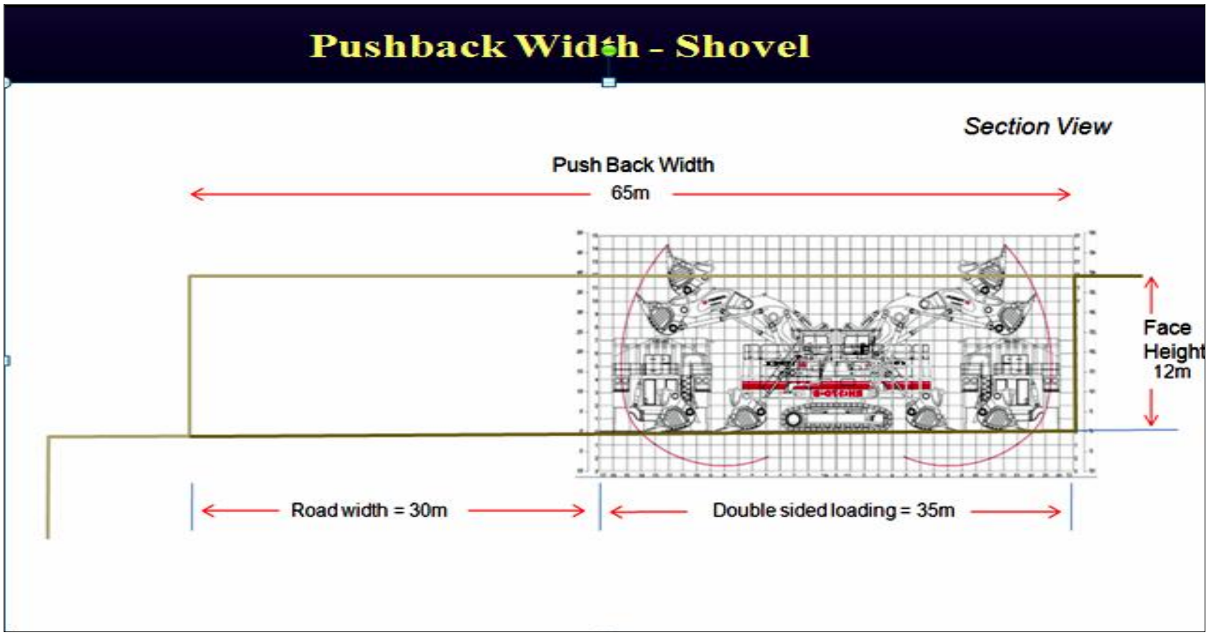
Figure 2.9-7 shows the proposed pushback width. The approximation of the pushback width was determined based on:

- 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):

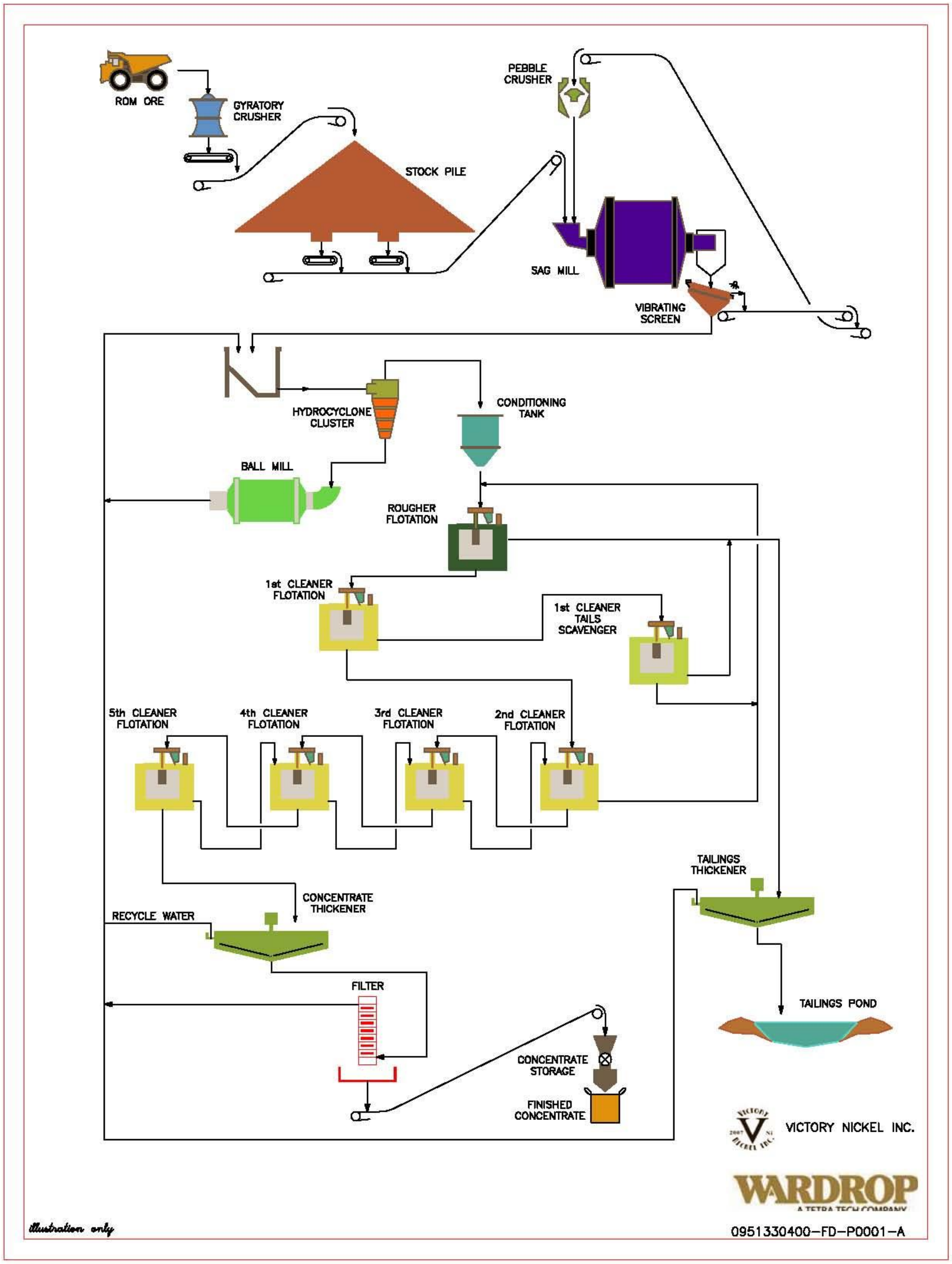
- primary crushing;
- ore stockpile and reclaim;
- grinding circuit and size classification;
- rougher/scavenger/cleaner flotation using reagants;
- concentrate dewatering using filter presses, bagging and load out; and
- tailings thickening.

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

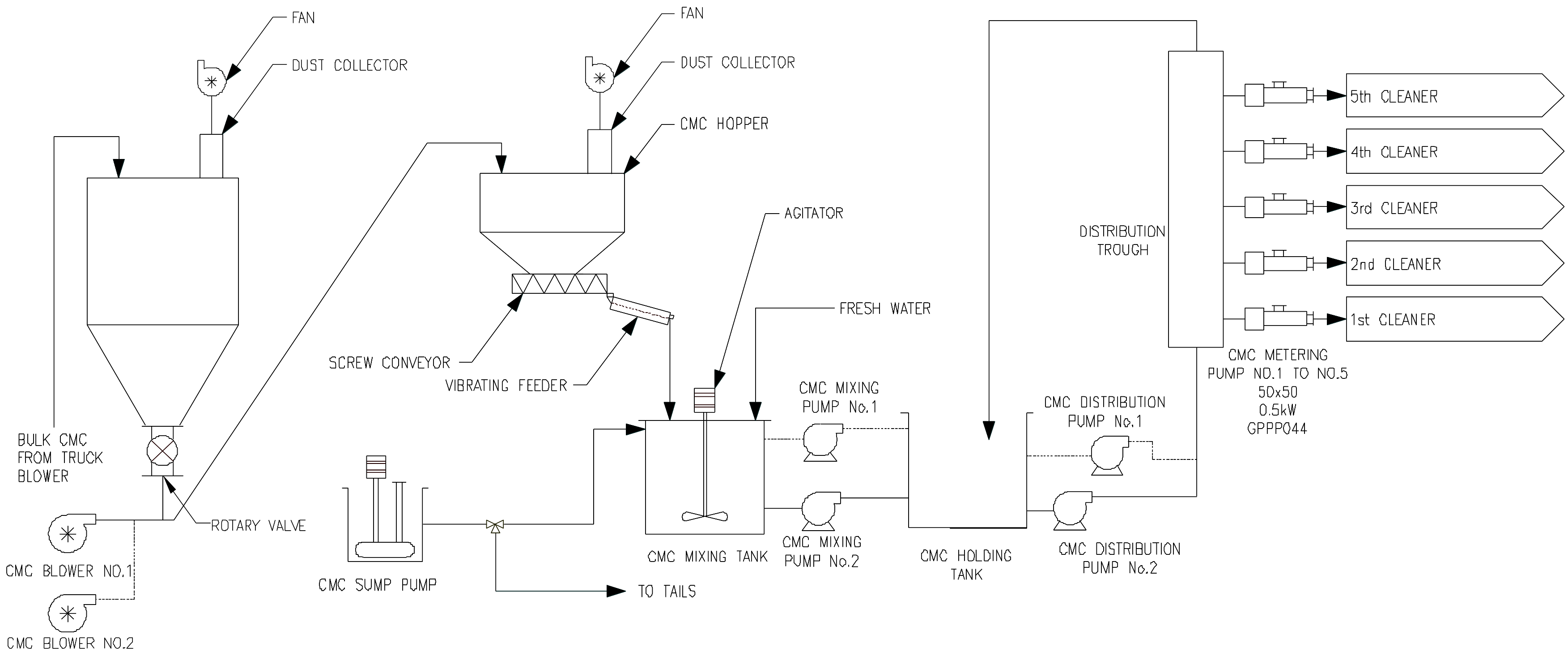


Figure 2.10-2 CMC Reagent Flow Sheet

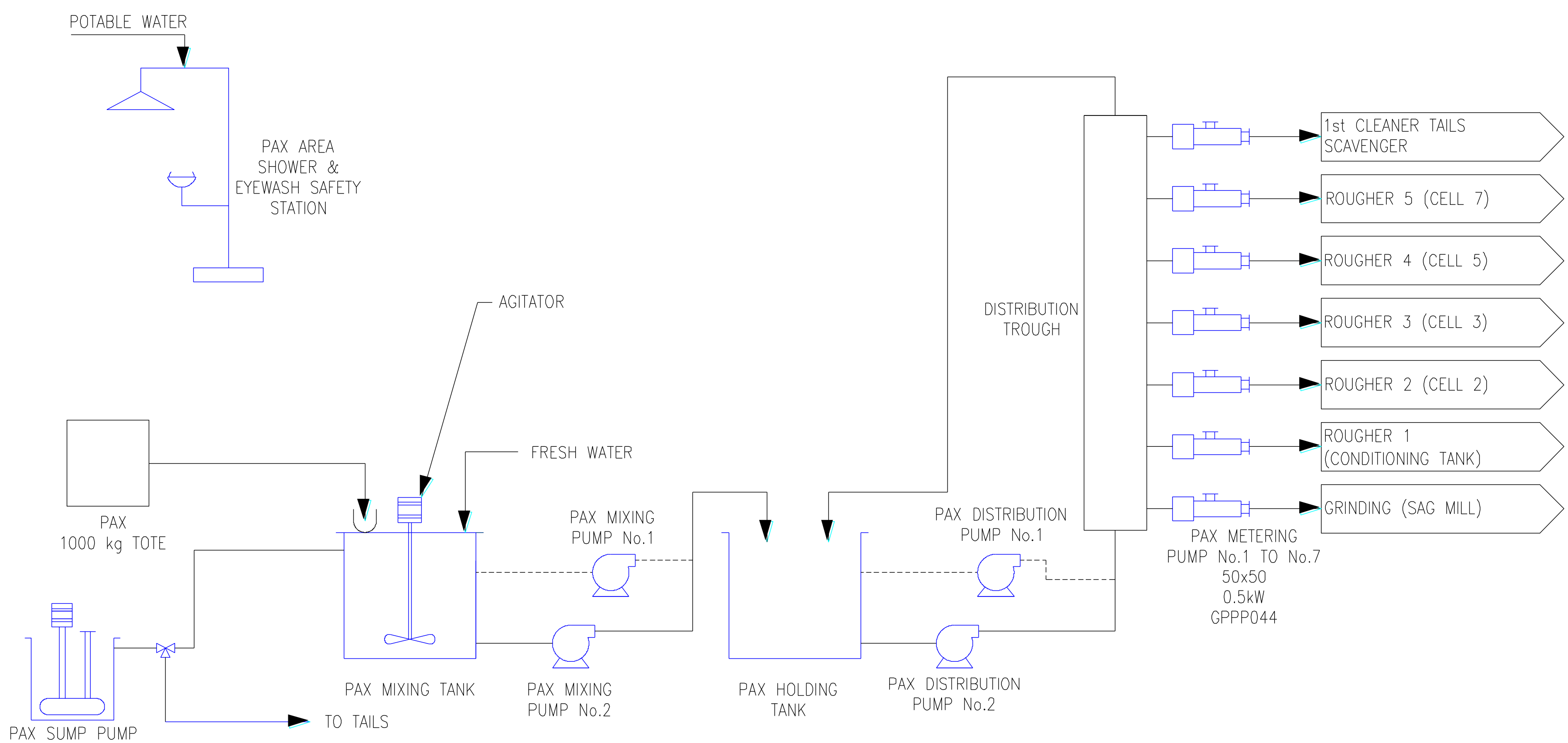


Figure 2.10-3 PAX Reagent Flow Sheet

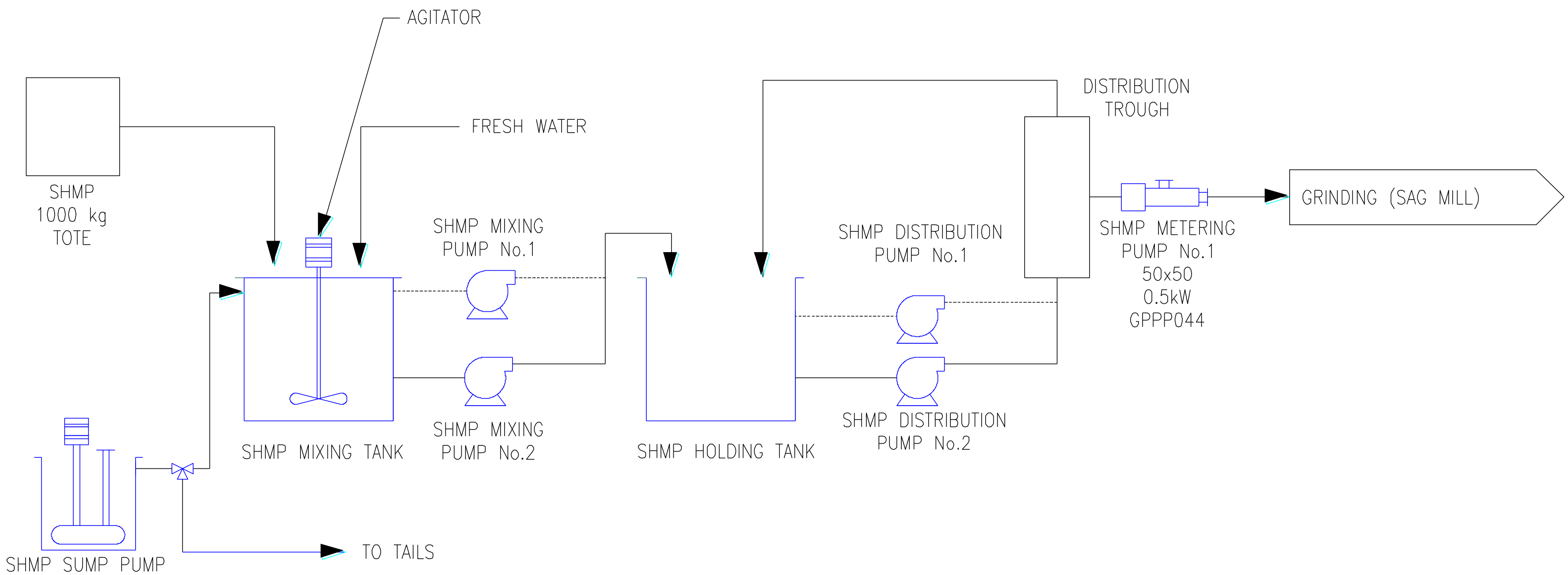


Figure 2.10-4 SHMP Reagent Flow Sheet

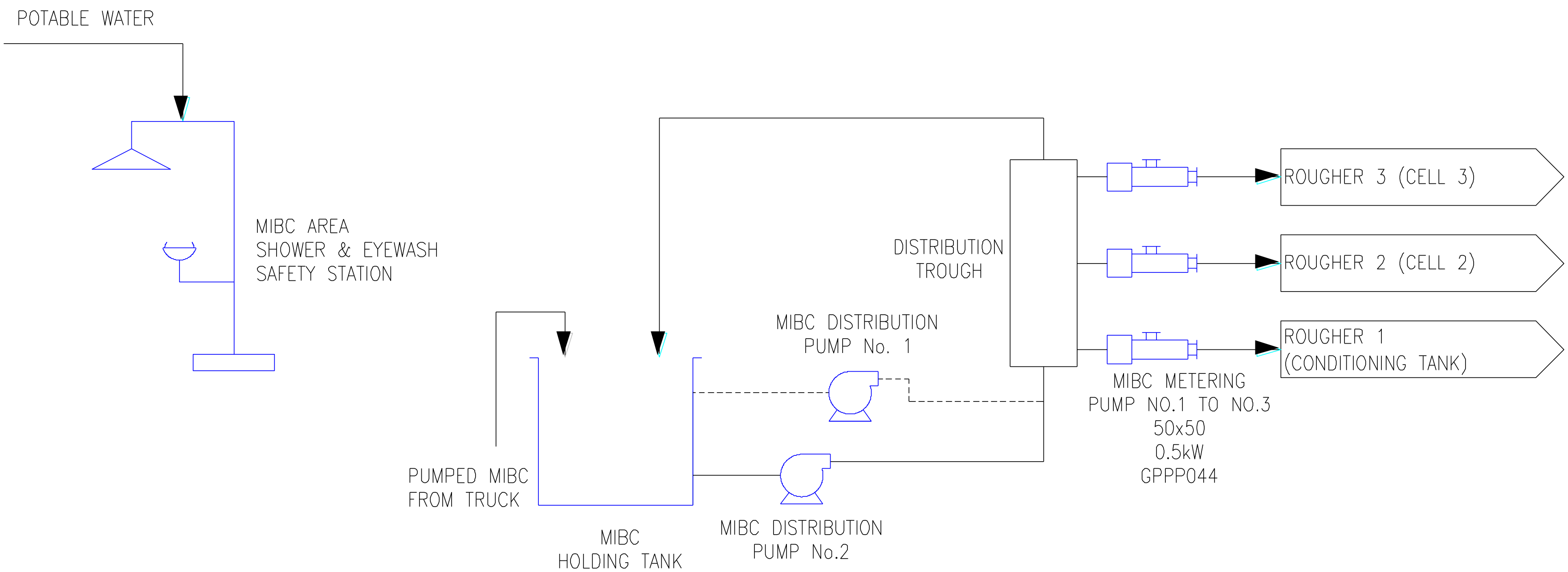


Figure 2.10-5 MIBC Reagent Flow Sheet

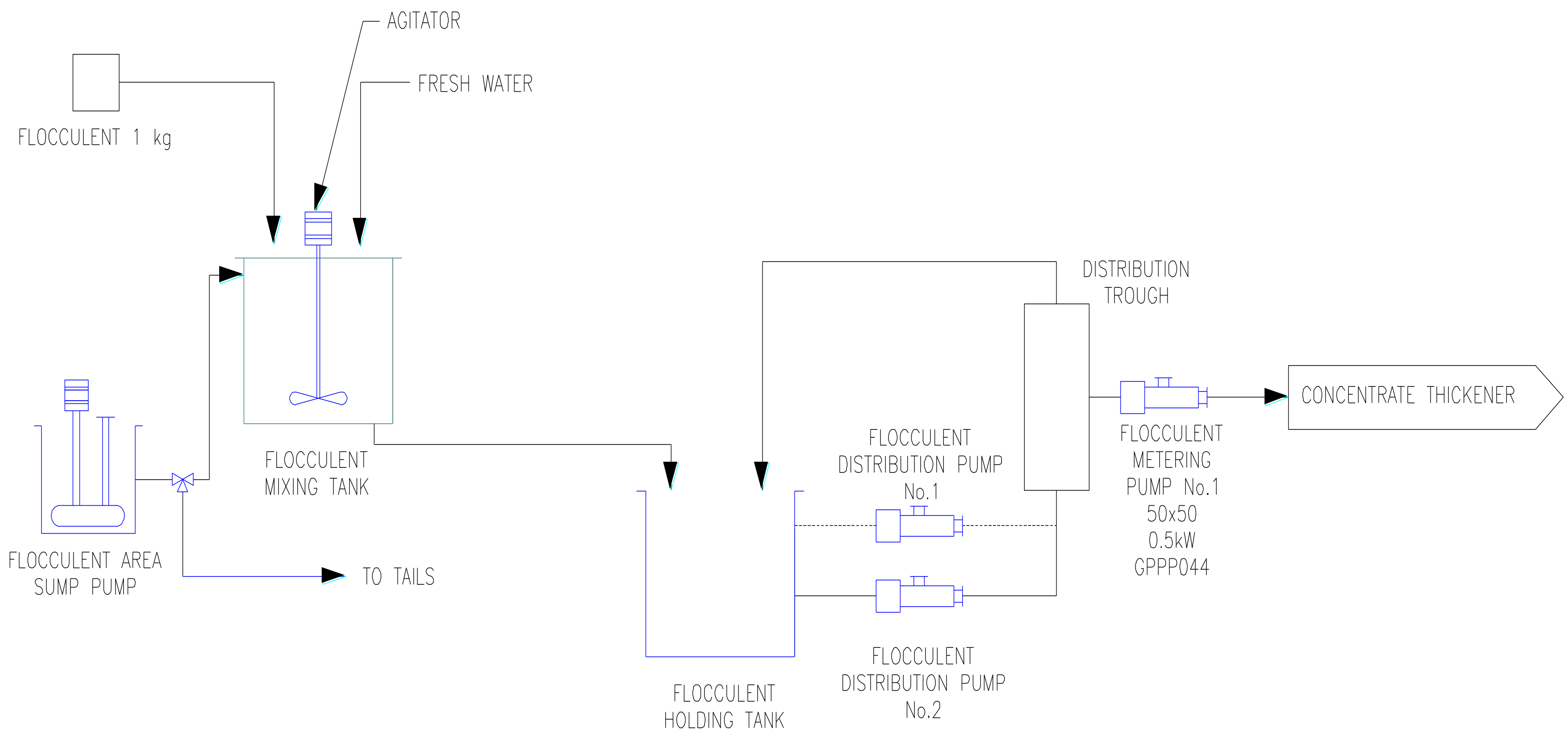


Figure 2.10-6 Concentrate Flocculant Flow Sheet

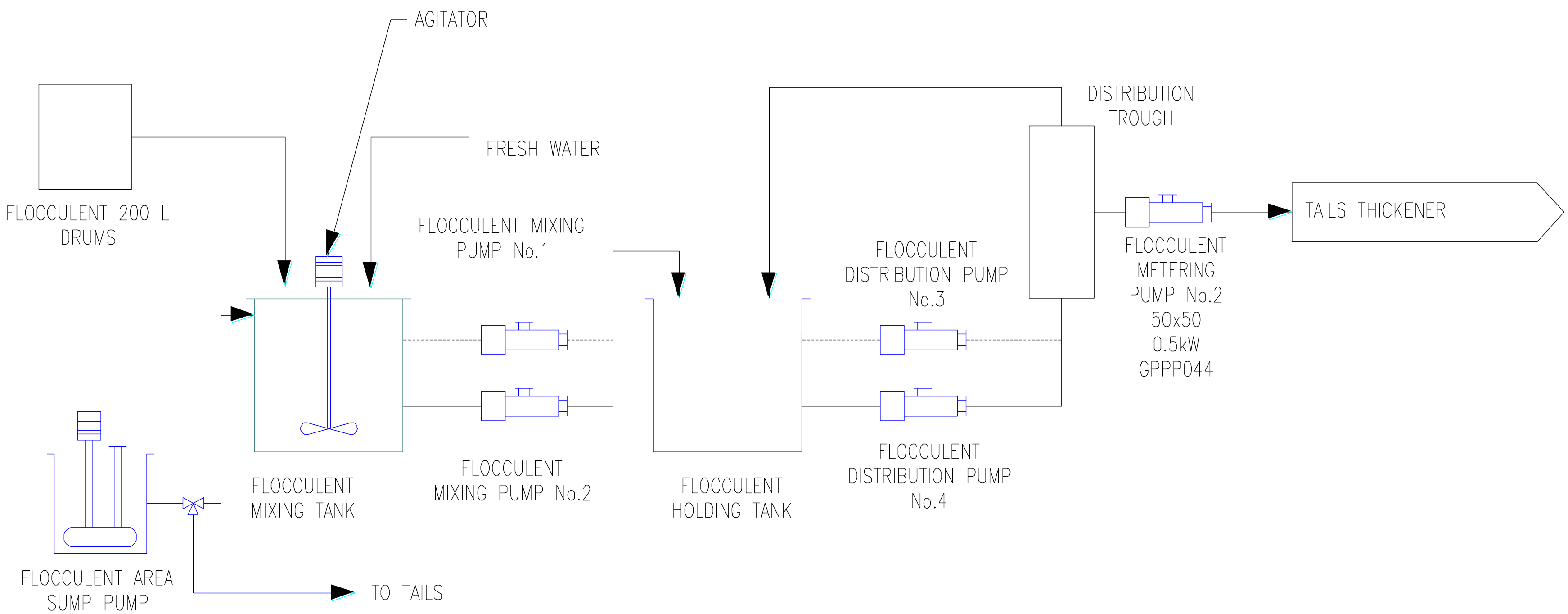


Figure 2.10-7 Tailings Flocculant Flow Sheet

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):

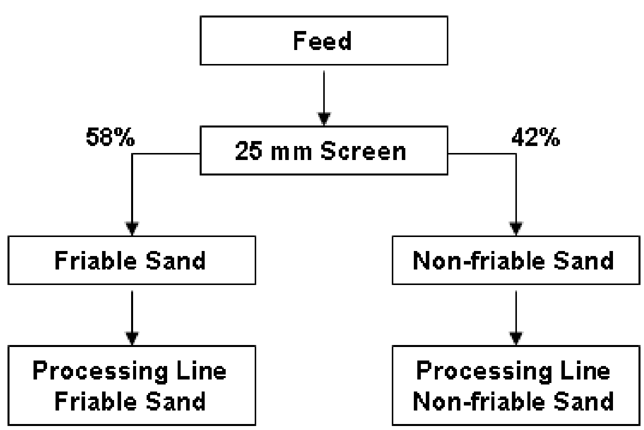
- Program 1: Between May and July 2007, Loring Laboratories Ltd. (Loring) of Calgary, AB performed mineralogical analyses, and EBA Consulting Engineers and Scientists (Material and Pavements Practice) (EBA) of Calgary, AB, performed material analyses.

- 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.

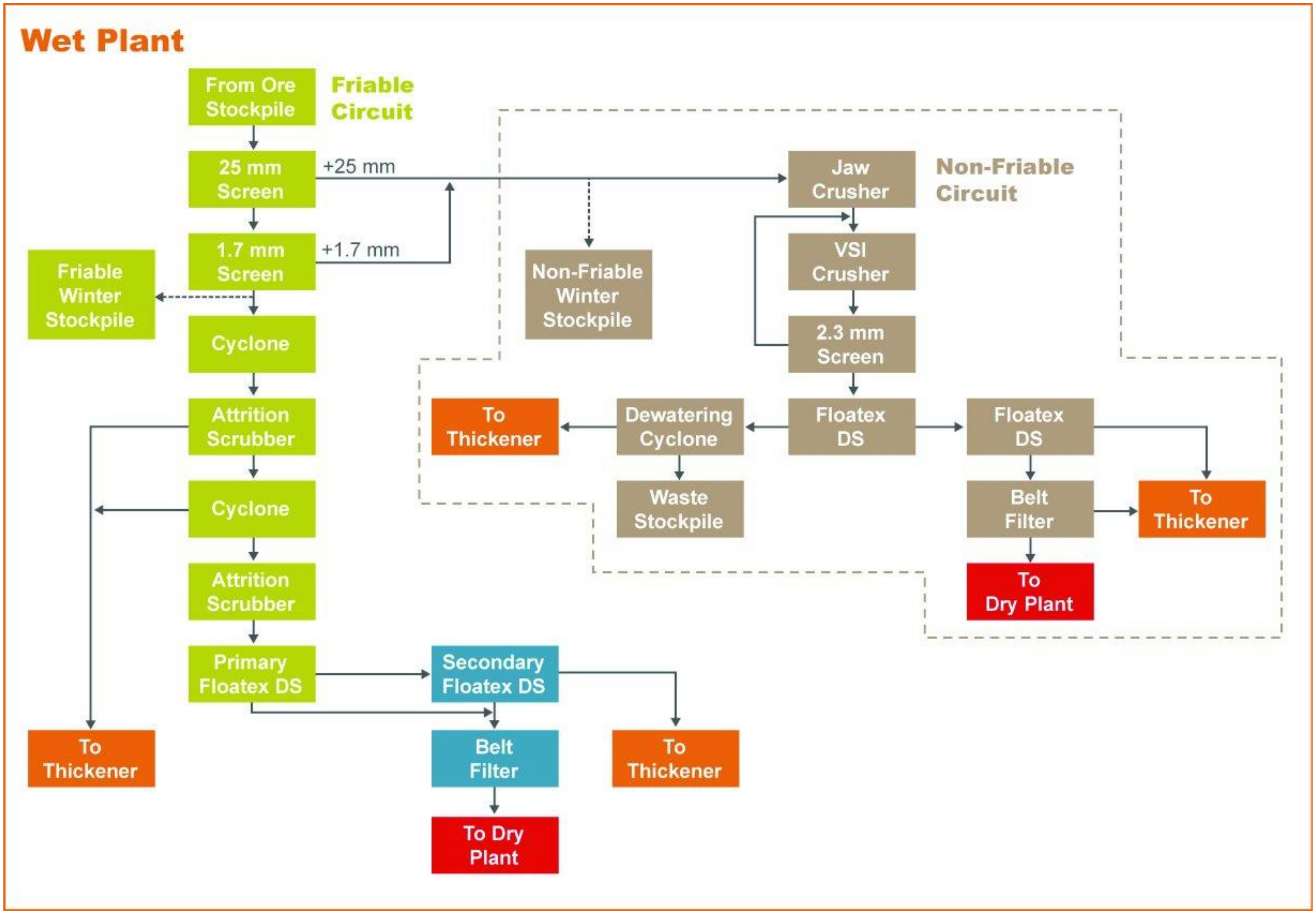
The Outotec Phase II design takes into account the seasonality fluctuating demands of the frac sand market, the inclement winter weather of Manitoba, Canada, and the need to operate the full plant year-round (Outotec, 2008). The wet and dry plants will operate together in series, and are designed to 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):

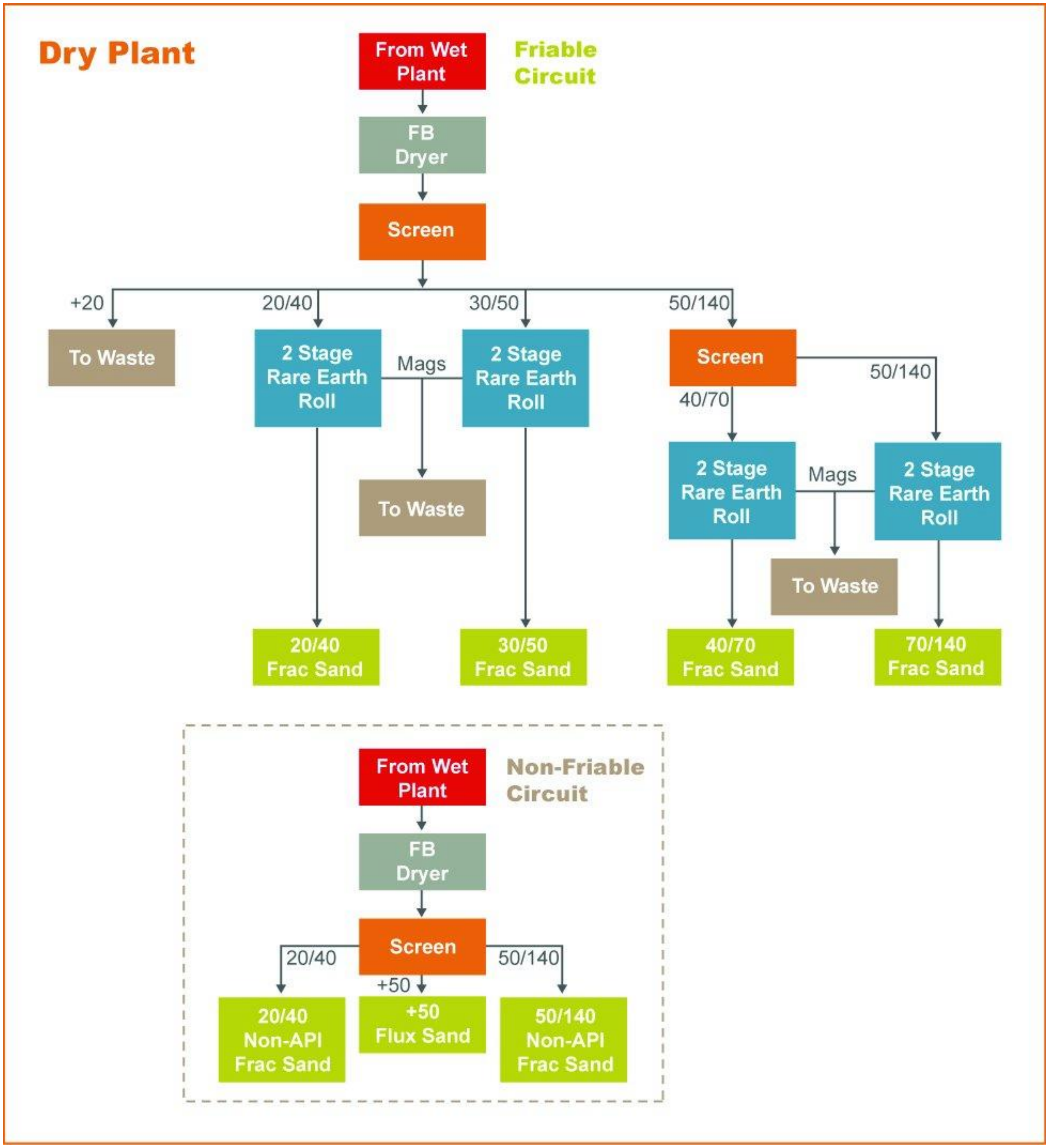
- Plant capacity of 1,142,805 t/y comprised of 612,863 t/y API frac sand, and 529,941 t/y non-API sand, which includes 62,500 t/y of flux sand;
- Plant feed rate of 265 t/h or 1,500,000 t/y,
- Yearly operating hours – 4,822, 12 months yearly operating window for wet and dry processes;
- Friable and non-friable 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).



Source: Outotec, 2008

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



Source: Outotec, 2008

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).

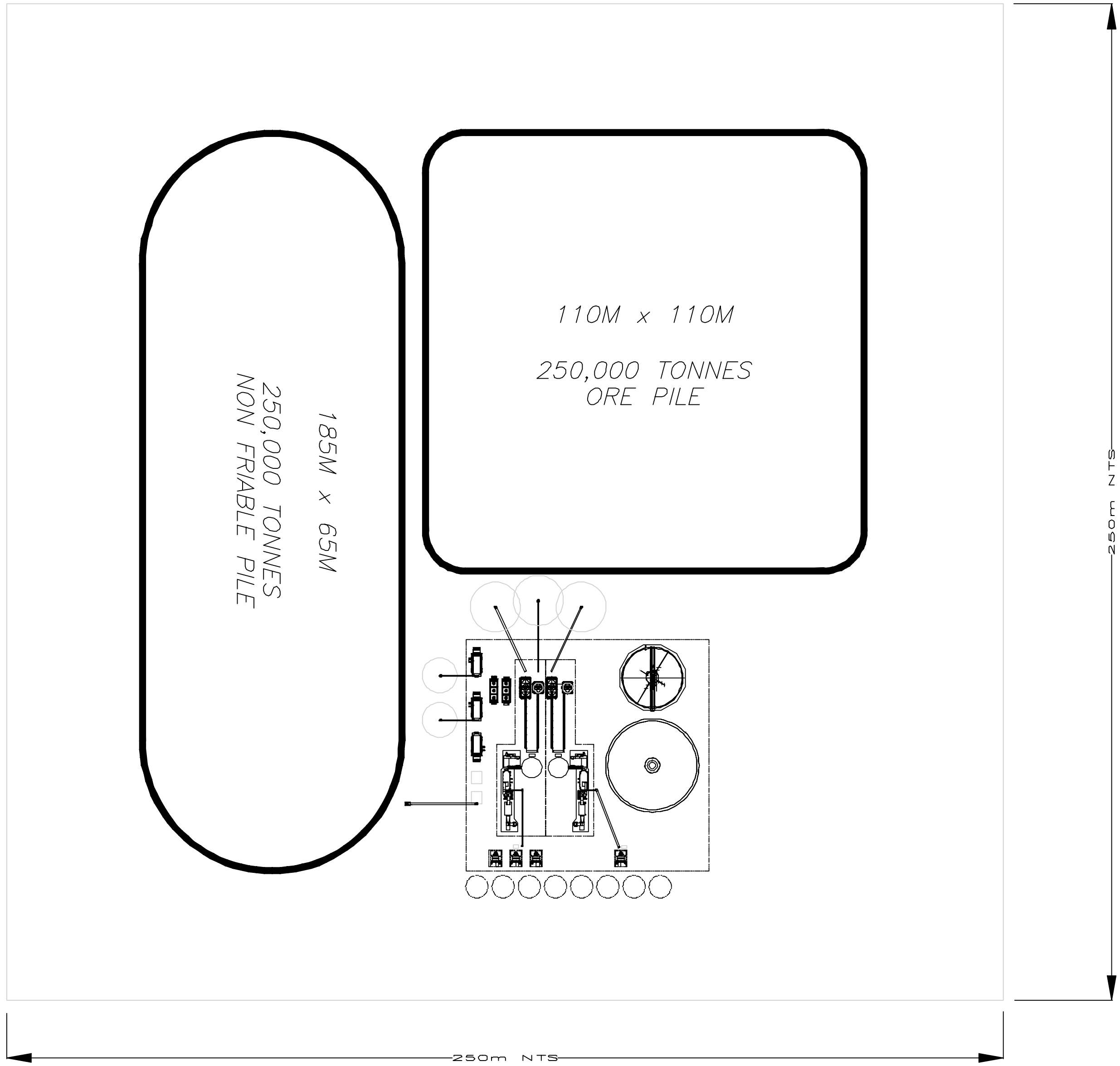
Outotec has been involved in the design and build of several fracturing sand plants. The estimate of bulk electrical and plant automation and control was based upon other similar frac sand plants designed by Outotec. Examples of P&ID diagrams for plants similar to the one envisioned for Minago are given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

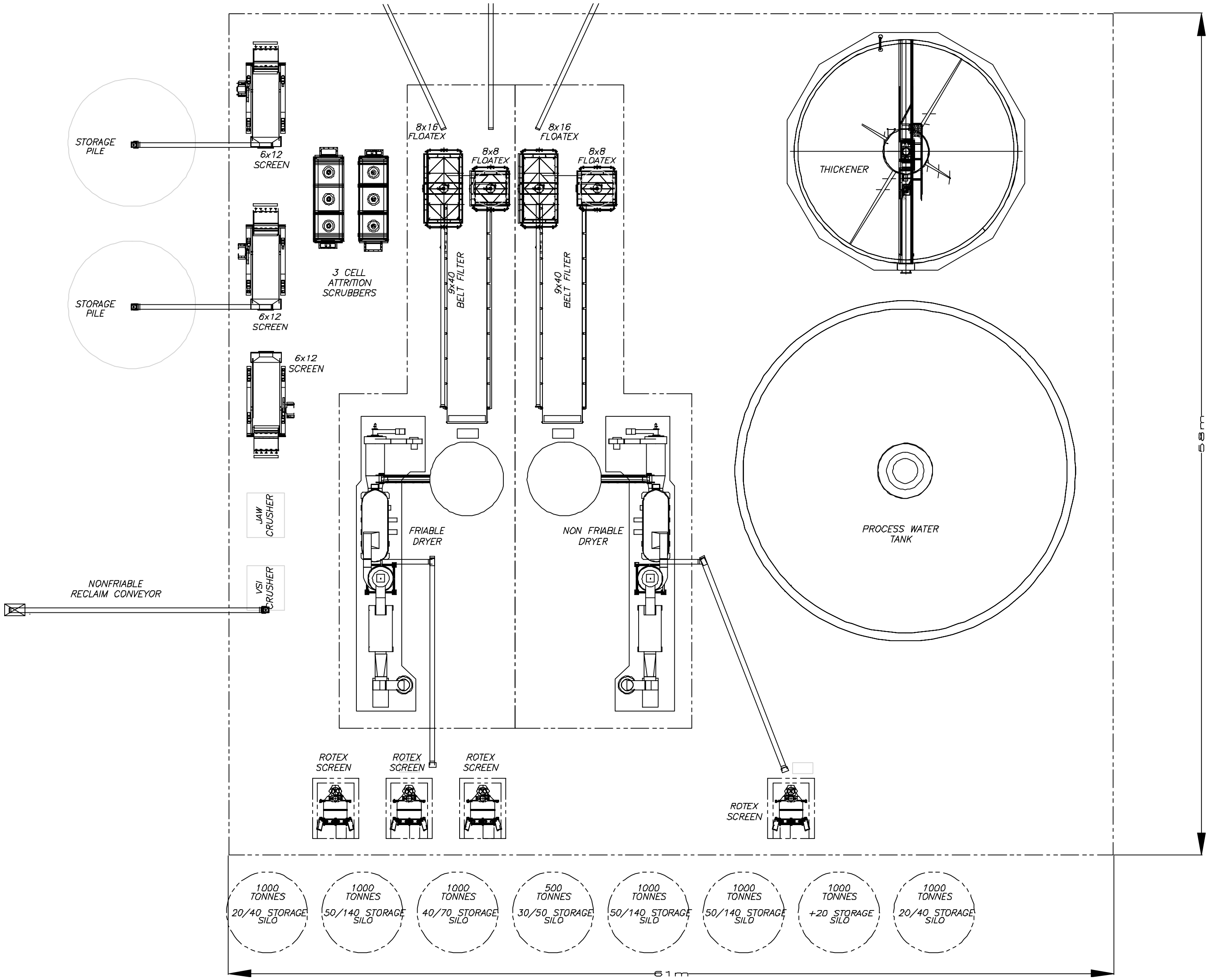
2.10.4.4 Rail Load-out Area

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.

2.11 Overburden Management

This section addresses the management of overburden material, which includes on-site clays and peat/muskeg. The management of dolomitic overburden will be presented in the Waste Rock Disposal Section (Section 2.12).

Overburden will be managed in several ways. The vast majority of peat and clay overburden that needs to be removed to gain access to the ore reserves and to built infrastructure will be stored in an Overburden Disposal Facility (ODF). Low permeability clays will be salvaged and stockpiled in sufficient quantities to enable the construction of low permeability liners where required. For example, a low permeability liner will be installed on the upstream side of the Tailings and Ultramafic Waste Rock Management Facility (TWRMF).

Dredging was selected as an overburden management option for the Minago Open Pit, because of logistical challenges, tight scheduling issues, and capital and operational costs related to safe disposal of mechanically excavated overburden (Wardrop, 2009b). Dredged material will be deposited in the ODF. Victory Nickel is also considering using mechanical equipment to remove the overburden material from the pit area. The mechanical removal option of the overburden will be undertaken during the winter months.

The ODF capacity will be approximately 15 Mm^3. The ODF will be capable of retaining a total of 11.2 Mt (~ 13.4 Mm^3) of overburden that will be discharged into the facility during an 8 months dredging period, scheduled to run from April to November “2011” (Year -3). A further 1.6 Mm^3 of swelled peat and soft clay will be added in “2012” (Year -2). This material will originate from the downstream side of the dam foundation of the TWRMF and from runoff and seepage collection ditches.

The ODF will be located immediately south and east of the open pit as shown in Figure 2.1-2.

2.11.1 Overburden Disposal Facility (ODF) Design Criteria and Design Basis

The in situ material quantities that were used as the design basis for the ODF are detailed in Table 2.11-1. The design basis for the ODF assumes that the overburden materials will be comprised of 50% solids and 50% water by weight. The change in solids mass from 70% prior to the dredging to 50% at the point of disposal will be a result of the mixing and pumping of the slurry. After deposition, a certain portion of the initial water content will be released to bring the longer term ratio to 65% solids and 35% water (Wardrop, 2009b). The estimated total mass and volume of solids and water upon deposition in the ODF are presented in Table 2.11-2.

The engineering design criteria used for the development of the ODF are presented in Table 2.11-3.

Table 2.11-1 In-situ Overburden Material Quantities

Item	Value
Effective Unit Weight	1.86 t/m³
Effective Moisture Content	52 %
Total Overburden Weight	11,200,000 t
Total Overburden Volume	6,021,000 m³
Effective Solids Content (By Weight)	66%
Effective Water Content (By Weight)	34%

Source: Wardrop, 2009b

Table 2.11-2 Design Basis Criteria for the ODF

Item	Value
In situ Solids Weight	
In situ Water Weight	
Solids Weight	7,347,000 t
Water Weight (at 50% water to 50% solids by weight)	7,347,000 t
Total Weight	14,694,000 t
Solids Volume	6,022,000 m³
Water Volume	7,347,000 m³
Total Volume	13,369,000 m³

Source: adapted from Wardrop, 2009b

Table 2.11-3 Basic Engineering Design Parameters for ODF

Item	Target	Comments
1. Geotechnical Slope Stability		
• Construction (in stages)	• Static F.O.S. 1.3, pseudo static F.O.S 1.05.	
• Normal Operating	• Same as above.	
• Closure	• Static F.O.S. 1.3, pseudo static F.O.S 1.05.	
2. Seismicity		
• Operating Design Basis Earthquake	• 1: 475 year return	
• Seismicity induced by pit blasting	•	• Input will be required for the detailed design.
• Closure Earthquake	1:2,475 year return	

Source: Wardrop, 2009b

2.11.2 ODF Design

The layout of the ODF is shown in Figures 2.11-1 and 2.11-2. The ODF will be surrounded by a perimeter dyke that will be approximately 4.5 m above the local topography and the dyke crest will be 12 m wide to accommodate construction traffic and facilitate feeder and discharge pipes (Wardrop, 2009b). Peat will be left in place in the dyke foundation.

The discharge of dredged peat and clay slurry will be through a number of discharge pipes spaced out along the ODF dyke crest. Carriage water that was used to transport the solids will be released from the ODF through a series of stop log weirs constructed in the perimeter dyke at the central apex of the ODF. The weirs will pass the water into a triangular collection pond contained by another dyke. The collected carriage water will then be reused for dredging operations. In addition, a 0.3 m perforated HDPE or ADS pipe will be installed in the ODF apex to enhance carriage water collection efficiency during and post the dredging operations (Wardrop, 2009b).

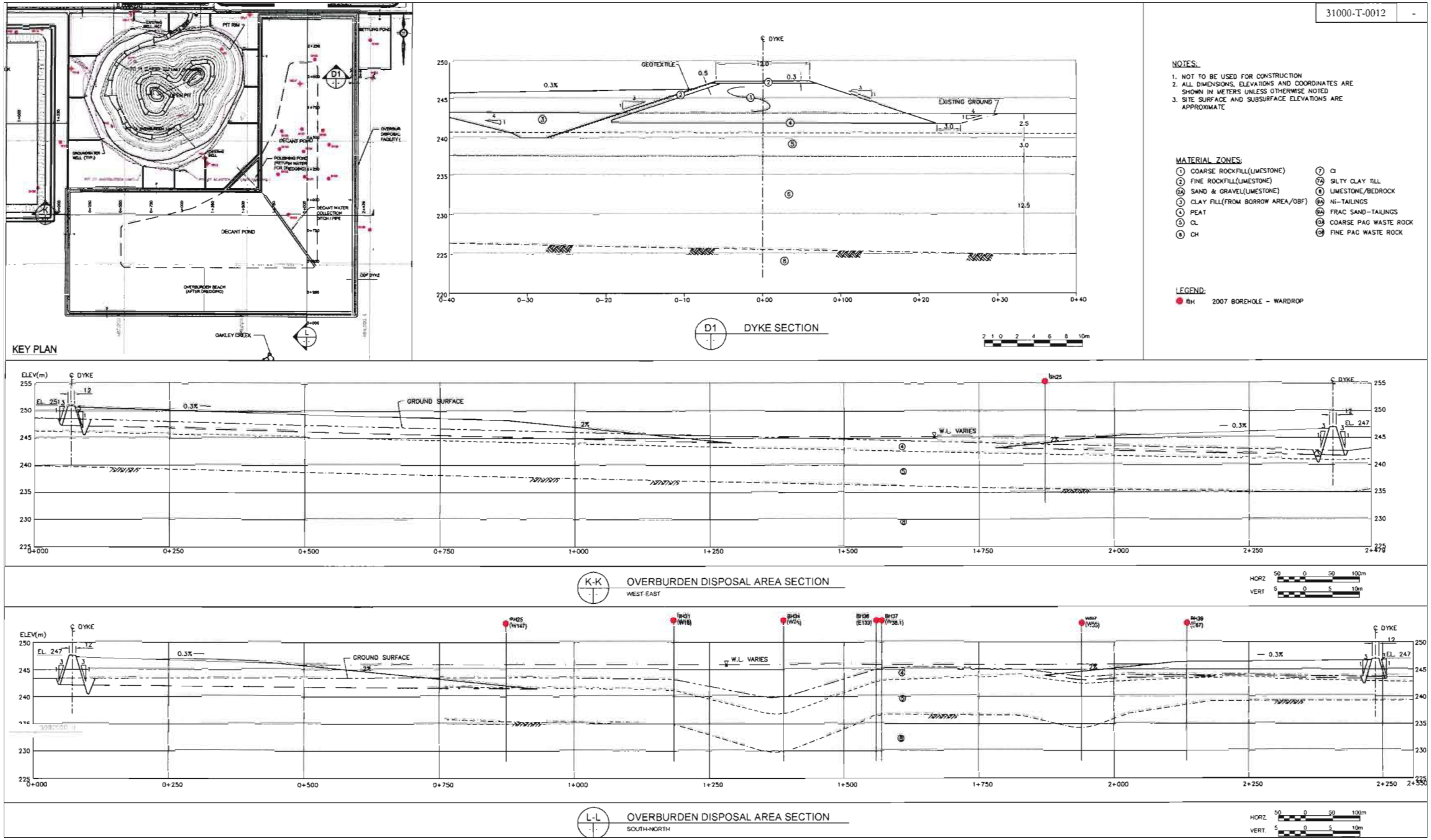
2.11.2.1 Dredging Operations

The peat and clay soils will be removed using a hydraulic dredging process utilizing a boom mounted rotating cutter attached to barge. The boom will have sufficient length and flexibility to cut the overburden material to vertically and horizontally control the cutter to accurately remove the overburden materials to the desired plan and profile (Wardrop, 2009b).

The selection of the cutter head size and number of dredge units will be identified in the detailed engineering design with input from dredging contactors. Preliminary discussions with a dredging contractor suggest that two 1 m diameter cutter units may be required for the Minago Project. Water will be added at the cutter head to facilitate the conveyance of the solids to the ODF. The water and solids slurry will be pumped through a pipeline system by booster pumps to the ODF and discharged within the operating cell of the ODF (Wardrop, 2009b).

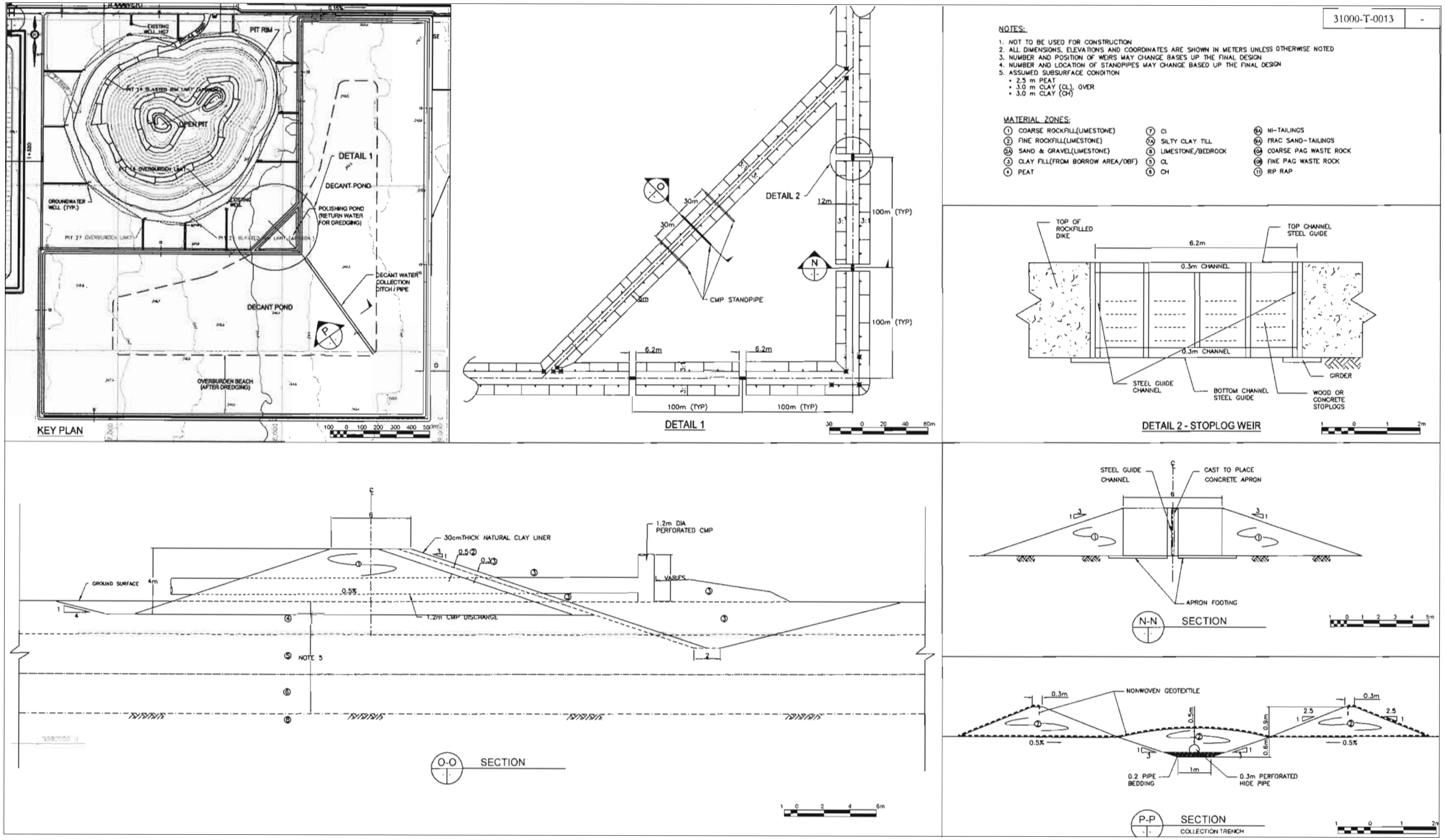
During the dredging operation, the slurry is expected to be comprised of 20% solids and 80% water by weight. For the planned 8-month dredging period, the estimated dredging production will be approximately 25,000 m^3/day (46,500 tonnes/day) of *in situ* overburden (Wardrop, 2009b).

The disposal strategy will involve perimeter discharge of a peat and clay slurry starting along the western side of the southern leg of the ODF and continuing in parallel along the northern and southern sides (Figure 2.11-1). The same strategy will apply to the eastern ODF leg where the deposition will start at the northern side and will continue along the western and easterm sides. The dredged material is expected to form a beach at a 0.3 % slope and a 2 % subaqueous slope (Wardrop, 2009b). The beach will divert decant water towards the pervious dyke section. Decant water from the dredging operations will be collected in the decant water collection pond, shown in Figure 2.11-2.



Source: adapted from Wardrop's drawing 0951330400-T0012 (Wardrop, 2009b)

Figure 2.11-1 Overburden Management Facility Plan and Sections



Source: adapted from Wardrop's drawing 0951330400-T0013 (Wardrop, 2009b)

Figure 2.11-2 ODF Water Management Structure Plan and Sections

The outboard pond dyke will be constructed out of coarse limestone rock fill that will be 4 m high, a 0.5 m of fine limestone rock fill on the upstream side, and a 0.3 m thick inboard clay liner to increase the dyke's water holding capacity.

To effectively manage water release and to support continued dredging operations, a total of three 1.3 m in diameter corrugated metal pipe (CMP) will run through the dyke (laterals) and these will be connected to perforated standpipes installed within the pond (Wardrop, 2009b). Collected water will be returned to the dredging operations for continued dredge water demand. It is estimated that over the eight months dredging period, approximately 7.4 Mm^3 of make-up water will be required. To support the dredging operations and assuming a 15 percent water loss, the estimated make-up water demand will be approximately 35,000 m^3/day (Wardrop, 2009b).

Water pumped from the pit dewatering wells will be used for the dredging operations. The cone of depression created by the groundwater dewatering wells will provide under drainage for the overburden clays. This will be considered in geotechnical analyses for major site earth/rock fill structures.

The water level in the dredging pit will be drawn down at the end of the dredging period to assist in the de-watering of the dolomite (Wardrop, 2009b).

On closure, the ODF will be reshaped and revegetated and overflow will be directed to the ditch near Highway #6 that reports to Oakley Creek (Wardrop, 2009b).

2.11.3 ODF Dyke

Plan and section views of the ODF dyke are shown in Figure 2.11-1 and Figure 2.11-2 illustrates a plan view, a dyke design section, a stop log structure section, and details for the ODF Polishing Pond.

The ODF dyke will be constructed out of coarse rockfill (Zone 1 material) that will be comprised of 800 mm minus dolomite waste rock originating from the limestone outcrop located approximately 3 km northwest of the facility (Figure 2.1-2). The upstream side of this zone will support a 0.5 m thick zone of fine rockfill (Zone 2 material) comprised of minus 75 mm dolomite waste rock. A geotextile layer will be placed on the upstream side over the top of Zone 2. The dyke crest will be 12 m wide and both upstream and downstream slopes will be 3H:1V (Wardrop, 2009b).

The ODF Polishing Pond dyke will also be constructed out of coarse rockfill (Zone 1 material) and a 0.5 m thick fine rockfill (Zone 2 material) on the upstream side. Both upstream and downstream slopes will be 3H:1V. A 0.5 m clay liner will be provided on the upstream side of Zone 2. A total of three DMP pipes, 1.3 m in diameter and sloped at 0.5%, will be installed within the dyke. These pipes will have vertical perforated intakes immediately upstream of the dyke (Wardrop, 2009b).

2.11.3.1 ODF Dyke Stability and Seepage Analyses

Seepage and slope stability analyses were performed on the ODF dyke Sections D1 and D2. Section D1 assumes that there will be 2.5 m of peat, over 3.0 m upper clay (CL), on top of 12.5 m of lower clay (CH) (Figure 2.11-1). Section D2 assumes that there will be 2.5 m of peat, over 3.0 m upper clay (CL), on top of 3.0 m of lower clay (CH) (Wardrop, 2009b).

Coupled analyses using Sigma/W and Slope/W, components of GeoStudio 2007, were used in the Seepage and slope stability analyses. Sigma/W uses finite element methods to solve both stress-deformation and seepage dissipation equations simultaneously. Pore water pressures generated during lift placement were calculated with Sigma/W and then incorporated into Slope/W for stability analysis. Slope/W was used to locate failures with the least factor of safety within defined search limits (Wardrop, 2009b).

Sections D1 and D2 were modeled assuming that the embankment was placed in a single lift on the first day, and then 20 days were allowed for consolidation. Slope stability analyses were conducted by assuming that 4 days had passed after the embankment had been placed and at the end of 20 days (Wardrop, 2009b).

Slope stability analyses were performed on the upstream and downstream sides of the ODF dyke. Another analysis was performed 30 days after the completion of the facility, assuming that the disposed peat and clay material were placed at once on the upstream side. After that, a seepage analysis was performed under steady state conditions to calculate the seepage through the ODF dyke (Wardrop, 2009b).

A pseudo static analysis was also performed to simulate earthquake conditions using an acceleration of 0.03 g (Wardrop, 2009b).

Material Properties

Assumed foundation material properties (CL, CH and bedrock) were based on field and laboratory data. Assumed properties for peat, coarse and fine rockfill, and dredged peat and clay were based on previous experience and professional judgement. Table 2.11-4 and Table 2.11-5 show material properties used in Sigma/W, Seep/W and Slope/W for the ODF dyke.

2.11.3.1.1 ODF Dyke Stability Results

Table 2.11-6 presents slope stability results assuming that 4 days and 20 days had passed after the placement of the facility, and 10 days after the ODF was filled with dredged peat and clay material. The slope stability results show that the ODF satisfies the minimum requirements for static and pseudo static conditions. Detailed slope stability results are given elsewhere (Wardrop, 2009b).

Table 2.11-4 Sigma/W Input Material Properties

Materials	Material Category	Material Model	Poisson's Ratio	Young's Modulus (kPa)	Hydraulic Conductivity (cm/s)*
Disposed Peat and Clay	Effective Parameters w/PWP Change	Linear Elastic	0.33	2,000	8.64E-03
Coarse Rockfill	Effective Parameters w/PWP Change	Linear Elastic	0.33	50,000	8.64E-01
Fine Rockfill	Effective Parameters w/PWP Change	Linear Elastic	0.33	7,000	8.64E-03
Sand and Gravel	Effective Drained Parameters	Linear Elastic	0.35	8,000	
Peat	Effective Parameters w/PWP Change	Linear Elastic	0.35	2,000	1.00E-01
Soft Clay (CL)	Effective Parameters w/PWP Change	Soft Clay (MCC)	0.36		1.36E-08
Soft Clay (CH)	Effective Parameters w/PWP Change	Soft Clay (MCC)	0.37		4.97E-09
Bedrock	Effective Parameters w/PWP Change	Linear Elastic	0.49	100,000	6.89E-04

Source: Wardrop, 2009b

Note: *Used in Seep/W.
w/PWP Change with porewater change

Table 2.11-5 Slope/W Input Material Properties

Materials	Model	Unit Weight (kN/m^3)	Cohesion (kPa)	Phi (º)
Disposed Peat and Clay	Mohr-Coulomb	16	18	0
Coarse Rockfill	Mohr-Coulomb	19	0	40
Fine Rockfill	Mohr-Coulomb	22	0	38
Sand and Gravel	Mohr-Coulomb	22	0	35
Peat	Mohr-Coulomb	13	18	0
Soft Clay (CL)	Mohr-Coulomb	21	20	29
Soft Clay (CH)	Mohr-Coulomb	18	10	25
Bedrock	Bedrock (Impenetrable)			

Source: Wardrop, 2009b

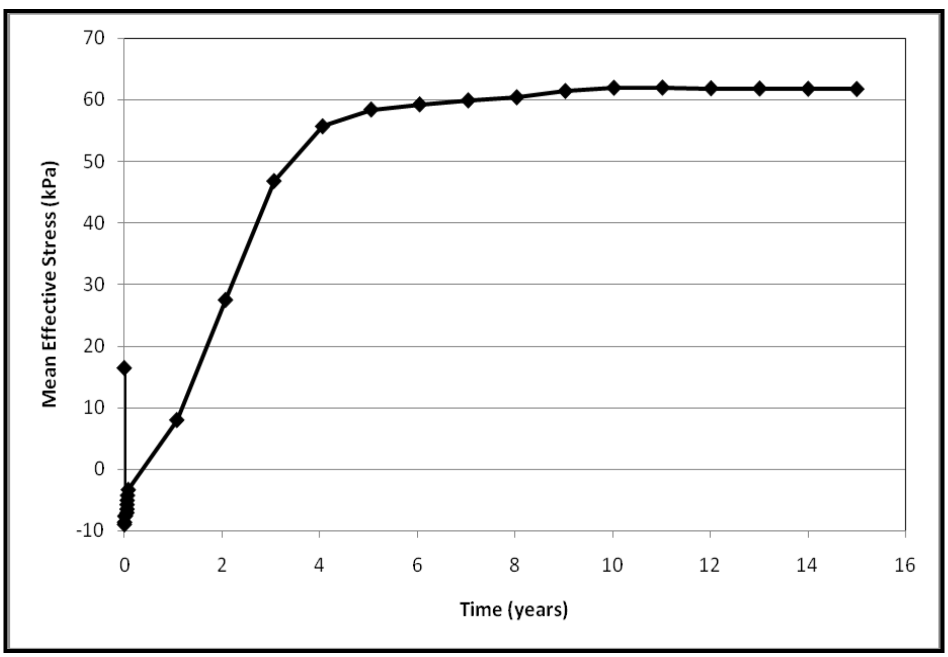
Table 2.11-6 Slope Stability Results for the ODF Dyke

Section	Elapsed Time (days)	Upstream		Downstream	
		Static F.O.S	Pseudo static F.O.S	Static F.O.S.	Pseudo static F.O.S.
D1	4	1.3/1.32		1.30/1.23	
	20	1.3/1.47	1.05/1.32	1.30/1.45	1.05/1.32
	30*			1.30/1.51	1.05/1.36
D2	4	1.30		1.32	
	20	1.39	1.25	1.36	1.25
	30*			1.48	1.34

Source: Wardrop, 2009b

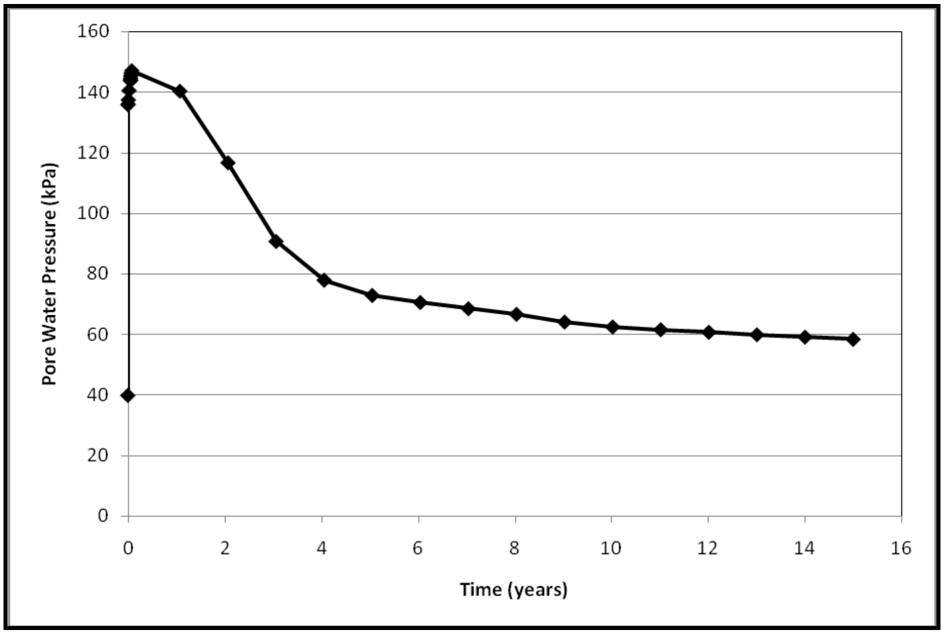
*Assumed disposed peat and clay material was placed on the upstream side of the embankment.

Figures 2.11-3 and 2.11-4 show modelling results for effective stress versus time, and pore water pressure versus time predicted for the foundation soils below the centerline of the dam. Figure 2.11-3 illustrates how the effective stress increases after placing the embankment, and then stabilizes over time. Figure 2.11-4 shows the pore water pressure increase upon the dyke construction and its dissipation over time. Based on these computations, full pore water pressure dissipation will occur in approximately 15 years.



Source: Wardrop, 2009b

Figure 2.11-3 Mean Effective Stress versus Time for the ODF Dyke



Source: Wardrop, 2009b

Figure 2.11-4 Pore Water Pressure versus Time for the ODF Dyke

2.11.3.1.2 ODF Seepage Results

Seepage through the embankment was estimated using Seep/W for a one meter wide slice or rockfill material against the upstream perimeter of the dam. The computed seepage quantities for sections D1 and D2 were in the order of 50 m^3/day. The initial seepage rate is expected to be much higher until a seal is created by the discharged peat and clay (Wardrop, 2009b).

2.11.4 Construction Considerations

2.11.4.1 Peat Overburden

The in-situ peat is unsuitable for construction purposes, but it may have potential for use in site reclamation. If pre-loaded, the peat may be used as foundation material for structures that are not sensitive to settlements, such as waste rock dumps (Wardrop, 2009b). Pre-loading tests on the peat were not carried out for determination of consolidation characteristics. These tests will be conducted during the detailed engineering design phase.

2.11.4.2 Clay Overburden

The construction of water containment structures and dykes across the site will require low permeability materials. Site clays were assessed during the pre-feasibility and feasibility geotechnical investigations and the results of laboratory tests on selected clay samples may be summarized as follows (Wardrop, 2009b):

- The optimum moisture content ranged from 16.3% to 18.6% at standard Proctor maximum dry densities (SPMDD) ranging between 1,600 and 1,752 kg/m^3.
- Clay with natural moisture contents reasonably close to the optimum for compaction may be found within the uppermost 5 m of the deposit. The moisture content of the tested clays was typically well above the optimum at depths greater than 5 m. The natural moisture content of tested clay was generally higher than 20% (Figure 7.3-7).
- It was found that site areas with shallow thickness of overburden contained stiff clays that exhibited natural moisture contents close to the optimum for compaction.
- Recovery of clays from perennially flooded terrain will pose formidable logistical challenges as the muskeg/peat is water logged. More specifically, these areas will require that the muskeg/peat are bermed off so that the upper stiff clay may be excavated in a “dry” condition. Also, clays may experience moisture uptake during excavation even if the borrow areas are bermed off (Wardrop, 2009b).

2.11.5 Overburden Removal using Mechanical Equipment

Victory Nickel is evaluating alternative options to hydraulic methods as the removal of the material using conventional methods (excavator, load, haul) are generally feasible during the winter months. There will be additional impacts should VNI decide to use mechanical methods.

2.12 Waste Rock Disposal

During the operation of the open pit, a total of 268.695 Mt of waste rock will be mined out of which 111.03 Mt will be limestone and 157.67 Mt will be basement rock. Basement rock will consist of two types: 122.01 Mt of granite (non-acid generating) and 35.66 Mt of ultramafic (potentially acid-generating and selenium containing). A summary of projected material quantities that will be mined from the Open Pit until closure is given in Table 2.9-6 and the yearly waste rock placement schedule is detailed in Table 2.12-1.

Waste rock will be deposited in three areas (Figure 2.1-2). Dolomitic waste rock will be deposited in the 191 ha Dolomite Waste Rock Dump, granitic waste rock will be deposited in the 301.4 ha Country Rock Waste Rock Dump, and ultramafic waste rock will be co-disposed with the tailings in the 219.7 ha Tailings and Ultramafic Waste Rock Management Facility (TWRMF). All of the waste rock disposal areas will be located close to the open pit to minimize haulage costs and to optimize utilization of the site.

Limestone will be used in the construction of roads, containment berms, the basement layer for the ultramafic waste rock and causeways inside the Tailings and Ultramafic Waste Rock Management Facility (TWRMF), and for the site preparation of a Crusher Pad and a Ore Stockpile Pad; excess limestone will be deposited in the Dolomite Waste Rock Dump (Dolomite WRD).

2.12.1 Design Criteria and Considerations for the Waste Rock Dumps

The key design objective is to construct non-reactive waste rock dumps in the proximity of the open pit within compact footprints to the maximum heights governed by geotechnical analyses to minimize operational costs. As the dolomitic and Country Rock waste rock is inert, no special environmental protection measures are necessary (Wardrop, 2009b).

Tables 2.12-2 and 2.12-3 summarize the basic design criteria and parameters adopted for the waste rock dumps.

2.12.2 Waste Rock Dump Designs

The design of the waste rock dumps focusses on minimizing dump footprints and maximizing their heights through staged construction and in accordance with the results of engineering analyses and the waste production schedule. With both dumps containing non-acid generating (NAG) waste rock, there will not be a need for a seepage collection system and the storm water can report directly to the natural environment.

The locations of Country Rock Waste Rock Dump (CRWRD) and Dolomite Waste Rock Dump (DWRD) were selected to be on muskeg/peat covered weak overburden clay characterized by average thicknesses of 15 m and 10 m, respectively.

Table 2.12-1 Yearly Waste Rock Placement Schedule

Product		2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	TOTAL kt
		Year												
Dolomite (Limestone)	kt	42,655	43,179	15,183	10,015	0	0	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	0	0	**122,005**
Ultramafic	kt	0	861	7,941	5,524	5,667	5,732	4,382	3,026	2,297	229	0	0	**35,659**
TOTAL	**kt**	**42,655**	**45,784**	**44,014**	**35,979**	**41,379**	**30,192**	**14,166**	**7,970**	**6,128**	**428**	**0**	**0**	**268,695**

Source: adapted from Wardrop, 2009b

Table 2.12-2 Design Basis for Rock Dumps

Item	Value
Life of the Open Pit mine	10 years
Total Waste Rock	268,696,000 t
Total Dolomite Waste Rock	111,032,000 t
Total Country Rock Waste Rock	122,005,000 t
Country Rock Waste Rock Specific Gravity	2.07 t/m³
Dolomite Waste Rock Specific Gravity	2.79 t/m³
Swelling	30%
Total Required Volume for Country Rock Waste Rock Dump	~ 59,000,000 m³
Total Required Dolomite for Construction of Mine Infrastructure (TWRMF, roads, dykes, etc.)	10,743,600 m³
Total Required Volume for Dolomite Waste Rock Dump	41,000,000m³

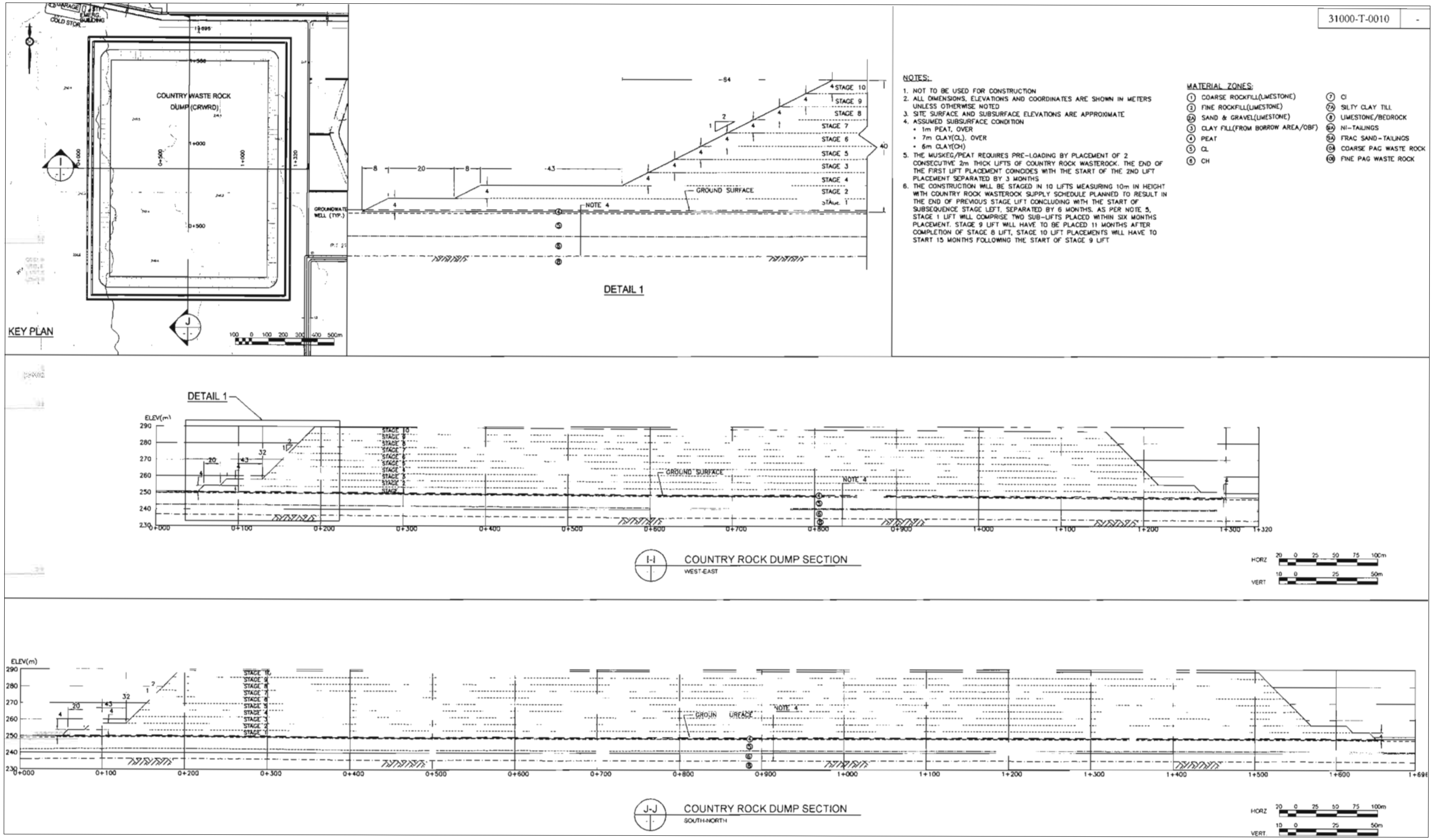
Source: Wardrop, 2009b

Table 2.12-3 Basic Engineering Design Parameters for Rock Dumps

Item	Target
1. Geotechnical Slope Stability: • Waste Dump	
• Construction (in stages)	• Static F.O.S 1.3, pseudo static F.O.S 1.05
• Normal Operation	• Same as above
• Closure	• Static F.O.S. 1.3, pseudo static F.O.S 1.05
2. Seismicity: • Operating Design Basis Earthquake	• 1: 475 year return
• Closure Earthquake	• 1: 2,475 year return
3. Max Dump Height	• Dependent on the results of engineering analyses in support of staged construction.

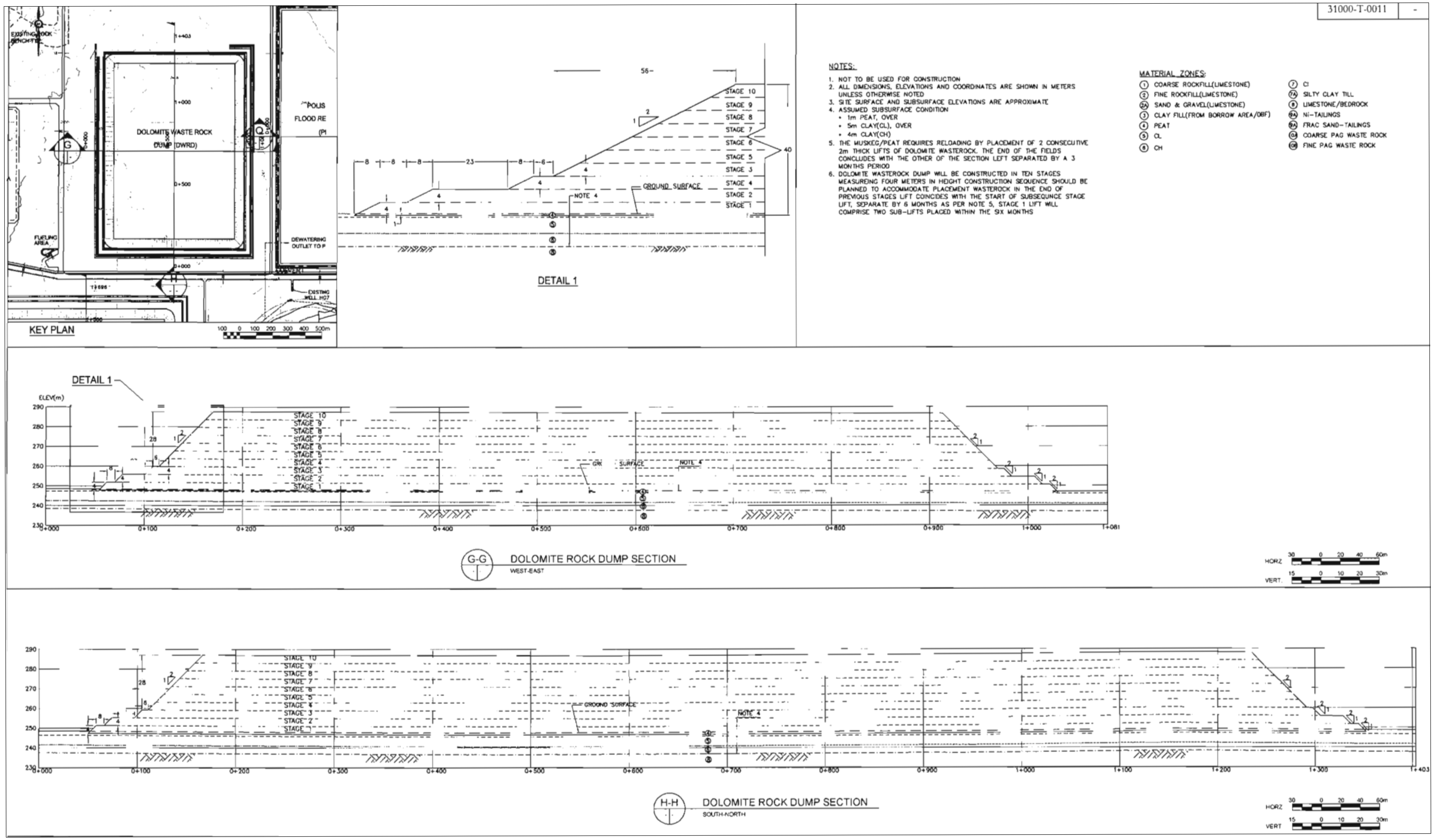
Source: Wardrop, 2009b

Plan and sectional details of the waste rock dumps are shown in Figures 2.12-1 and 2.12-2.



Source: adapted from Wardrop's drawing 0951330400-T0010 (Wardrop, 2009b)

Figure 2.12-1 Country Rock Waste Rock Dump Plan and Sections



Source: adapted from Wardrop's drawing 0951330400-T0011 (Wardrop, 2009b)

Figure 2.12-2 Dolomite Waste Rock Dump (DWRD) Plan and Sections

2.12.2.1 Country Rock Waste Rock Dump (CRWRD)

The Country Rock Waste Rock Dump (CRWRD) is designed for storing 59 Mm^3 of inert granitic waste rock. The dump will be founded on existing overburden comprised of muskeg/peat and clay averaging approximately 15 m in thickness. This dump will measure 1,596 m by 1,240 m in plan and will be staged in ten (10) lifts of 4 m for an ultimate dump height of 40 m. The dump configuration includes a 20 m and a 43 m setback for the toes of the Stage 2 and Stage 3 lifts with subsequent lifts set-back to give a 2H:1V slope (Wardrop, 2009b).

To allow for sufficient time for consolidation of the soft clay layer, successive lifts of this waste rock dumps will be sequenced with sufficient time for consolidation. Assuming 4 m lifts and a repetitive placement operation, any subsequent lift may only be started after the current lift has been in place for sufficient time for consolidation to be effective. Stages 2 to 8 may be sequenced 6 months after the previous stage, Stage 9, 11 months after that and Stage 10 after 15 months.

Construction of the Country Rock WRD will commence with the grubbing of all trees.

2.12.2.2 Dolomite Waste Rock Dump (DWRD)

The Dolomitic Waste Rock Dump is designed for storing 41 Mm^3 of inert dolomite rock. This dump will be founded on existing overburden comprised of muskeg/peat and clay averaging approximately 10 m in thickness. The dump will measure 1,303 m by 974 m in plan and will be staged in ten (10) lifts for a maximum height of 40 m. The dump configuration will be formed with overall slopes of 2H:1V and setbacks of 8 m, 23 m and 6 m for the toes of Stage 2, Stage 3 and Stage 4 lifts, respectively (Wardrop, 2009b).
Successive lifts of this dump will be sequenced with a set period of time (as will be done for the Country Rock WRD) to allow for sufficient time for consolidation of the soft clay layer underlying the dump. Assuming 4 m lifts and a repetitive placement operation, all subsequent lifts may only be started after a consolidation period of 6 months (Wardrop, 2009b).

Construction of the Dolomite WRD will commence with the grubbing of all trees.

2.12.2.3 Stability Analyses for the Waste Rock Dumps

Stability and settlement analyses were carried out in support of developing dump design sections that satisfy the design criteria (Table 2.12-2). Coupled analyses using Sigma/W and Slope/W, components of GeoStudio 2007, were used in the dam stability and settlement analyses. Sigma/W uses finite element methods to solve both stress-deformation and seepage dissipation equations simultaneously. Pore water pressures generated during lift placement were calculated with Sigma/W and then incorporated into Slope/W for stability analysis. Slope/W was used to locate failures with the least factor of safety within defined search limits (Wardrop, 2009b).

The Country Rock WRD and Dolomite WRD were modelled as underlain by 15 and 10 m of overburden, respectively. In the modelling, the overburden was divided into peat, and, upper (CI) and lower (CH) clay horizons. Both clay horizons were modeled using the non-linear Modified Cam-Clay (MCC) constitutive relationship (Wardrop, 2009b).

Initial pore pressure conditions were defined with an initial water table at the ground surface in the peat material. Zero pressure boundary conditions were applied to the bottom of the bedrock to model dewatering wells pumping water out of the bedrock layer. The duration between placement of each lift was assumed to be 6 months (Wardrop, 2009b). However, the Stage 9 and Stage 10 lifts of the Dolomite WRD were assumed to have a longer time interval between the placement of successive lifts. The time interval was assumed to be 11 and 15 months for the Stage 9 and the Stage 10 lifts, respectively. In the modelling for lifts 1 through 8, each lift was assumed to be placed on the first day, and then 182 days were allowed for consolidation prior to the placement of the next lift.

The stability analyses are representative of conditions immediately after placement of each lift (Wardrop, 2009b).

Pseudo static analysis was performed to simulate an earthquake condition of 0.03 g (Wardrop, 2009b).

Material Properties

Material properties for soft clays (CL and CH) and bedrock properties were based on laboratory data; whereas peat and waste rock material properties were based on professional judgement and previous experience (Wardrop, 2009b). Table 2.12-4 and Table 2.12-5 present the material properties used for the waste rock dump stability analyses in Sigma/W and Slope/W models, respectively.

2.12.2.3.1 Results of Stability Analyses for the Waste Rock Dumps

Table 2.12-6 presents results of the stability analyses. These results satisfy the minimum factor of safety requirements for static and pseudo static conditions, except for the short times following completion of some lifts in the Country Rock WRD, shown bolded numbers in Table 2.12-6. For these cases, the lower factors of safety are considered acceptable, because of their very short duration and their relatively fast increase beyond the specified factor of safety (Wardrop, 2009b). For the Country Rock WRD, lifts 9 and 10 will reach a factor of safety of 1.3 after 11 and 15 months of placement of the last lift, respectively. Detailed slope stability results for Country Rock WRD and Dolomite WRD are presented elsewhere (Wardrop, 2009b).

Table 2.12-4 Assumed Sigma/W Material Properties for the Waste Rock Dump Stability Analyses

Materials	Material Category	Material Model	Poisson's Ratio	Young's Modulus (kPa)	Hydraulic Conductivity (cm/s)
Waste Rock	Effective Drained Parameters	Linear Elastic	0.35	70,000	-
Peat	Effective Parameters w/PWP Change	Linear Elastic	0.35	2,000	1.00E-01
Soft Clay (CL)	Effective Parameters w/PWP Change	Soft Clay (MCC)	0.36	-	1.36E-08
Soft Clay (CH)	Effective Parameters w/PWP Change	Soft Clay (MCC)	0.37	-	4.97E-09
Bedrock	Effective Parameters w/PWP Change	Linear Elastic	0.49	100,000	6.89E-04

Source, Wardrop, 2009b

Note: PWP Porewater pressure.

Table 2.12-5 Assumed Slope/W Material Properties for the Waste Rock Dump Stability Analyses

Materials	Model	Unit Weight (kN/m^3)	Cohesion (kPa)	Phi (º)
Waste Rock	Mohr-Coulomb	20	0	40
Peat	Mohr-Coulomb	13	18	0
Soft Clay (CL)	Mohr-Coulomb	21	20	29
Soft Clay (CH)	Mohr-Coulomb	18	10	25
Bedrock	Bedrock (Impenetrable)			

Source, Wardrop, 2009b

Table 2.12-6 Slope Stability Results

Lift No.	Country Rock Waste Rock Dump (CRWRD)			Dolomite Waste Rock Dump (DWRD)		
	Static (10 day) Required/ Computed	**Static (6 months) Required/ Computed**	**Pseudo static (6 months) Required/ Computed**	**Static (10 day) Required/ Computed**	**Static (6 months) Required/ Computed**	**Pseudo static (6 months) Required/ Computed**
1	1.30/**1.15**	1.30/1.69	1.05/1.53	1.30/1.90	1.30/2.04	1.05/1.87
2	1.30/**1.28**	1.30/1.46	1.05/1.20	1.30/1.34	1.30/1.33	1.05/1.18
3	1.30/1.67	1.30/1.93	1.05/1.45	1.30/1.37	1.30/1.31	1.05/1.20
4	1.30/1.75	1.30/1.89	1.05/1.47	1.30/1.37	1.30/1.46	1.05/1.23
5	1.30/1.77	1.30/1.75	1.05/1.46	1.30/1.36	1.30/1.45	1.05/1.24
6	1.30/1.53	1.30/1.58	1.05/1.36	1.30/1.37	1.30/1.46	1.05/1.26
7	1.30/1.35	1.30/1.38	1.05/1.31	1.30/1.38	1.30/1.44	1.05/1.27
8	1.30/**1.26**	1.30/1.32	1.05/1.22	1.30/1.39	1.30/1.44	1.05/1.28
9	1.30/**1.22**	1.30/1.30*	1.051.20*	1.30/1.40	1.30/1.45	1.05/1.29
10	1.30/**1.23**	1.30/1.30**	1.05/1.18**	1.30/1.40	1.30/1.44	1.05/1.29

Source: adapted from Wardrop, 2009b

Notes: * 11 months after lift placement.
** 15 months after lift placement.

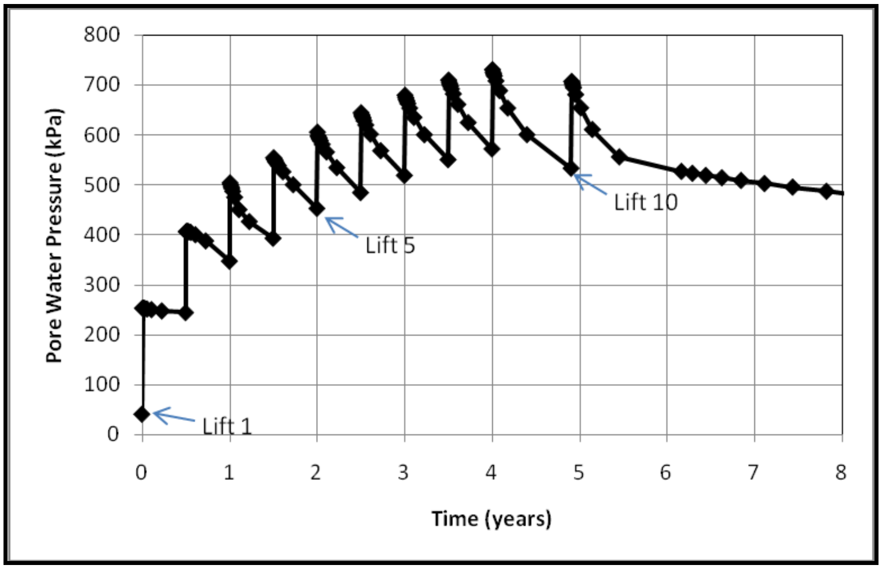
In order to achieve design heights of 40 m, the configuration of the dumps must include setbacks as summarized in Table 2.12-7 (Wasrdrop, 2009b).

Table 2.12-7 Required Setbacks for the Waste Rock Dumps

Lift No.	Country Rock Waste Rock Dump Setback (m)	Dolomite Waste Rock Dump Setback (m)
Stage 1	20	8
Stage 2	43	23
Stage 3	0	6

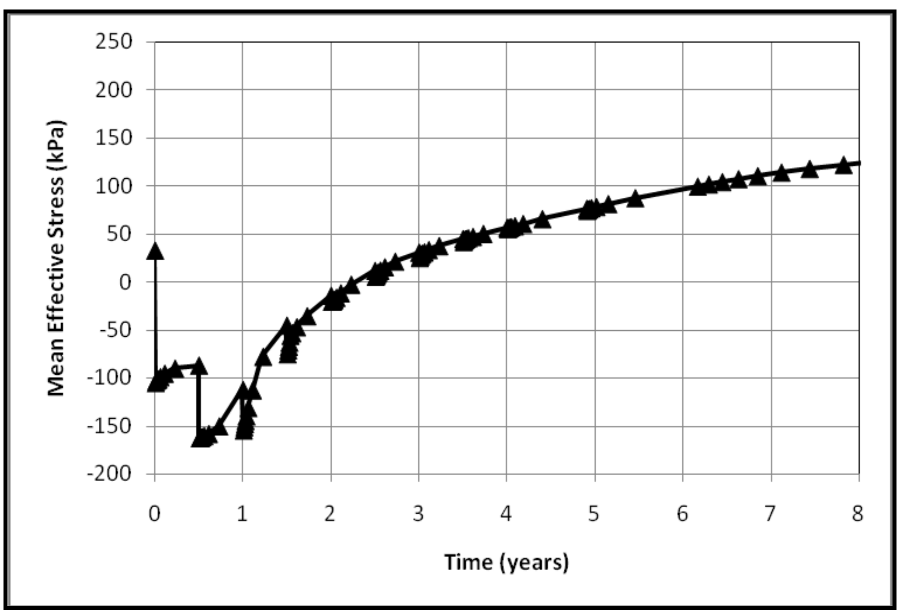
Source: Wardrop, 2009b

Figure 2.12-3 through Figure 2.12-10 show the effective stress versus time, and pore water pressure versus time for the short- and long-term conditions as computed in the foundation soils underneath the Dolomite WRD and Country Rock WRD. Figures 2.12-3, 2.12-5, 2.12-7 and 2.12-9 illustrate the effective stress increases after placement of each lift and their stabilization over time. Figures 2.12-4, 2.12-6, 2.12-8 and 2.12-10 show the pore water pressure generation after placing each lift and its dissipation over time. The estimated period for the pore water pressures to dissipate are 31 years for the Country Rock WRD and 16 years for the Dolomite WRD (Wardrop, 2009b).



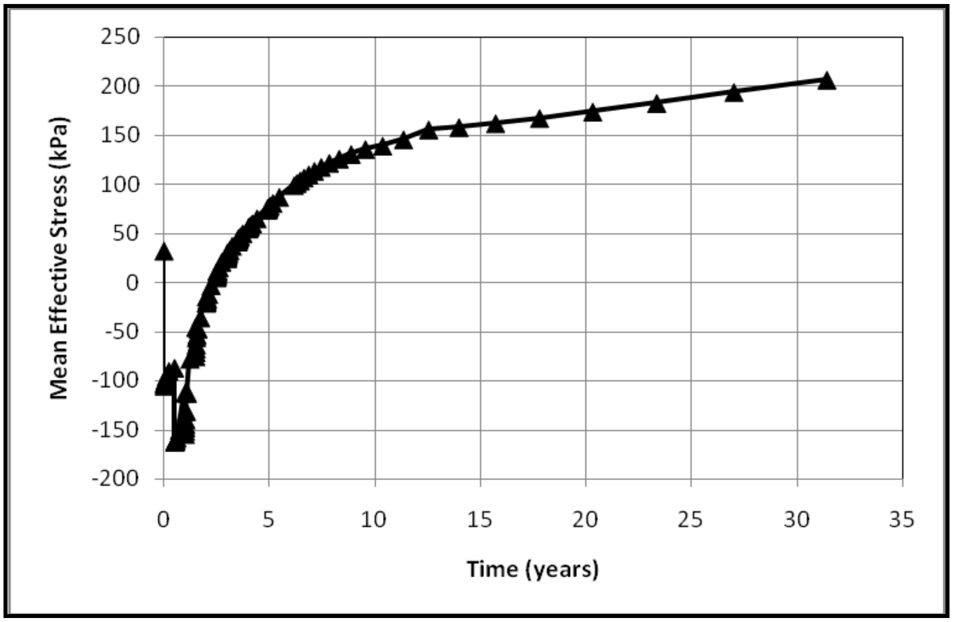
Source: Wardrop, 2009b

Figure 2.12-3 Short-term Mean Effective Stress versus Time for the Country Rock WRD



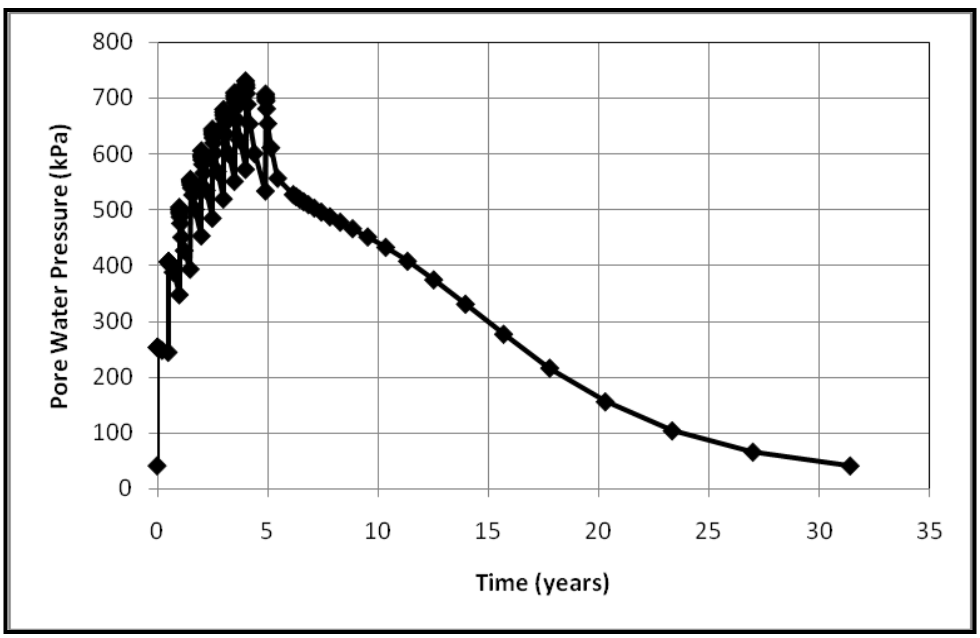
Source: Wardrop, 2009b

Figure 2.12-4 Short-term Pore Water Pressure versus Time for the Country Rock WRD



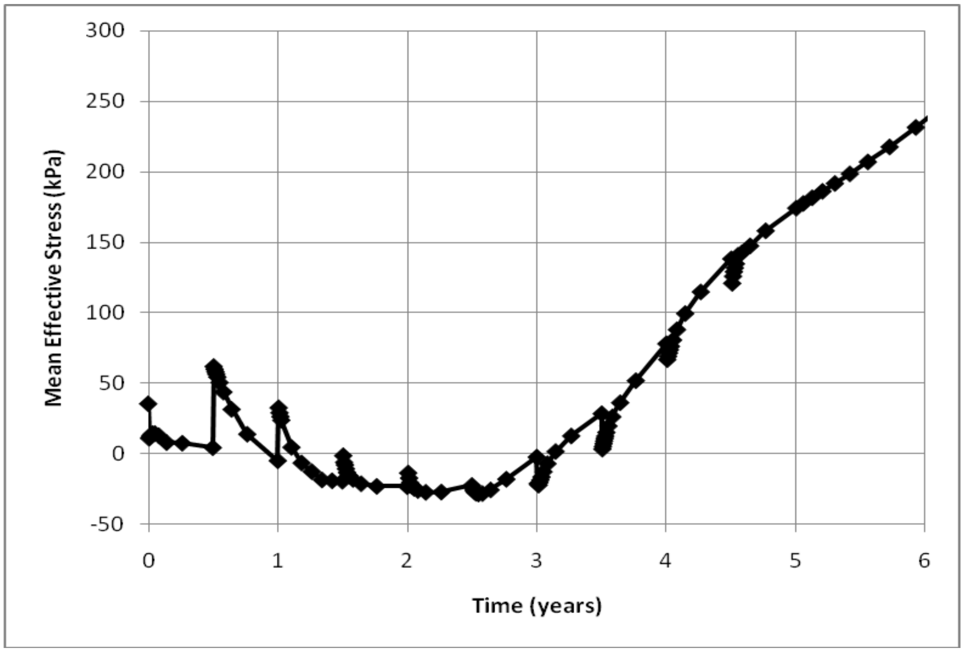
Source: Wardrop, 2009b

Figure 2.12-5 Long-term Mean Effective Stress versus Time for the Country Rock WRD



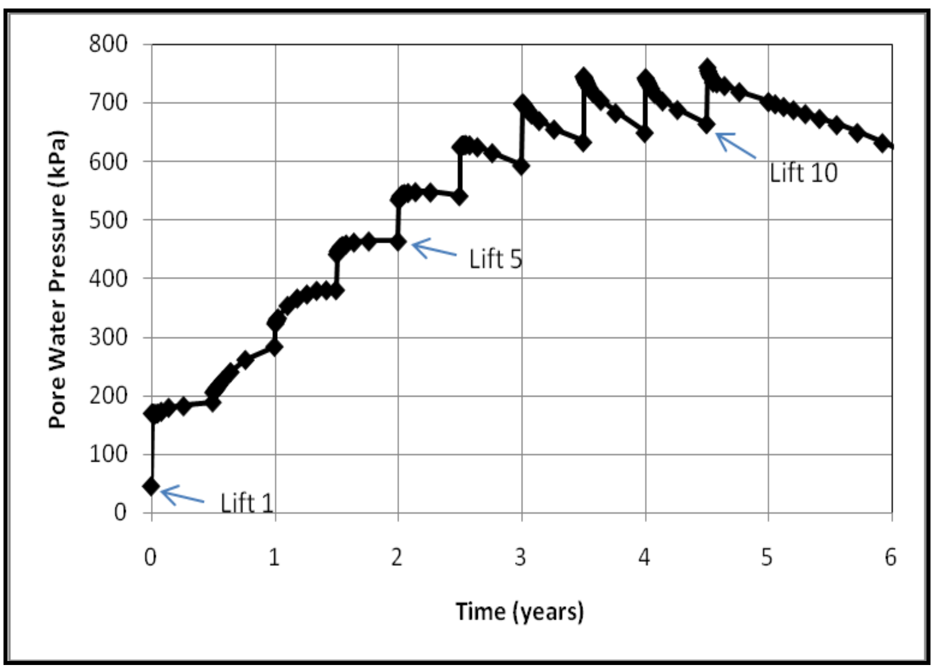
Source: Wardrop, 2009b

Figure 2.12-6 Long-term Pre Water Pressure versus Time for the Country Rock WRD



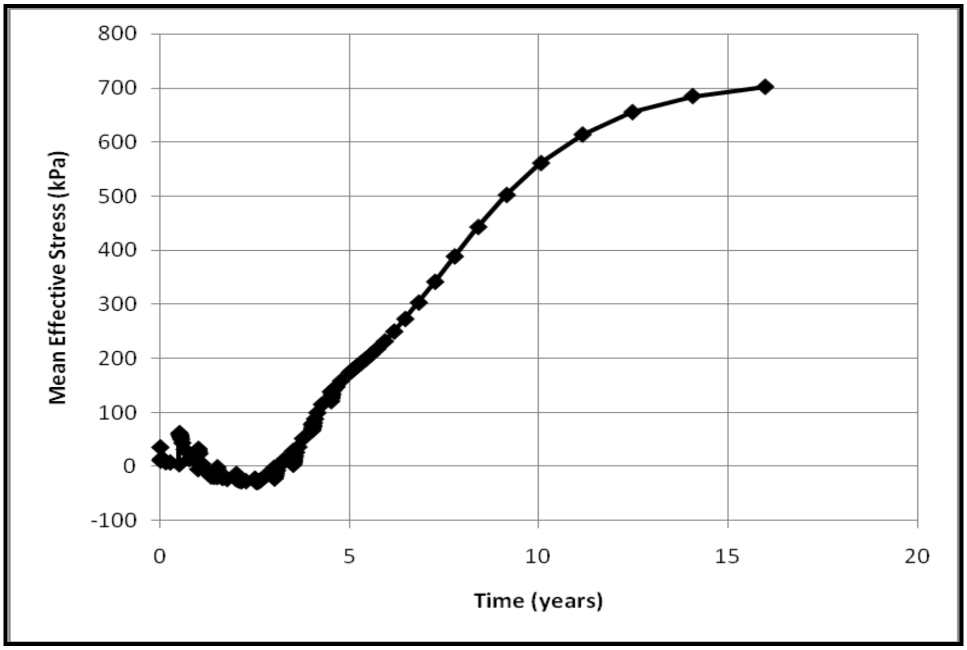
Source: Wardrop, 2009b

Figure 2.12-7 Short-term Mean Effective Stress versus Time for the Dolomite WRD



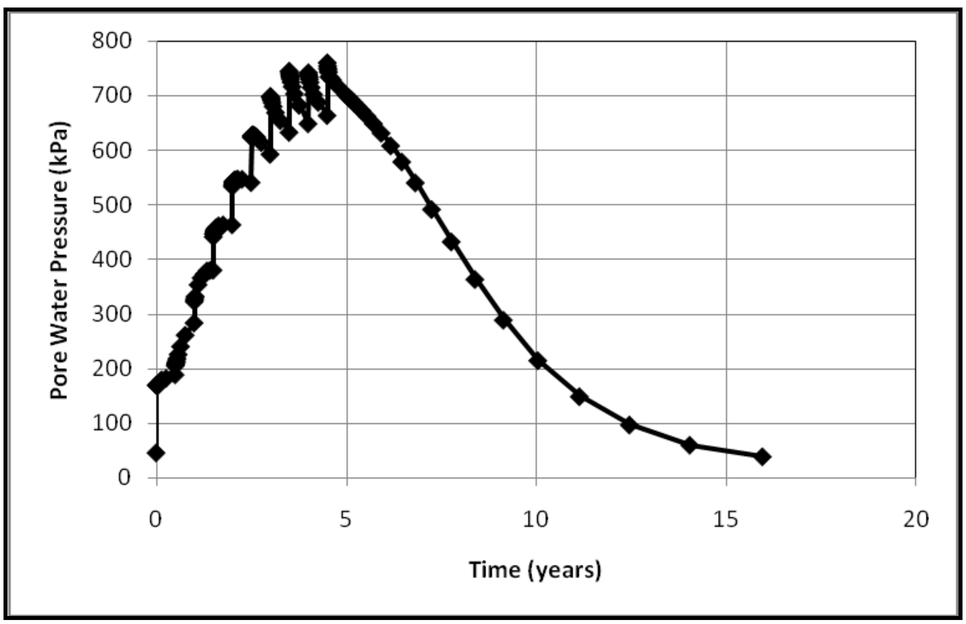
Source: Wardrop, 2009b

Figure 2.12-8 Short-term Pore Water Pressure versus Time for the Dolomite WRD



Source: Wardrop, 2009b

Figure 2.12-9 Long-term Mean Effective Stress versus Time for the Dolomite WRD



Source: Wardrop, 2009b

Figure 2.12-10 Long-term Pre Water Pressure versus Time for the Dolomite WRD

2.12.3 Deposition Strategy for Waste Rock Dumps

The main construction issue in relation to the dumps is foundation preparation by pre-loading. This will be achieved by placing 2 consecutive 2 m thick waste rock lifts as a part of the Stage 1 lift. The start of the second lift will have to coincide with the end of the first lift placement, separated by 3 months (Wardrop, 2009b). The second lift will have to be completed by the end of 6 months. Spreading of waste rock will be progressive over the entire dump area in advance of the Stage 2 lift placement (Wardrop, 2009b).

From a construction standpoint, it is preferable to proceed with the preloading during the winter season. It is estimated that the preloading will need to remain in place for at least 90 days (~3 months). This estimate can be confirmed by test fills during the detailed design stage. The placement of the Stage 2 lift in both dumps should proceed by slow gradual advancement of another 4 m of waste rock over larger areas to promote finalization of consolidation of the muskeg and peat and gradual load transfer into underlying clays in accordance with the staged construction (Wardrop, 2009b).

2.13 Tailings and Ultramafic Waste Rock Management Facility and Polishing Pond

The Tailings and Ultramafic Waste Rock Management Facility (TWRMF) is a key component of the water and waste management system at Minago for tailings, liquid waste and ultramafic waste rock. The disposal of tailings and waste rock has been studied from a number of different perspectives. The selected alternative is tailings co-disposal with ultramafic waste rock behind a lined rockfill embankment dam. Muskeg and/or clay will be forming the base of the embanked repository. The remaining waste rock will be disposed of in the Dolomite Waste Rock Dump, if it is dolomite/limestone, or in the Country Rock Waste Rock Dump otherwise (Figure 2.1-2).

The TWRMF location within the project area (Figure 2.1-2) was selected to take into account factors such as the exclusion zones, the distance from the open pit and the favourable subsurface conditions, including shallow soft clay overburden (Wardrop, 2009b).

One key objective for the co-disposal is to initially induce invasion of tailings into the voids of end-dumped PAG/ML waste rock to encapsulate the PAG waste rock in tailings for the ultimate goal of providing acceptable seepage water quality from the facility. Other key objectives are to facilitate closure without long-term water treatment and to significantly lower CAPEX/OPEX and closure cost (Wardrop, 2009b).

Material in the TWRMF will be stored subaqueously whenever possible. Subaqueous disposal is practiced at many metal mines to keep oxidative rates at a minimum and to minimize metal leaching. Based on geochemical work done to date, Minago's mill tailings contain low sulphide levels and were deemed to be non acid generating (NAG) (URS, 2009i). Sulphide levels were less than or equal to 0.07 % in the Master tailings samples tested. However, ultramafic waste rock has been found to be potentially acid generating (PAG) (URS, 2009i).

The TWRMF will remain in place after all operations have ceased at the site. The TWRMF inflow will consist of:

1) mill tailings;
2) tailings and liquid waste from the Frac Sand Plant;
3) outflow from the sewage treatment system;
4) sludge from the potable water treatment plant; and
5) precipitation.

Outflows from the TWRMF include the TWRMF Decant, losses due to evaporation and sublimation, and seepage. Seepage will be captured by interceptor ditches surrounding the TWRMF and will be pumped back to the TWRMF. The seepage design criteria has tentatively been set at 250 m^3/day to satisfy walk-away requirements (Wardrop, 2009b). The TWRMF Decant will be discharged to the Polishing Pond (Figure 2.1-2) and will be regulated automatically by a control system.

2.13.1 TWRMF Design Criteria

The TWRMF design requires compliance with permitting requirements as well as dam design and water quality guidelines. The TWRMF dam design is controlled to a significant extent by the presence of weak peat and clay foundation soils and a sufficient separation of the dam from Highway 6. The TWRMF must accommodate a total of 27.4 Mt of nickel and frac sand tailings and 36 Mt PAG-waste rock over the course of 9 years and provide secure storage for the long-term.

The Design Basis and Basic Engineering Design Parameters are summarized in Tables 2.13-1 and 2.13-2, respectively. Additional Design Criteria for the TWRMF are as follows (Wardrop, 2009b):

- The rate for the construction of successive stages of the TWRMF Dam should be governed by foundation strength and consolidation characteristics as well as the mine waste production schedule.
- The cone of depression created by pit dewatering is predicted to extend laterally in the dolomite to a distance of approximately 5,000 m to 6,000 m from the proposed open pit. The cone of depression will provide under drainage for the overburden clays and should be considered in geotechnical analyses for the TWRMF dam.
- A designated decant pond should be located between the causeways.
- The tailings deposition plan should ensure minimal exposure of PAG waste rock to atmospheric conditions during operations, closure and post closure.
- The configuration of PAG waste rock within the facility should allow for 2 m tailings cover at the end of the tailings deposition.
- Based on experience, tailings deposition slopes of 0.5% sub-aerial and 2% subaqueous should be assumed in the design.

2.13.2 Deposition Plan for the TWRMF

Construction of the TWRMF dam will take place in 2011 and 2012. Concurrently disposed tailings and ultramafic waste rock will be fully contained behind a perimeter dam to be constructed as a part of a robust operation. Key elements of the concurrent disposal of tailings and ultramafic waste rock in the TWRMF are illustrated in Figure 2.13-1 and the deposition strategy is briefly described in the following paragraphs (Wardrop, 2009b):

- In order for the frac sand deposition to start and subsequently to support the initial phase of Ni-tailings deposition in 2014, a dolomite waste rock base will be constructed where the coarse PAG-waste rock rind will be placed and underneath the north and south causeways. The construction of the dolomite waste rock base will be completed during the last stages of the TWRMF dam construction in 2012.

Table 2.13-1 Design Basis for the TWRMF

Item	Value
Life of TWRMF	9 years
Total Nickel Tailings (tonnes)	24,847,889
Total Sand Tailings (tonnes)	2,571,804
Total Combined Tailings to TWRMF (tonnes)	27,419,693
Total PAG Waste Rock (tonnes)	35,660,000
Tailings Specific Gravity (Nickel)	2.6
Initial Tailings Void Ratio (Nickel)	1.0
Initial Tailings Density (Nickel)	1.3 t/m^3
Average Final Tailings Density (Nickel)	1.5 t/m^3
Tailings Pulp Density (solid weight) (Nickel)[1]	45%
Water in Tailings Voids (Nickel)	22%
Average Initial Tailings Density (Sand)	1.4 t/m^3
Average Final Tailings Density (Sand)	1.6 t/m^3
Tailings Pulp Density (solid weight) (Sand)	20%
Ultramafic Waste Specific Gravity	2.59
Ultramafic Waste Swelling	30%
Void Space in PAG Waste Rock	4,130,502 m^3
Void Space in Coarse PAG Waste Rock	3,304,402 m^3
Void Space in Fine PAG Waste Rock	826,100 m^3
Total Volume of Ni Tailings	16,565,259 m^3
Total Volume of Sand Tailings	1,607,378 m^3
Total Combined Tailings Volume	18,172,637 m^3
Total PAG Waste Rock (solids and voids)	17,898,842 m^3
Total Ni-Tailings Ingress into Voids of Coarse Ultramafic Waste Rock (at initial tailings density)[2]	2,478,301 m^3
Total Ni- and Frac-Sand Tailings ingress into Voids of Fine Ultramafic Waste Rock (at initial tailings density)[3]	413,050 m^3
Total Ni-Tailings Between the Ultramafic Waste Rock Rind and Central Causeway (at final tailings density)	15,376,725 m^3
Required TWRMF Storage	33,275,567 m^3
Required TWRMF Storage (with 15% contingency included)	38,300,000 m^3

Source: adapted from Wardrop, 2009b

NOTES:

1. A 45% solids density is used in the feasibility study water balance. However, higher water-to-solids ratios to enhance transport into and through the rock fill may be considered in the detailed engineering.

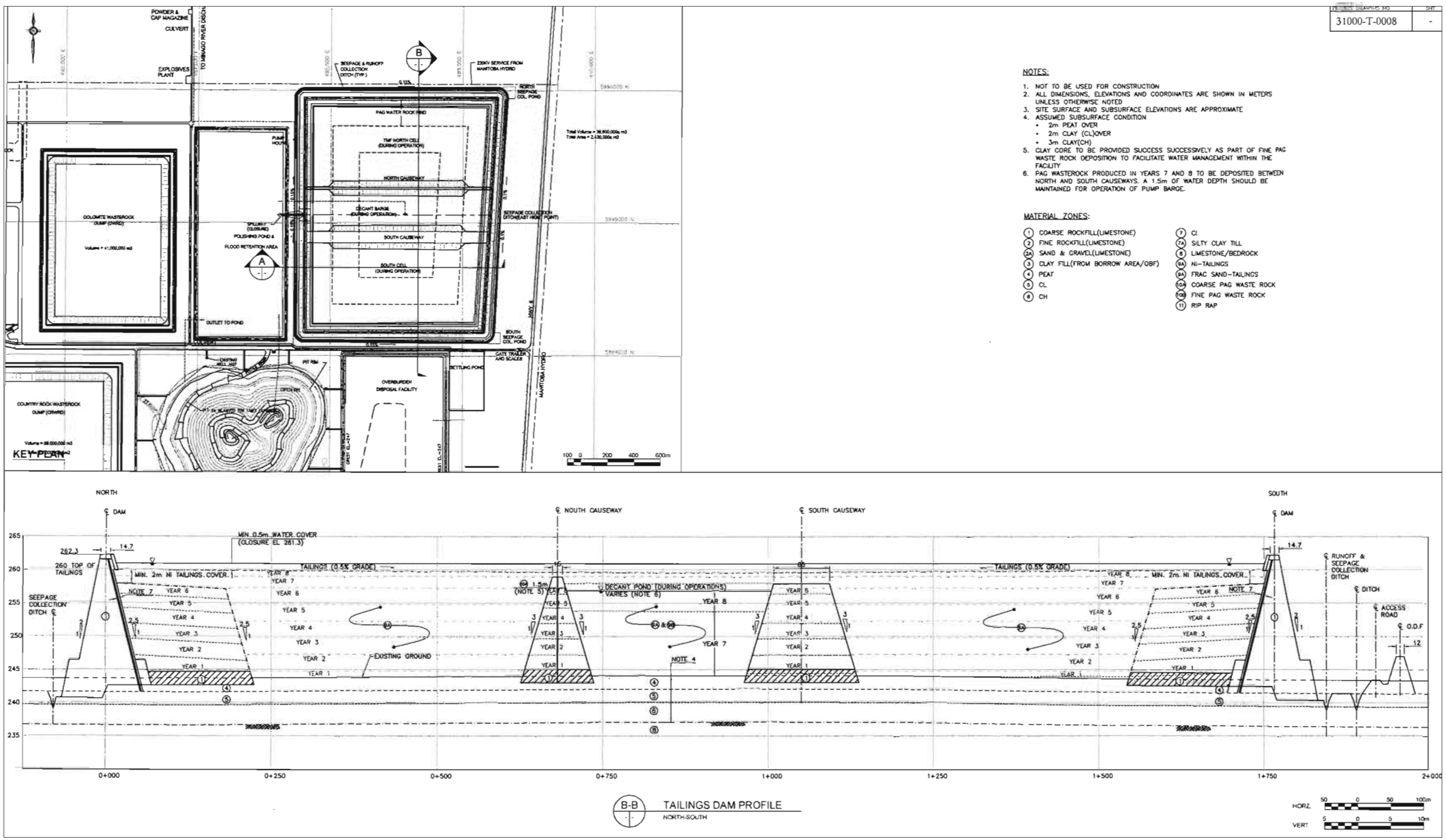
2. Coarse ultramafic waste rock, represented by fractions larger than 0.2 m, is estimated to be 80% of total ultramafic waste rock. Infilling of voids within coarse ultramafic waste rock with tailings is estimated to be 75%. Ingressed tailings were assumed to remain at their initial density due to the relative incompressibility of the waste rock matrix.

3. Fine ultramafic waste, represented by fractions finer than 0.2 m, is estimated at 20% of total ultramafic waste. Infilling of voids within fine ultramafic waste rock with tailings is estimated to be on the order of 50%. Ingressed tailings are assumed to remain at their initial density due to the relative incompressibility of the waste rock matrix.

Table 2.13-2 Basic Engineering Design Parameters for the TWRMF

Item	Target	Comments
1. Geotechnical Slope Stability		
• Construction (in stages)	• Static F.O.S. 1.3, pseudo static F.O.S 1.05.	
• Normal Operating	• Same as above.	
• Closure	• Static F.O.S. 1.5, pseudo static F.O.S 1.05.	
2. Seepage	• Limit on Contaminants of Concern (CoC) concentrations	• Analyses using SEEP/W targeting a total estimated seepage volume less than 250 m^3/day. • Low permeability barrier to be provided on the upstream face of the containmant structure to reduce seepage through the ultramafic waste rock – tailing composite. • Seepage from the TWRMF to be collected via collection ditches and ponds.
3. Hydrotechnical • Construction Diversion Peak Flow	• 1:20 yr - 24 hr rainfall	• All peak flows are estimated from catchment times of concentration and storm. Seepage to be collected via collection ditches reporting to the overall water management system.
• Operation peak flow	• 1:200 yr – 24 hr rainfall	
• Closure Spillway and Diversion peak flow	• 1:1,000 yr – 24 hr rainfall	• Determine wave run-up in the freeboard.
• Freeboard	• 1.0 m on the top of Closure Spillway wet section for 1:200 year runoff	
• Closure Flood	• 1:1,000 yr – 24 hr rainfall	
• Runoff Coefficient	• 1	
4. Decant System (if applicable) • Water Storage	• Minimum five days retention or 1.5 m of water level at all times, whichever is higher	
5. Closure Cover	• A minimum of 0.5 m of water on the top of final tailings at the containment structure.at all times.	• Runoff (dry year), seepage, infiltration and evaporation to ensure a minimum thickness water cover.
6. Seismicity • Operating Design Basis Earthquake	• 1: 475 year return	
• Closure Earthquake	• 1:2,475 year return	

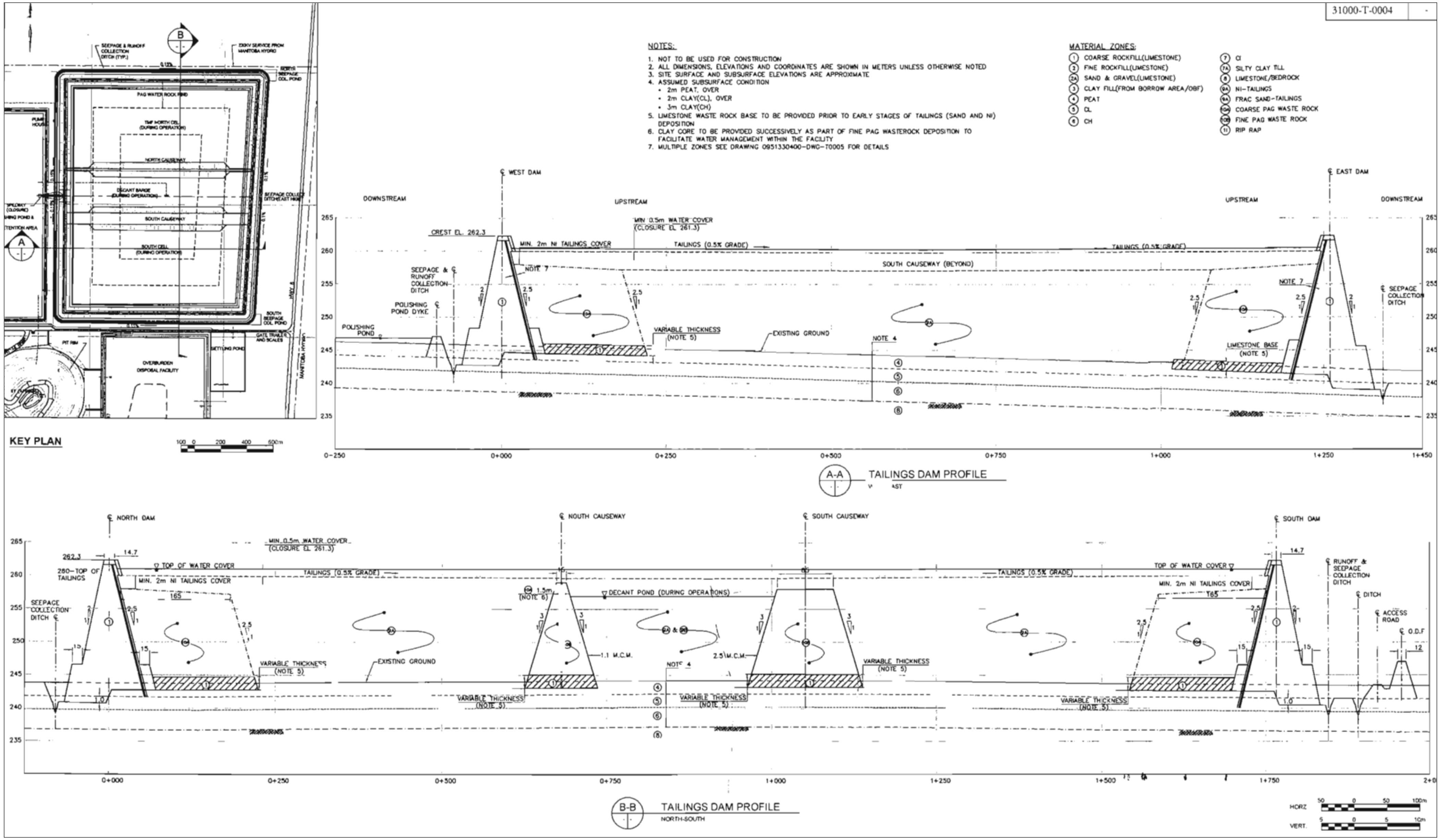
Source: Wardrop, 2009b



Source: adapted from Wardrop's drawing 0951330400-T0008 (Wardrop, 2009b)

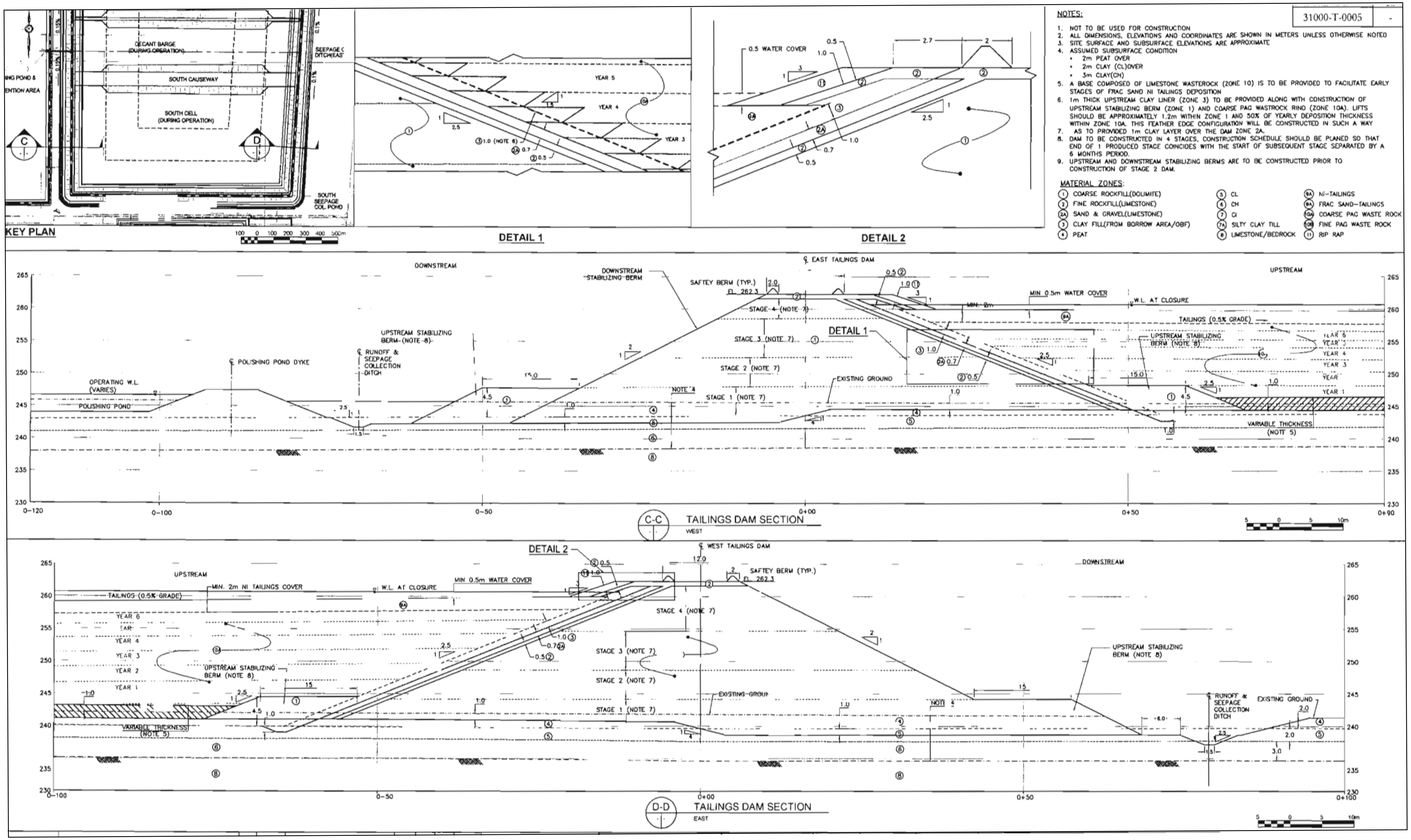
Figure 2.13-1 Deposition Plan and Profiles of theTailings and Ultramafic Waste Rock Management Facility

- The retaining structure construction will be carried out in lifts corresponding to yearly ultramafic waste rock production. A 1 m clay liner will be provided between the rind and the upstream face of the dam as depicted in Figures 2.13-2 and 2.13-3. The clay liner in between the waste rock rind and the dam will ensure full containment by minimizing seepage reporting to the downstream environment as per design criteria.
- The clay cutoff trench within the north causeway will facilitate intermittent flooding and dewatering in both of the cells (north and south cells). Maximizing PAG waste rock saturation during waste rock placement will minimize oxidation and reduce their ARD/ML potential.
- The coarse ultramafic waste rock (estimated at 80% of total PAG-waste rock production) will be deposited in a rind to be constructed immediately upstream of the dam. The rind construction will be carried out in lifts corresponding to the yearly PAG waste rock production.
- The fine ultramafic waste rock (estimated at 20% of total PAG-waste rock production) will be deposited in the north and south causeways. The north causeway will have a clay cutoff trench built in stages, also in accordance with the yearly waste rock production schedule.
- Ultramafic waste rock will be placed simultaneously in both the northern and southern cells and flooding will closely follow advancement of the ultramafic waste rock placement. Dewatering of the north cell will take place prior to the start of tailings deposition in order promote hydraulic gradients and thereby increase invasion of Ni-tailings into the void space of the ultramafic waste rock. Tailings deposition in the northern cell will take approximately 6 months. Dewatering of the southern cell will precede Ni-tailings placement and it will take another 6 months to complete the deposition in the southern cell.
- Ni-tailings deposition will be carried out from the dam crest by running feeder pipes from the main tailings supply pipe at the dam crest and down the upstream dam slope. The feeder pipes will eventually distribute tailings over the rockfill through perforated spreader pipes. Ripping of the uppermost PAG-waste rock surface might be done as an expedient to open up the uppermost fines in order to promote tailings ingress into the waste rock void space.
- Ultramafic waste rock placement and tailings deposition will alternate in the same fashion for 6 years. During this time, a decant pond will be created between the north and south causeways from which water will be pumped to the Polishing Pond (Figure 2.1-2). In 2019 and 2020, coarse ultramafic waste rock will be deposited in the area between the north and south causeways. A minimum of 1.5 m of decant water above the waste rock will be maintained to facilitate free flow and prevent potential blockage during operations of barge mounted pumps. An alternative arrangement may be pumping from perforated decant towers installed within the rockfill placed in causeways. This alternative will be examined more closely in the detailed design stage.



Source: adapted from Wardrop's drawing 0951330400-DWG-T0004 (Wardrop, 2009b)

Figure 2.13-2 Tailings and Ultramafic Waste Rock Management Facility (TWRMF) Dam Plan and Profile



Source: adapted from Wardrop's drawing 0951330400-DWG-T0005 (Wardrop, 2009b)

Figure 2.13-3 Tailings and Ultramafic Waste Rock Management Facility (TWRMF) Dam Plan and Sections

- In 2019 and 2020, the ultramafic waste rock in the rind will receive a minimum of 2 m of Ni-tainings cover by peripheral discharging from the dam crest. It is estimated that the slopes for the tailings beach and the subaqueous tailings will be 0.3% and 2%, respectively.
- The frac sand tailings deposition will be carried out from the top of causeways until the Ni-tailings deposition ceases. Thereafter, sand tailings will also be deposited from the dam crest through the main tailings supply pipe system.
- After Ni-tailings deposition will have ceased, frac sand tailings will be deposited as a final layer on top of the Ni-tailings. Frac sand tailings will be produced approximately 2 years longer than the Ni-tailings. Frac sand tailings have low metal concentrations and will leave the top surface of the TWRMF in an inert condition. On top of the Frac sand tailings, a minimum of 0.5 m of water cover will be provided on closure.

Concurrent disposal of tailings and ultramafic waste rock will ensure total encapsulation of PAG-waste rock on closure and the water cover will ensure subaqueous disposal, both of which will minimize ARD/ML concerns.

Decant from the TWRMF will be discharged to the Polishing Pond to address concerns regarding the resuspension of tailings due to wind and wave action on the water cover. Suspended solids will settle out in the Polishing Pond prior to water discharge from that facility to the receiving environment.

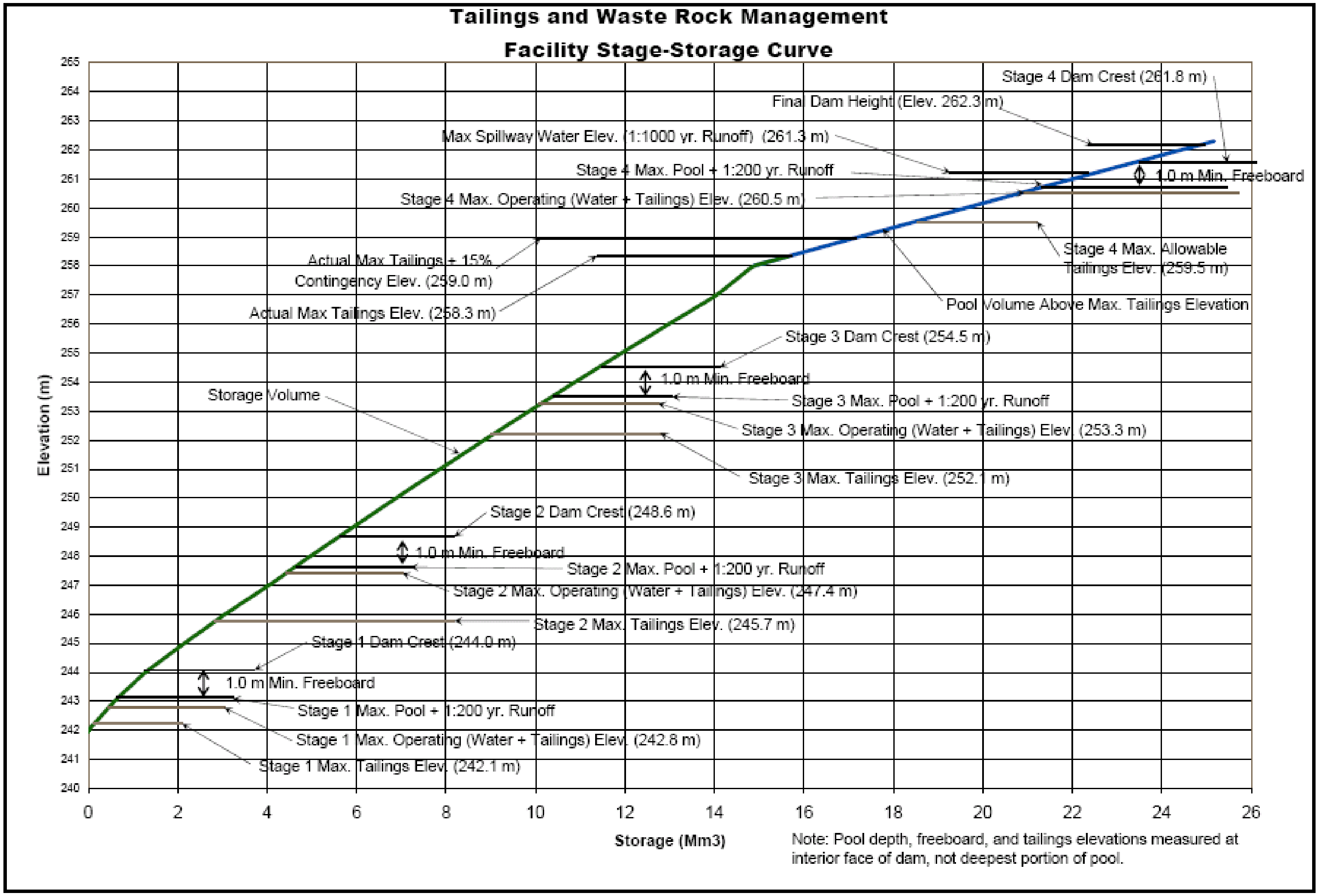
Figure 2.13-4 shows stage storage curve with critical design elevations for the TWRMF based on estimates given in Minago's Feasibility Study (Wardrop, 2009b).

2.13.3 TWRMF Dam Options and Selections

2.13.4 TWRMF Dam Section Design

The dam will be located in an area where the geotechnical profile lends itself to a higher containment structure with a small footprint. Also, the geotechnical profile will allow for construction staging to meet the mine production schedule. To bound uncertainties related to extrapolation of confirmed shallow overburden characteristics in the southeastern part of the TWRMF, a deeper overburden was assumed to underlie the rest of the TWRMF in the geotechnical analyses considered in this report. Final confirmation of the TWRMF foundation will be part of a detailed design geotechnical investigation. The plan and profile of the TWRMF is shown in Figure 2.13-2 with typical dam sections illustrated in Figure 2.13-3.

The TWRMF dam was designed as an earth/rockfill structure varying in settled height from approximately 19 m to 21 m above the local topography. Peat will be left in place within the upstream part of the dam foundation and removed along with a 1.0 m of soft underlying clay within



Source: Wardrop, 2009b

Figure 2.13-4 TWRMF Stage Storage Curve

the downstream part. The upstream and downstream dam slopes of the rockfill dam will be 2.5H:1V and 2H:1V, respectively (Wardrop, 2009b).

Based on stability analyses, the dam will be constructed in four (4) stages to meet the consolidation requirements. The construction schedule will be planned so that the end of previous stage coincides with the start of the subsequent stage. The heights of dam fill will be up to 4.5 m and 6 m for Stages 1 and 2 and Stages 3 and 4, respectively. Stabilizing berms (4.5 m high and 15 m wide downstream and upstream) will be required prior to the start of the Stage 2 lift (Figure 2.13-3).

The construction of the dam will take two years from the start in “2012” (Year -1) to completion at the end of “2013” (Year +1). The dam shell will be constructed of coarse rockfill (Zone 1 material) comprising an estimated 800 mm minus dolomite waste rock originating from the open pit (Figure 2.13-3). The upstream side of this zone will support a 0.5 m thick zone of fine rockfill (Zone 2 material) comprised of minus 75 mm dolomite waste rock and finally a 0.5 m sand and gravel zone (Zone 2A material). The dam will have an upstream clay lining with a nominal thickness of 1 m placed over the Zone 2A in four sequences as shown in Figure 2.13-3 and briefly described below.

Sequence 1: The clay liner will extend through peat to be keyed in the native clay. The clay liner (Zone 3) will be provided in a feather-edge like gap between the top of Zone 2A on the dam upstream slope and Zone 1 within the upstream stabilizing berm. This will coincide with the completion of about 1.2 m thick lifts within the upstream stabilizing berm constructed ahead of the start of the Stage 2 lift within the main dam structure (Wardrop, 2009b).

Sequence 2: The clay liner (Zone 3) will be provided in a feather-edge like gap as depicted on Detail 1 between the top of Zone 2A on the dam upstream slope and Zone 10A within PAG-waste rockfill rind. This will coincide with the completion of about 50% of yearly lift thicknesses within Zone 10A (PAG-waste rock rind) (Wardrop, 2009b).

Sequence 3: A 1.0 m clay (Zone 3) liner above the PAG rockfill rind will be placed over Zone 2A at the dam upstream slope ahead of tailings discharge (Wardrop, 2009b).

Sequence 4: Extention of the clay (Zone 3) liner to the dam crest will be placed ahead of the water cover implementation. The thickness of the Zone 3 in this last stage will increase as dictated by a 3H:1V upstream slope. The clay liner in this uppermost zone will be protected with a 0.5 m thick fine rockfill (Zone 2), which in turn will be covered by a 1.0 m of rip rap (Zone 11) to protect the dam crest from the ice scour action. In initial stages, material for Zone 3 will be obtained from local borrow pits containing stiff clays. Subsequently, Zone 3 material may be obtained from the ODF, if it meets design specifications (Wardrop, 2009b).

A 0.3 m thick pavement surface composed of Zone 2 material will be provided on the dam crest. Appropriate safety berms composed of Zone 2 material will also be provided on the crest (Wardrop, 2009b).

2.13.5 Dam Stability and Settlement Analyses

Dam stability and settlement analyses were carried out in support of developing a dam design section that satisfies design criteria outlined in Tables 2.13-1 and 2.13-2.

Methods of Analysis

Coupled analyses using Sigma/W and Slope/W, components of GeoStudio 2007, were used in the dam stability and settlement analyses. Sigma/W uses finite element methods to solve both stress-deformation and seepage dissipation equations simultaneously. Pore water pressures generated during lift placement were calculated with Sigma/W and then incorporated into Slope/W for stability analysis. Slope/W was used to locate failures with the least factor of safety within defined search limits (Wardrop, 2009b).

In the modelling, initial pore pressure conditions were specified with an initial water table at the ground surface. Zero pressure boundary conditions were applied to the bottom of the bedrock to model dewatering wells pumping water out of the bedrock layer.

Sigma/W modelling of the dam’s section assumed that the first lift will be placed on the first day, and that 6 months will pass thereafter for consolidation. Slope stability analyses were performed assuming that 10 days had passed since the lift had been placed and at the end of 182 days. All four lifts were modeled assuming no waste rock or tailings had been placed on the upstream side of the TWRMF until construction was completed (Wardrop, 2009b).

Another analysis was performed that simulated conditions six months after the completion of the facility, assuming that the waste rock and tailings had been placed at the same time. The total computed construction time of the facility was assumed to be 2 years (Wardrop, 2009b).

A small buttress with a height of 4.5 m and a width of 15 m was incorporated into the design on both the upstream and the downstream sides of the TWRMF. Construction of these buttresses was assumed to coincide with the time of placement of the Stage 2 lift to enhance stability (Wardrop, 2009b).

Pseudo static analyses were completed to simulate earthquake conditions using 0.03 g (50% of the Peak Ground Acceleration (PGA) for a 1:2,475-year return period), which is consistent with generally accepted practices adopted by the United States Army Corps of Engineers (Hynes-Griffin and Franklin, 1984).

Assumed Material Properties

Assumed material properties for the foundation materials (CL, CH and bedrock) were based on field and laboratory data. The properties of the waste rock (dolomite and PAG/ML (ultramafic), coarse and fine rockfill material were estimated based on previous experience and professional judgement (Wardrop, 2009b). Tables 2.13-3 and 2.13-4 show material properties used in Sigma/W and Slope/W, respectively.

Results of Dam Stability and Settlement Analyses

Table 2.13-5 presents results of the slope stability analyses after placement of each lift assuming that the TWRMF is filled with PAG waste rock and tailings. Except for a very short time following the completion of the Stage 1 lift (see bolded and underlined number in Table 2.13-5), the slope stability results show that the TWRMF dam satisfies the minimum requirements for static and pseudo static conditions during operations and at closure. Because of a very short duration and relatively fast increase beyond the specified factor of safety, this is considered acceptable. Detailed slope stability results are presented elsewhere (Wardrop, 2009b).

Table 2.13-3 Sigma/W Input Material Properties

Materials	Material Category	Material Model	Poisson's Ratio	Young's Modulus (kPa)	Hydraulic Conductivity (cm/s)
Dolomite Waste Rock	Effective Parameters w/PWP Change	Linear Elastic	0.35	50,000	1.00E-01
Ultramafic Waste Rock	Effective Parameters w/PWP Change	Linear Elastic	0.35	50,000	1.00E-01
Coarse Rockfill	Effective Drained Parameters	Linear Elastic	0.33	50,000	-
Fine Rockfill	Effective Drained Parameters	Linear Elastic	0.33	7,000	-
Sand and Gravel	Effective Drained Parameters	Linear Elastic	0.35	8,000	-
Peat	Effective Parameters w/PWP Change	Linear Elastic	0.35	2,000	1.00E-01
Soft Clay (CL)	Effective Parameters w/PWP Change	Soft Clay (MCC)	0.36	-	1.36E-08
Soft Clay (CH)	Effective Parameters w/PWP Change	Soft Clay (MCC)	0.37	-	4.97E-09
Bedrock	Effective Parameters w/PWP Change	Linear Elastic	0.49	100,000	6.89E-04

Source: Wardrop, 2009b

Note:

PWP Porewater pressure.

Table 2.13-4 Slope/W Input Material Properties

Materials	Model	Unit Weight (kN/m^3)	Cohesion (kPa)	Phi (º)
Dolomite Waste Rock	Mohr-Coulomb	18	0	40
Ultramafic Waste Rock	Mohr-Coulomb	18	0	40
Coarse Rockfill	Mohr-Coulomb	19	0	40
Fine Rockfill	Mohr-Coulomb	22	0	38
Sand and Gravel	Mohr-Coulomb	22	0	35
Peat	Mohr-Coulomb	13	18	0
Soft Clay (CL)	Mohr-Coulomb	21	20	29
Soft Clay (CH)	Mohr-Coulomb	18	10	25
Bedrock	Bedrock (Impenetrable)			

Source: Wardrop, 2009b

Table 2.13-5 Slope Stability Results for the TWRMF

Case	Time (days[1])	Downstream F.O.S.		Upstream F.O.S.	
		Static Required/ Computed	Pseudo static Required/Computed	Static Required/ Computed	Pseudo static Required/Computed
Lift 1	10	1.3/**<u>1.11</u>**	-	1.31.48	-
	182	1.3/1.59	1.05/1.46	1.3/1.56	1.05/1.42
Lift 2	192	1.3/1.32	-	1.3/1.49	-
	364	1.3/1.77	1.05/1.58	1.3/1.59	1.05/1.38
Lift 3	374	1.3/1.60	-	1.3/1.52	-
	546	1.3/1.65	1.05/1.46	1.3/1.58	1.05/1.39
Lift 4	556	1.3/1.51	-	1.3/1.65	-
	728	1.3/1.56	1.05/1.40	1.3/1.69	1.05/1.52
Full of Tailings and towards Closure	738	1.5/1.46	-	-	-
	910	1.5/1.55	1.05/1.37	-	-
	2,577	1.5/1.94	1.05/1.75	-	-

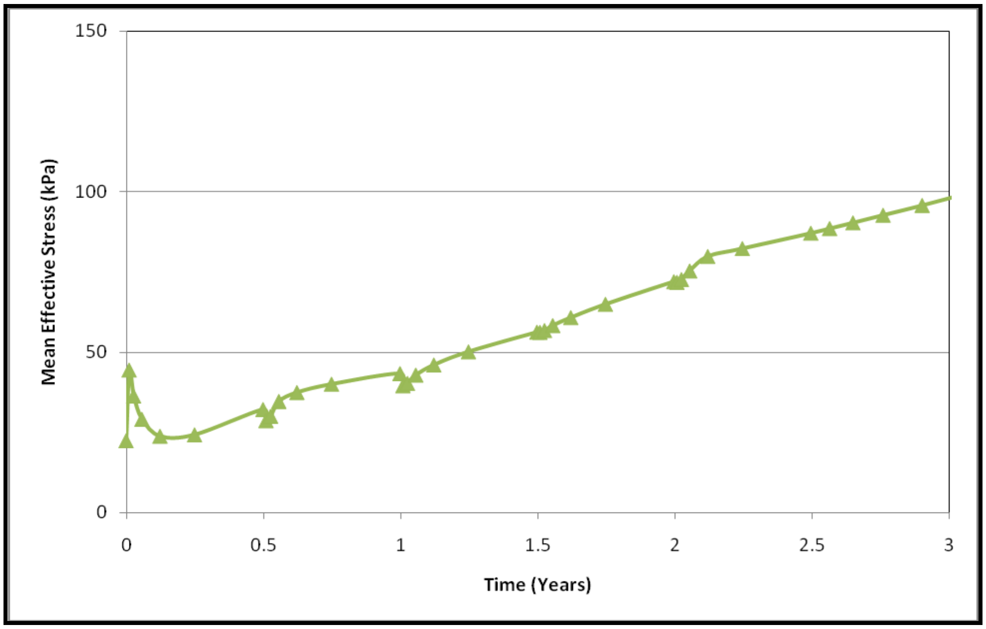
Source: Wardrop, 2009b

Note: 1 After placement of Stage 1 lift

Figure 2.13-5 through Figure 2.13-8 show the effective stress versus time, and pore water pressure versus time for the short- and long-term as computed in the foundation soils below the centerline of the dam. Figure 2.13-5 and Figure 2.13-7 illustrate how the effective stress will increase after placement of each lift and then stabilize at a later time. Figure 2.13-6 and Figure

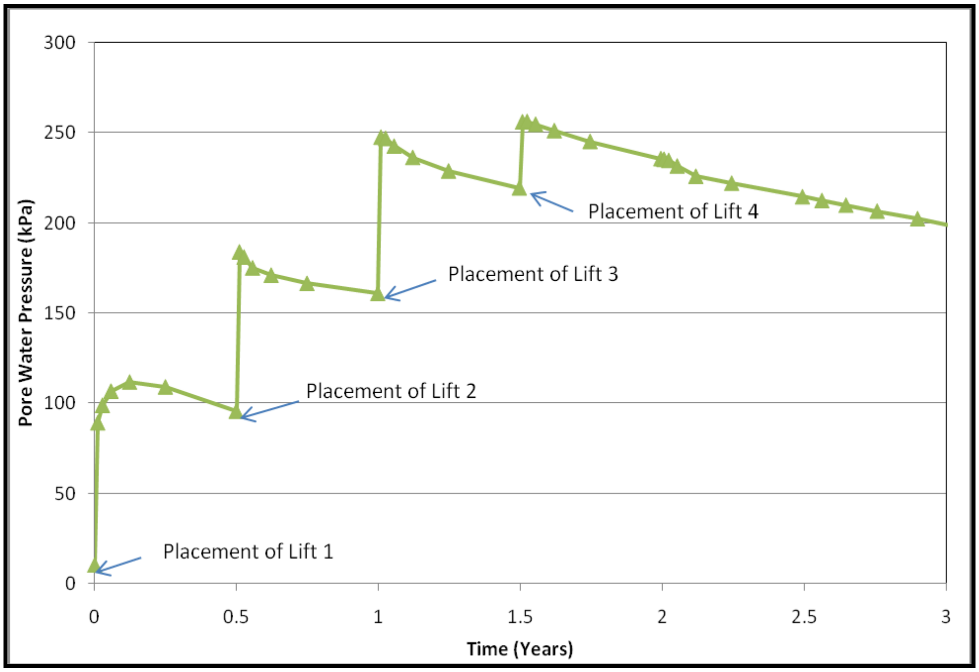
2.13-8 show estimates of pore water pressure build up after placement of each lift for dam construction for Stages 1 through 4 and its corresponding dissipation over time. The pore water pressures will dissipate in approximately 12.5 years (Wardrop, 2009b).

Figure 2.13-9 shows the settlement along the base of the TWRMF with time. The total settlement for the facility was estimated to be approximately 1 m (Wardrop, 2009b).



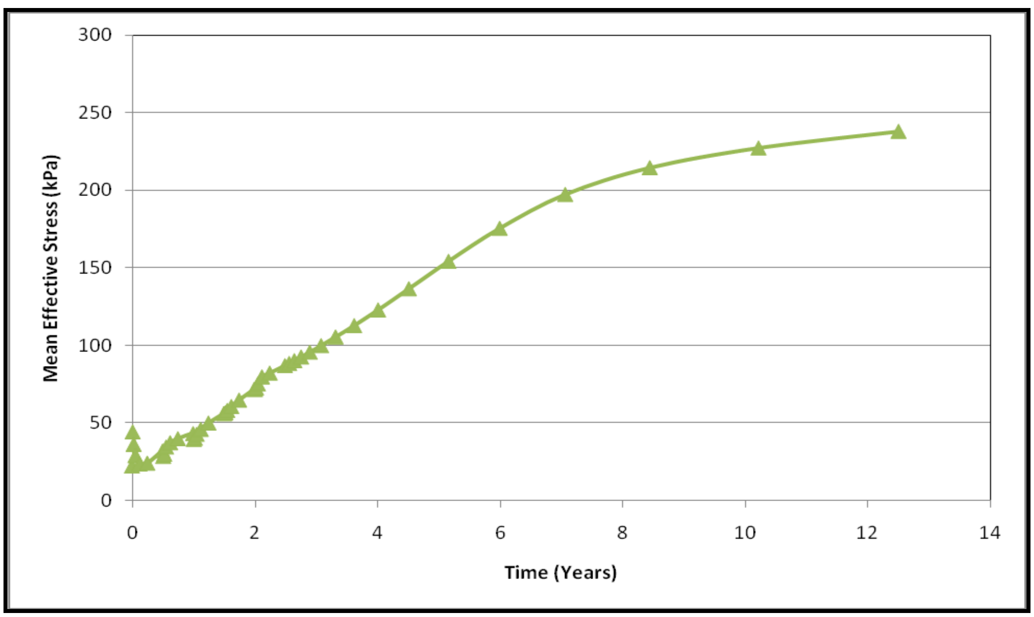
Source: Wardrop, 2009b

Figure 2.13-5 Short-term Mean Effective Stress versus Time for the TWRMF



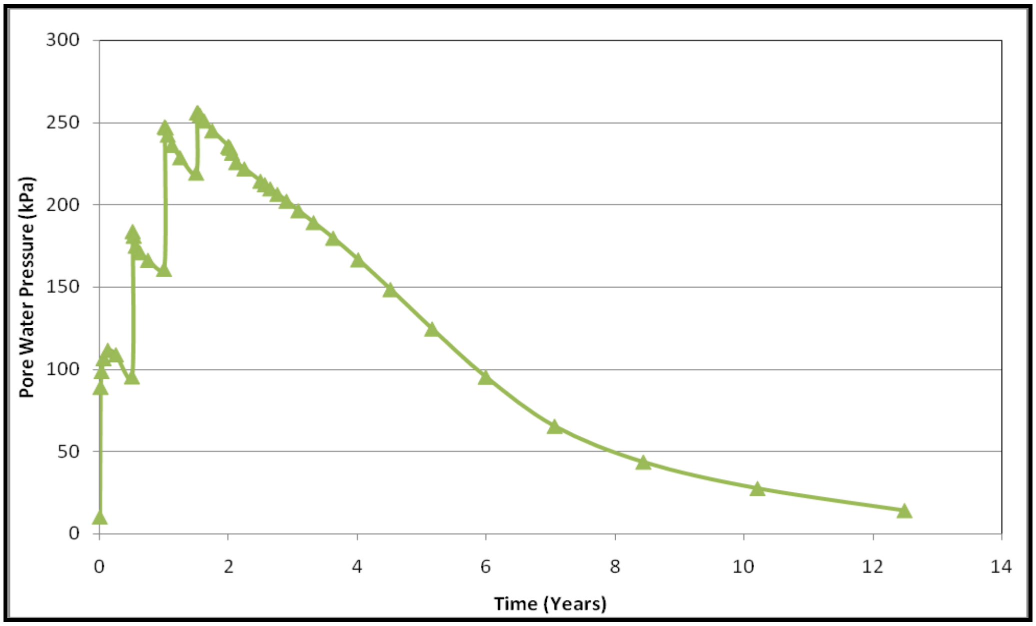
Source: Wardrop, 2009b

Figure 2.13-6 Short-term Pre Water Pressure versus Time for the TWRMF



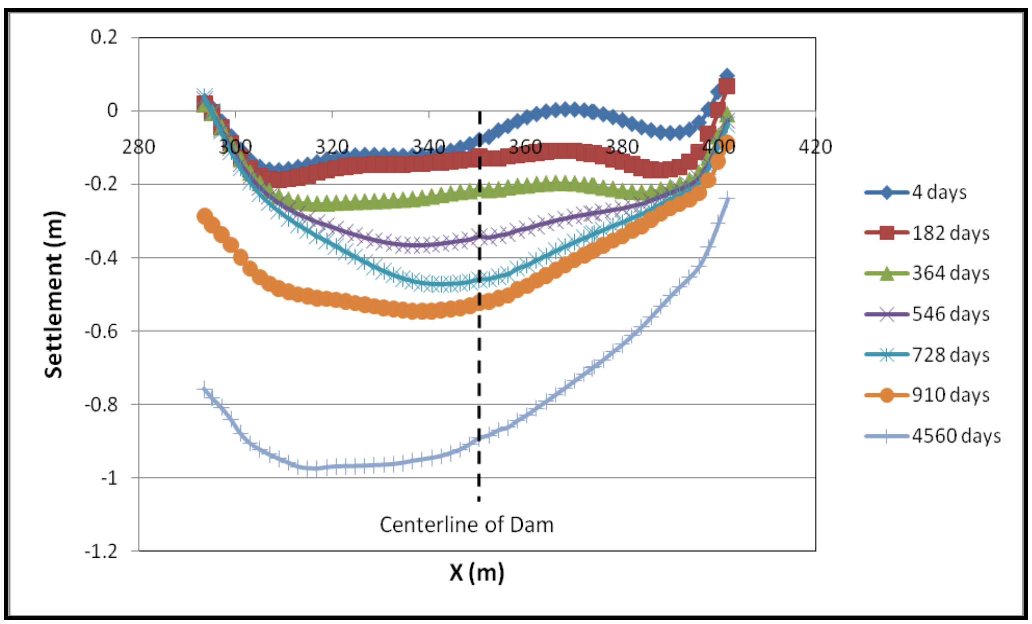
Source: Wardrop, 2009b

Figure 2.13-7 Long-term Mean Effective Stress versus Time for the TWRMF



Source: Wardrop, 2009b

Figure 2.13-8 Long-term Pre Water Pressure versus Time for TWRMF



Source: Wardrop, 2009b

Figure 2.13-9 Settlement along the Base of the TWRMF Dam

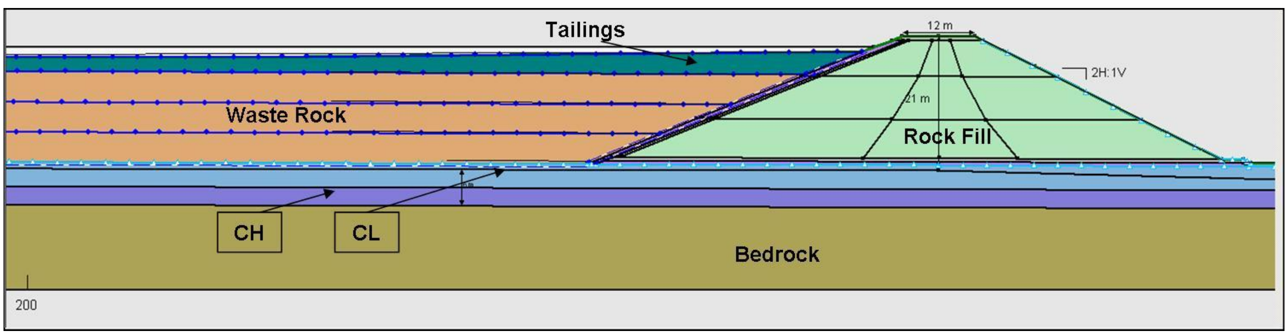
2.13.6 Seepage Analyses

Seepage analyses were critical in determining potential advantages of PAG waste rock encapsulation by Ni-tailings at closure, especially with respect to seepage water quality.

In order to develop a methodology for co-disposal of tailings and waste rock and work backwards to develop a final configuration of the containment structure, the following two scenarios were modelled (Wardrop, 2009b):

1. Dam structure comprising a gravel filter zone (Zone 2A) between the rockfill shell and the combined mass of PAG/ML waste rock and Ni-tailings. This scenario relies on attenuation of the seepage flux through the combined mass of PAG/ML waste rock and Ni-tailings.
2. Dam structure allowing for full containment of the combined mass of PAG/ML waste rock and Ni-tailings by placing clay at a variable thickness (Zone 3) over the sand and gravel filter (Zone 2A). This scenario uses the low permeability barrier to achieve better seepage control than Scenario 1.

Figure 2.13-10 illustrates the configuration of the TWRMF that was used in the seepage modeling.



Source: Wardrop, 2009b

Figure 2.13-10 Tailings Storage Facility Layout used in the Seepage Modelling

Seepage Model for the TWRMF

The seepage modeling was completed using SEEP/W (GEO-SLOPE, 2007), a two dimensional finite-element model. The modeling, which was completed in steady state, was limited to the closure conditions of the facility. The available field and laboratory data were used to estimate the hydraulic conductivity values for tailings, native clays and bedrock. The hydraulic conductivities for TWRMF dam zones and combined PAG/ML waste rock were estimated based on previous experience and professional judgement (Wardrop, 2009b). Table 2.13-6 presents a summary of the anticipated material properties and the model parameters assigned to simulate them.

Table 2.13-6 Material Properties assumed for the TWRMF Seepage Model

Material Type	Estimated Hydraulic Conductivity (m/sec)	Used in Model		
		Parameter Name	Saturated Hydraulic Conductivity (m/sec)	Saturated Volumetric Water Content (m^3/m^3)
Soft Clay (CL)	1e-10	CL	1.36e-10	0.385
Soft Clay (CH)	1e-11	CH	6.75e-11	0.3
Bedrock	1e-5 to 1e-6	Bedrock	6.90e-6	0.3
Fine Rockfill	1e-7	Fine Sand	4.30e-6	0.35
Coarse Rockfill	1e-4 to 1e-5	Uniform Sand	1.00e-5	0.3
Tailings	2e-7 to 8.2e-8	Sandy Silty Clay	1.40e-7	0.41
Clay	1e-10	CL/Well Graded High Clay	1.36e-10	0.35
Combined Waste rock and Tailings[1]	2e-7	Glacial Till (compacted)	1.00e-7	0.23

Source: adapted from Wardrop. 2009b

Note: 1 75% of void space assumed to be invaded by Ni-tailings as per design criteria.

Results of the TWRMF Seepage Analyses

The computed seepage volume reporting to the collection system immediately downstream of the 5 km long structure for Scenario 1 (leaky dam) was in the order of 2,920 m^3/day. The seepage rates for Scenario 2 were 250 m^3/day and 100 m^3/day for a 1 m and 2 m thick clay zone (Zone 3), respectively (Wardrop, 2009b).

It follows that a 1 m clay zone fulfills the seepage volume requirement towards meeting the water quality standards based on environmental concentrations and geochemistry of the seepage water. This was applied to the dam design section (Wardrop, 2009b).

2.13.7 Geotechnical Construction Considerations for the TWRMF

All peat/muskeg and the soft clay layer underneath the peat must be removed from the downstream part TWRMF dam foundation and the runoff/seepage collection ditch (Wardrop, 2009b). The muskeg/peat excavated from the downstream part of the TWRMF dam foundation will be disposed of in the Overburden Disposal Facility (ODF). The muskeg/peat removal will require prior excavation of a system of drainage ditches reporting to the collection ditch that will coincide with the future runoff/seepage collection ditch located immediately east of the eastern side of the future TWRMF dam.

The system of drainage ditches will excavated in the winter as the frozen top of the muskeg will facilitate movement of construction equipment. The rate/depth of frost penetration may also be accelerated by snow removal in the construction area (Wardrop, 2009b).

Preliminary rockfill gradation specifications for fine and coarse rockfill for the TWRMF dam are outlined below. Boundaries for rockfill and other filling materials in the TWRMF dam are illustrated in Figures 2.13-2 and 2.13-3.

2.13.7.1 Coarse Rockfill (Zone 1)

Dolomite waste rock from the open pit will be the source of coarse rockfill (Zone 1 material) for the construction of the TWRMF dam and dolomite rockfill base for the ultramafic waste rock rind and north and south causeways construction. Grading requirements for the coarse rock material are shown in Table 2.13-7.

2.13.7.2 Fine Rockfill (Zone 2)

Filter criteria were used to determine the rockfill (Zone 2 material) gradations presented in Table 2.13-8. The Zone 2 material will be obtained by primary and secondary crushing of Zone 1 dolomite waste rock.

Table 2.13-7 Gradation Requirements – Coarse Rockfill (Zone 1)

Dimension or U.S. Standard Sieve Size (mm)	% Passing by Weight
810	100
450	60-100
200	37-100
130	25-60
75	10-45
25	0-15
#4	0

Source: Wardrop, 2009b

2.13.7.3 Sand and Gravel (Zone 2A)

There are no known natural sources of sand and gravel within economic distances for the Minago project. Therefore, Zone 2A material may have to be obtained by further crushing some of the Zone 2 dolomite rockfill.

Table 2.13-8 Gradation Requirements – Fine Rockfill (Zone 2)

Dimension or U.S. Standard Sieve Size (mm)	% Passing by Weight
75	100
50	90-100
30	60-100
25	54-100
19	46-60
#4	10-22
#8	0-7
#16	0

Source: Wardrop, 2009b

2.13.7.4 Clay (Zone 3)

Clay (Zone 3 material) that will be used in the upstream TWRMF dam liner and in cut-off trenches will be initially obtained from local borrow sources within the uppermost “drier” clay. This may be replaced with clay deposited in the ODF, if it is of suitable quality.

2.13.8 TWRMF Associated Facilities

Runoff Diversion Berm

Surface water runoff will be diverted away from the TWRMF by the construction of a runoff diversion berm along its western and eastern sides. Diverting surface water will decrease the amount of water entering the system. The diversion berm will be constructed using peat and clay from the excavaton of the runoff collection system.

Runoff and Seepage Collection System

The runoff and seepage collection system will collect seepage and precipitation that falls on and near the TWRMF dam. Runoff will be collected in ditches built around the entire perimeter of the TWRMF and directed to two existing ponds, located to the northeast and southeast of the dam. Water reporting to the ponds will be pumped back to the TWRMF. Figure 2.13-2 shows the plan view of ditches and ponds that will make up the runoff and seepage collection system.

The western and eastern sides of the TWRMF will have ditch inverts sloped at 0.10%. The flow divide will be at the mid point of the western and eastern sides of the facility from where the water will be diverted north and south. The northern and southern ditches will slope at 0.15% and report to the southern and northern collection ponds (Wardrop, 2009b).

The base of eastern and western ditches will be 1.5 m wide and this will increase to 2.5 m for northern and southern ditches. Ditches will be have side slopes of 2.5H:1V and 4H:1V in native clays and peat, respectively. There will be a 0.5 m setback at the peat and clay interface. All of the ditches will be designed to have freeboard within peat without erosion protection for the design ditch invert slopes (Wardrop, 2009b).

2.13.9 Pertinent Precedents

Based on Wardrop's knowledge, there is no direct long-term precedent for a combined waste rock and tailings disposal for geographic and climatic conditions similar to Minago.

The importance of a lack of a directly-related precedent for the Co-Disposal Scheme involving PAG/ML mine waste rock and tailings in a single repository must be recognized. The handling of geochemical, environmental, and permitting issues associated with the co-disposal scheme has been developed through the incorporation of the combined experience from a variety of operations listed below (Wardrop, 2009b).

- Algoma Ore Properties, Wawa, ON (Tailings transport into unfiltered rock fill by through flow).
- Mines Gaspe Ltd., Murdochville, Que (Tailings transfer into unfiltered rock fill by static liquefaction).
- Vale Inco Limited, Sudbury, ON (Densification of tailings by blasting).
- Falcondo, Dominican Republic (Silt transport into voids of slag fill dam by through flow).
- Syncrude Canada, AB (Dredging experiments using tailings fines as the dredging fluid).
- Giant Mine, Yellowknife, NT (Tailings transport into unfiltered rock fill by through flow).

The post closure environmental considerations and costs for water treatment in perpetuity dictated the selection of co-disposal of PAG/ML waste rock and tailings in a single repository. Co-disposal of tailings and PAG-waste rock will fully contain them behind a perimeter dam to be constructed as a part of a robust operation.

2.13.10 TWRMF Dam Classification

Dam classification in accordance with the Canadian Dam Association Dam Safety Guidelines 2007 (CDA) is based on the evaluation of the consequences of dam failure in terms of risk to population, loss of life, and environmental, cultural, and economic losses. The TWRMF dam can be classified as "Significant Dam Class" and the selection of the hydrology, hydrotechnical and seismic design criteria presented in previous sections were selected in accordance with the CDA criteria considering the following:

- Dam is located in an unpopulated area of Manitoba, relatively far away from urban settlements.

- During the life of the mine, only personnel required for the operation of the mine will be temporarily resident near the mine.
- The temporary housing to accommodate the personnel of the mine and the infrastructure for the processing of the ore will be located at a distance of approximately 2 km from the TWRMF dam.
- Co-disposal of rockfill and tailings provides additional reinforcement of the dam structure which minimizes potential of a dam breach resulting in uncontrolled discharge of tailings towards to the open pit (to the southwest) or Highway 6 (to the east) of the TWRMF.

2.13.11 TWRMF Closure Considerations

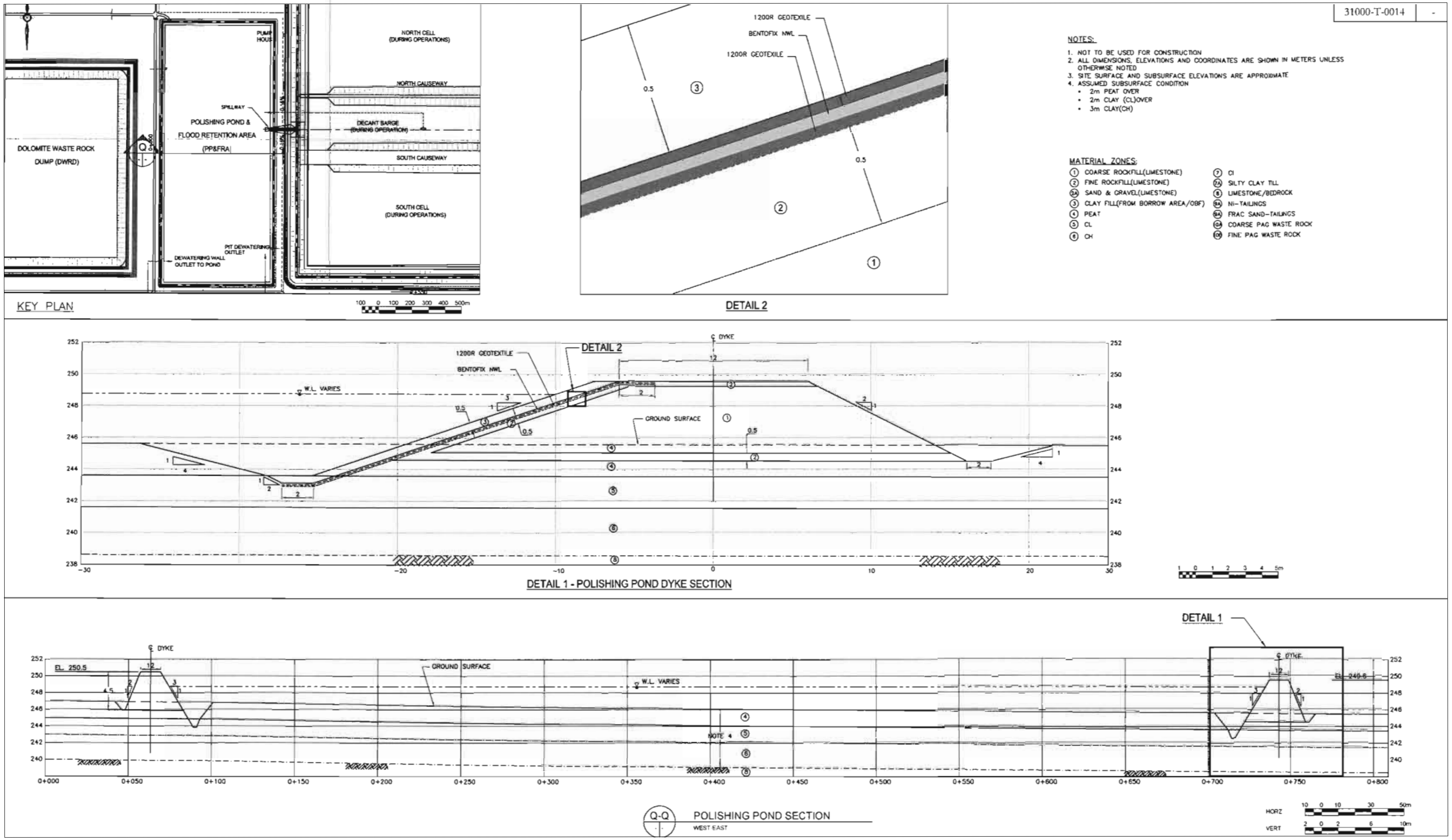
TWRMF closure aspects are covered in a separate report on closure.

2.13.12 Polishing Pond

Water in the Polishing Pond will be contained by a perimeter dyke. The plan view, section view and detail of the Polishing Pond dyke are shown in Figure 2.13-11. The dyke is designed as an earth/rock fill structure varying in height from 4.0 m to 6.0 m above the local topography. The upstream and downstream embankment slopes will be 3H:1V and 2H:1V, respectively. The dyke is scheduled to be raised in 2011 prior to the end of the dredging operations to receive water from the open pit dewatering coinciding with the last phase of the dredging operations.

The main rock fill zone (Zone 1) of the Polishing Pond dyke will be composed of 800 mm minus coarse rock, supporting a 0.5 m thick zone of 75 mm minus fine rock fill, which in turn, will support Geotextile 1200R and a Bentofix liner (Wardrop, 2009b). A 0.5 m clay cover over the Bentofix will be provided for confinement and frost protection. The geotextile at the base of the dyke along with the fine rock fill is designed to prevent migration of fines from the foundation soils into the coarse rock fill. A 0.3 m thick pavement surface, composed of fine rock fill, will be provided over the crest of the embankment (Wardrop, 2009b).

An anchor trench around the upstream toe of the Polishing Pond dyke will be extended approximately 0.5 m into the native clay to ensure full containment of the stored water. The Bentofix liner will be anchored into the trench and backfilled with locally available clay. Dewatering of the cut-off trench may be required to facilitate the installation of the Bentofix liner under dry conditions (Wardrop, 2009b). The dyke embankment will be constructed using rockfill originating from the neighboring limestone bluff located about 2 km west of the Polishing Pond.



Source: adapted from Wardrop's drawing 0951330400-DWG-T0014 (Wardrop, 2009b)

Figure 2.13-11 Polishing Pond Plan and Sections

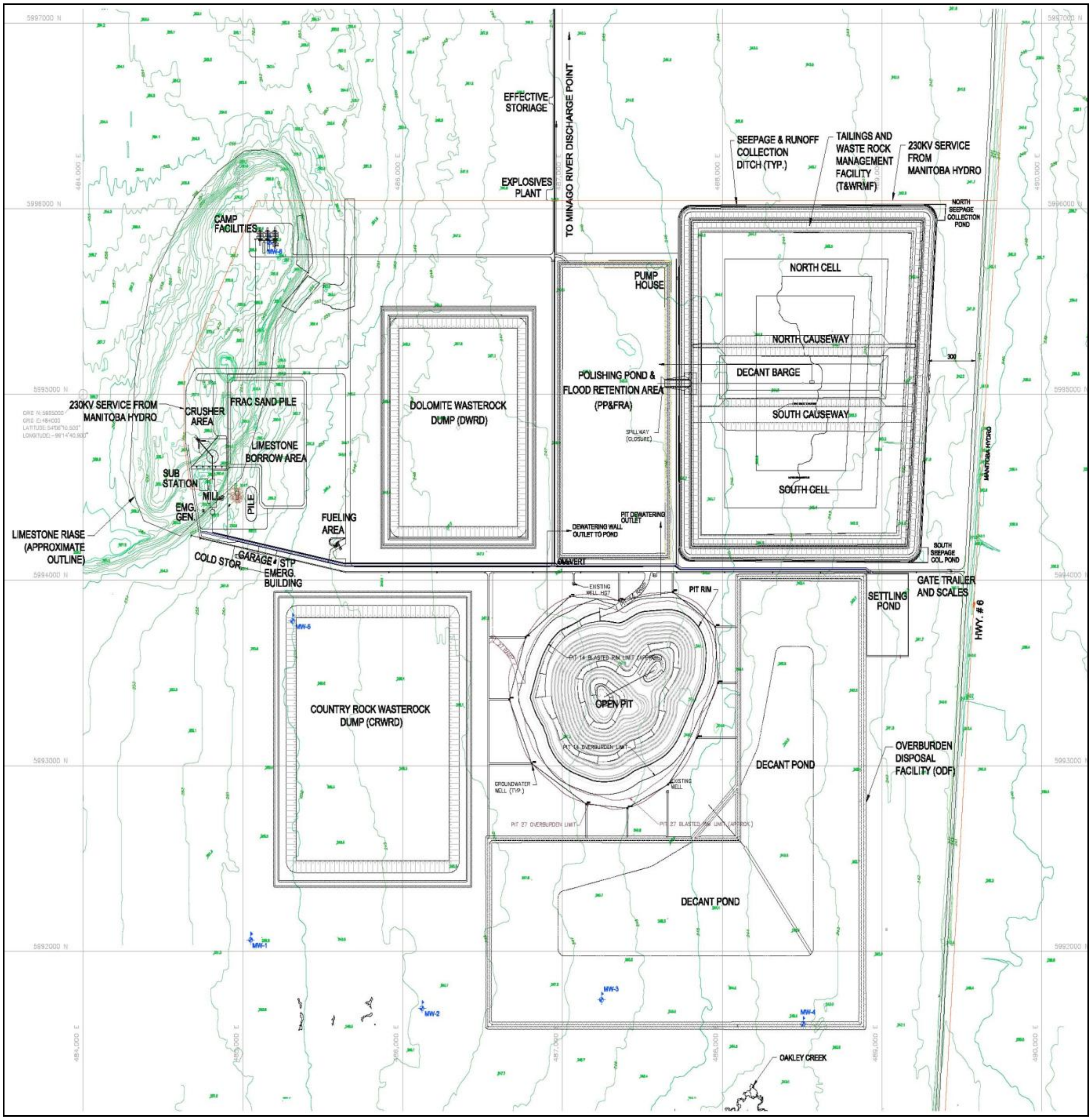
2.14 Site Water Management

This Section presents the general site water management and the description and discussion of a water balance model that was developed for the Minago Project based on the mine site layout as shown in Figure 2.14-1; metallurgical, hydrological, hydrogeological, and geochemical conditions; and related environmental baseline study results obtained to date. The goal is to manage and control site waters to ensure compliance with applicable regulations.

The water management components presented in this Section include:

- twelve dewatering wells to dewater the open pit area;
- a water treatment plant to produce potable water;
- a sewage treatment system (extended aeration system) for the disposal and treatment of on-site grey water and sewage;
- mill and Frac Sand Plant tailings and effluents that will be discharged into a Tailings and Ultramafic Waste Rock Management Facility (TWRMF);
- a Tailings and Ultramafic Waste Rock Management Facility (TWRMF) that will store tailings and the ultramafic waste rock permanently and effluents from various site operations temporarily;
- waste rock dump seepages that will be discharged to the receiving environment or into the TWRMF depending on their water quality;
- overburden dump runoff that will be discharged directly into the receiving environment (if it meets discharge requirements);
- an open pit dewatering system that will ensure safe working conditions in and around the open pit;
- a Polishing Pond and flood retention area to serve as holding pond for water that will either be recycled to site operations or discharged to the receiving environment (if it meets discharge water standards);
- a site drainage system to prevent flooding of site operations;
- site wide water management pumping systems; and
- discharge pipelines to Minago River and Oakley Creek to discharge excess water from the Polishing Pond to the receiving environment.

Among the sources of water that need to be managed are the pit dewatering well water, TWRMF supernatant and precipitation (rainfall and snowfall). Primary losses of precipitation include sublimation, evaporation, and retention as pore water in sediments and soils. Seepage losses to groundwater (e.g. from the TWRMF), which should increase due to dewatering, will likely be very small due to the thick layer of clay that is underlying the muskeg.



Source: Wardrop, 2009b

Figure 2.14-1 Mine Site Layout

The vertical hydraulic conductivity (K_V) of the overburden clay, which is an aquitard overlying the limestone, was estimated to range from 4×10^{-9} m/s to 6×10^{-9} m/s and the horizontal hydraulic conductivity, K_H, was estimated to range from 6×10^{-6} m/s to 6×10^{-9} m/s, with a geometric mean of 4×10^{-8} m/s (Golder Associates, 2008b). These hydraulic conductivities are indicative of an anisotropy ratio (K_H/K_V) of 10 (Golder Associates, 2008b).

2.14.1 General Description of the Site Water Management System

Water at Minago will be managed to ensure safe working conditions and minimum impacts to the local and regional surface and groundwater flow regimes and the aquatic environment. As water will be managed to suit site activities, the discussion of the site water management system was broken down into the following seven scenarios:

- Water Management during Construction;
- Water Management during Nickel and Frac Sand Plants Operations (Yr 1 through Yr 8);
- Water Management during Frac Sand Plant Operations (Years 9 and 10);
- Water Management during Closure;
- Water Management during Post Closure;
- Water Management during Temporary Suspension; and
- Water Management during the State of Inactivity.

Closure involves decommissioning of processing facilities and buildings and infrastructure that are no longer needed. The closure period is a transition stage between the operational and the post closure periods.

The post closure period refers to the period after all decommissioning activities of mining facilities and infrastructure have been completed and the site is in its final, post mining state.

"Temporary suspension" means that advanced exploration, mining or mine production activities have been suspended due to factors such as low metal prices and mine related factors such as ground control problems or labour disputes. Temporary suspension does not occur under normal operating conditions. The site will be monitored continuously during the Temporary Suspension (TS) of operations and dewatering of the open pit will continue as it did during operations. TS may become a "State of Inactivity", if the TS is extended indefinitely.

The "State of Inactivity" implies that mine production and mine operations at the mine site have been suspended indefinitely. The State of Inactivity also does not occur under normal operating conditions. The State of Inactivity (SI) may turn into a state of permanent closure, if prevailing conditions for the resumption of operations are not favourable. During the State of Inactivity, mine dewatering will be reduced significantly and only a minimal crew will be assigned to the site to monitor and ensure safety on site.

2.14.1.1 Water Management System during Construction

To facilitate the description of the water management model during construction, key components are illustrated with boxes in a schematic water balance diagram, given in Figure 2.14-2, and flow(s) in and out of each box are numbered (Q1 through Q24). All flows in the schematic water balance diagram are from left to right.

Following is a description of the water management model during construction, depicted in Figure 2.14-2:

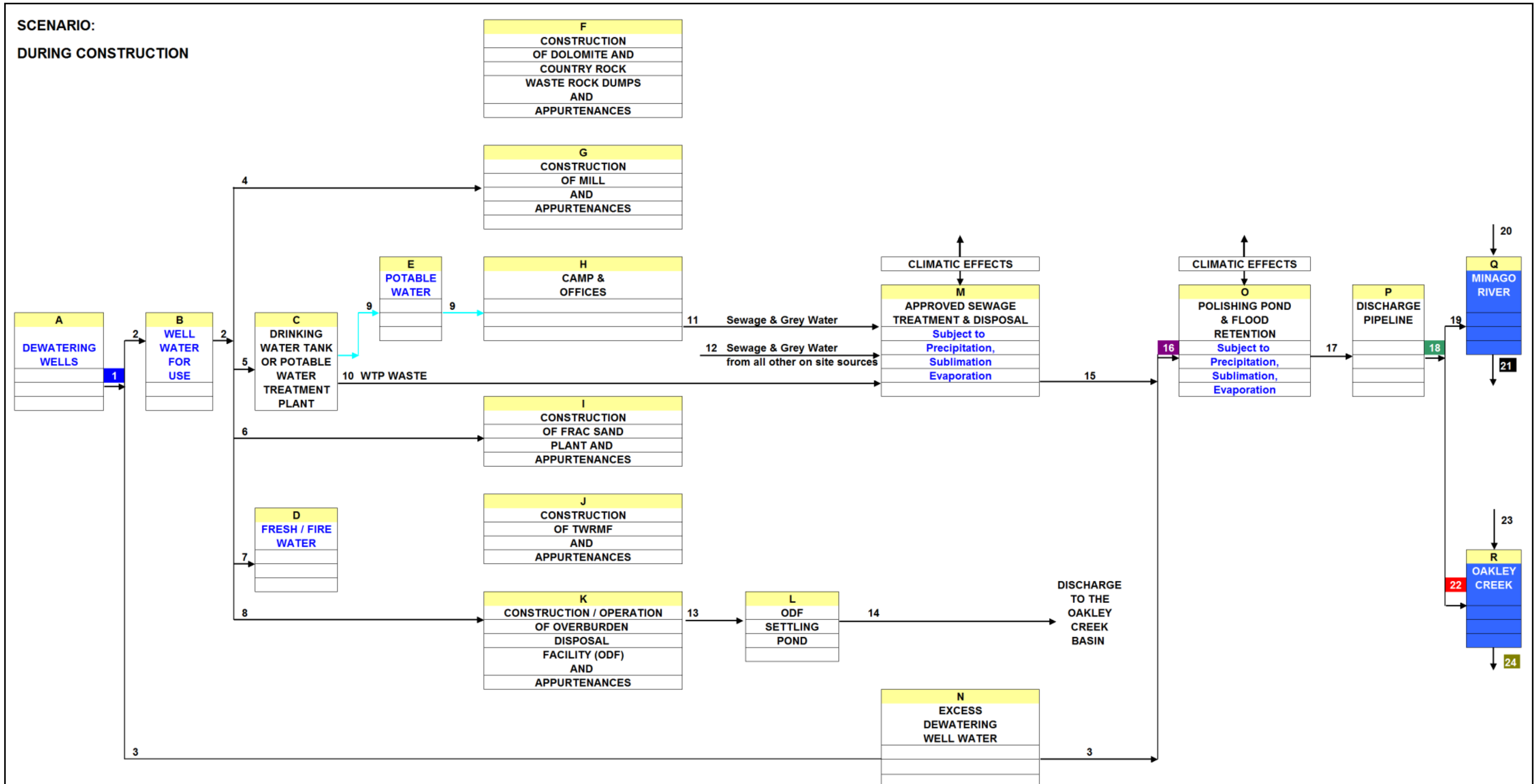
- **Dewatering Well Water (Flow Q1):**

  To allow ore extraction, the open pit area needs to be dewatered. Dewatering will start during the construction phase. Based on pumping tests conducted by GAIA in 2008, 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 geological 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 Associates, 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 sensitivity analyses, 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) (Golder Associates, 2008b).

  In the Minago water balance model, presented towards the end of this section, a dewatering rate of 40,000 m^3/day was assumed (32,000 m^3/day originating from the dewatering wells and 8,000 m^3/day from dewatering of the Open Pit).

- **Process Water and Dewatering Well Water (Flows Q2, Q3, Q4, Q5, Q6, Q7, and Q8):**

  Water from the dewatering wells will be used as process water (Q2) for construction activities of the mill complex and appurtenances (Q4), as input to the potable water treatment plant (Q5), as input to the Frac Sand Plant construction site (Q6), as fire water (Q7), and for the construction of the Overburden Disposal Facility (ODF) and dredging of overburden (Q8).



SYSTEM OF LINEAR EQUATIONS (FOR FLOWS)

DEWATERING WELLS	Q1 = Q2+Q3
WELL WATER FOR USE	Q2 = Q4 + Q5 + Q6 + Q7 + Q8
POLISING POND DISCHARGE WATER	Q18
MINAGO RIVER DOWNSTREAM	Q21 = Q19+Q20
OAKLEY CREEK DOWNSTREAM	Q24 = Q22+Q23

Figure 2.14-2 Water Management System during Construction

- **Potable Water / Grey Water / Sewage (Flows Q9, Q10, Q11, Q12 and Q15):**

  A water treatment plant to produce potable water will be operated at the Minago site to produce sufficient potable water for the camp (Q9), all other on-site personnel and any other processes that require potable water. Sludge from the potable water treatment plant (Q10) will be disposed of in an approved sewage treatment system.

  All on-site grey water and sewage (Q11 and Q12) will be collected and discharged to an approved sewage treatment system. Outflow from the sewage treatment system (Q15) will be discharged to the Polishing Pond.

  The sewage treatment system will be subject to the climatic effects of precipitation, sublimation, and evaporation.

- **ODF Settling Pond (Flows Q13 and Q14):**

  Construction of the Overburden Disposal Facility may require some dewatering well water and dredging of the overburden (Q13), while underway, will require almost all of the dewatering water (~35,000 m^3/day) (Wardrop, 2010). Discharge of ODF seepage will be released to the environment via an ODF Settling Pond (Q14). Only water meeting the discharge criteria will be discharged to the Oakley Creek basin for ultimate discharge to Oakley Creek.

- **Polishing Pond (PP) (Flows Q3, Q15, Q16, and Q17):**

  Storm water, outflow from the approved sewage treatment system (Q15) and excess dewatering well water (Q3) will be discharged to the Polishing Pond. This water containment will ensure that quality standards are met prior to discharge. Water contained in the Polishing Pond will be discharged to the receiving environment via a discharge pipeline system (Q18), to the Minago River (Q19) and the Oakley Creek (Q22). Detailed engineering will be undertaken to determine the exact location of the pipeline and of the discharge point. Stream crossings will be avoided and environmental impacts will be minimized as much as possible.

  The Polishing Pond will be used as water storage, final settling pond, and flood retention area. The Polishing Pond will be approximately 75 ha in area with a gross storage capacity of approximately 3.04 million m^3. The Polishing Pond will be subject to the climatic effects of precipitation, sublimation, and evaporation.

- **Discharge System to Minago River (year round) (Flow Q19):**

  Discharge to the Minago River (Q19) willl occur year round at rates that will be adjusted seasonally to ensure that the discharged flows will not impact the flow regime nor the flora and fauna in Minago River negatively.

In the water balance model, it was assumed that 70% of all water to be discharged from the Polishing Pond will be directed towards Minago River during the non-winter months (May to October). In the winter months (Nov. – Apr.), 65% of all excess Polishing Pond water will be discharged to the Minago River and 35% will be stored in the Polishing Pond for discharge during the subsequent freshet (May).

- **Discharge System to Oakley Creek (Summer) (Flow Q22):**

  It was assumed that Oakley Creek will be completely frozen during the winter months and therefore no discharges are planned to Oakley Creek in the winter months. Discharge to Oakley Creek (Q22) will occur from May to October. Discharges to Oakley Creek will be adjusted seasonally to ensure that the discharged water will not impact the flow regime nor the flora and fauna in Oakley Creek negatively. It was assumed that 30% of excess Polishing Pond water will be discharged to Oakley Creek during the non-winter months (May to October).

2.14.1.2 Water Management System during Operations

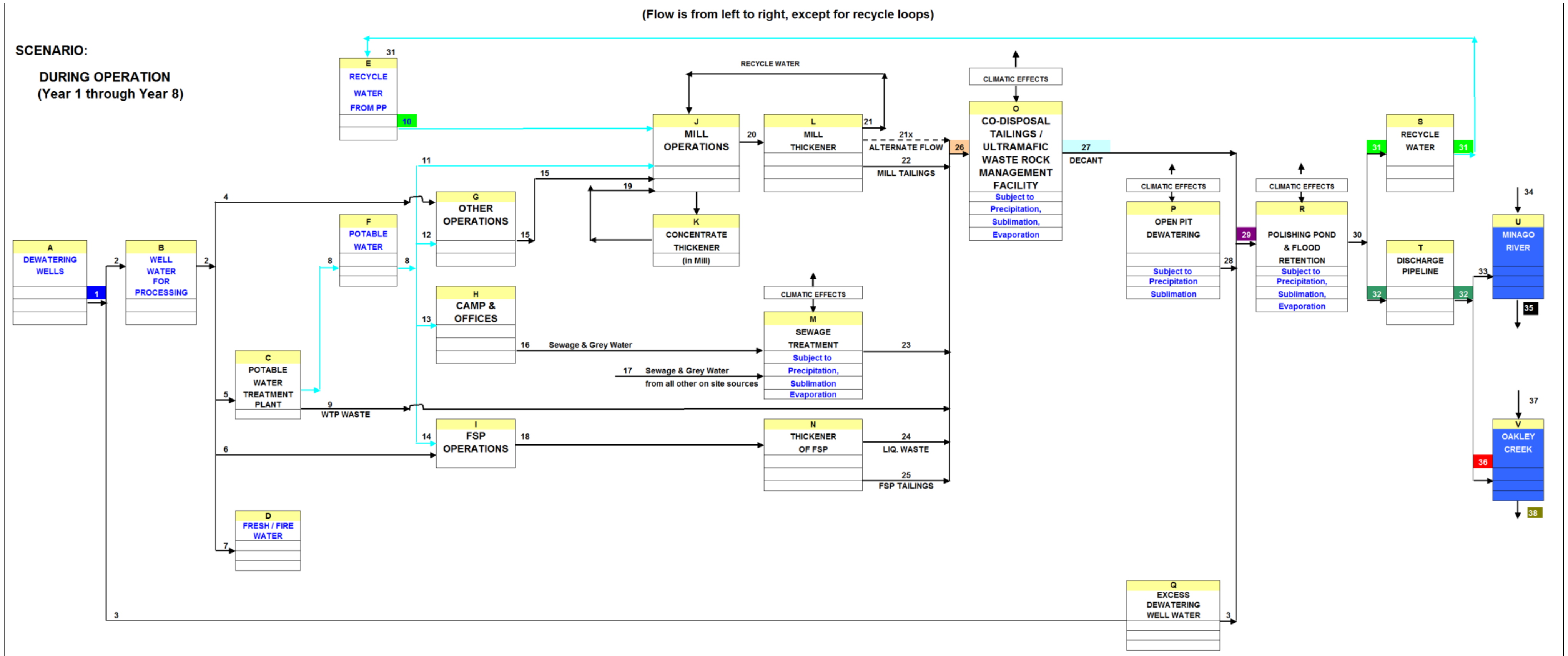
The operational period at Minago will consist of two distinct periods. In Year 1 through Year 8, both the Nickel Processing Plant and the Frac Sand Plant will be operating. In Year 9 and Year 10, the Nickel Processing Plant will be decommissioned based on current projections of nickel resources, but the Frac Sand Plant will be operating.

To facilitate the description of the water management model, key components are illustrated with boxes in the schematic water balance diagram (Figure 2.14-3) and flow(s) in and out of each box are numbered (Q1 through Q38). All flows in the schematic water balance diagram are from left to right (which is the typical flow direction) except for flows in recycle loops, which flow from right to left.

Following is a description of the water management model during the Year 1 through Year 8:

- **Dewatering Well Water (Flow Q1):**

  To allow ore extraction, the open pit area needs will be dewatered. Based on pumping tests conducted by GAIA in 2008, 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 geological 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 Associates, 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 sensitivity analyses, the actual dewatering



SYSTEM OF LINEAR EQUATIONS (FOR FLOWS)

DEWATERING WELLS	Q1 = Q2+Q3
PROCESS WATER	Q2 = Q4 + Q5 + Q6 + Q7
RECYCLED WATER FROM FINAL POLISHING POND	Q10 = Q31
TMF INFLOW	Q26 = Q9+Q21x+Q22+Q23+Q24+Q25
TMF DECANT	Q27
POLISHING POND INFLOW	Q29 = Q3+Q27+Q28
DISCHARGED WATER	Q32
MINAGO RIVER DOWNSTREAM	Q35 = Q33+Q34
OAKLEY CREEK DOWNSTREAM	Q38 = Q36+Q37

Figure 2.14-3 Water Management System during the Nickel and Frac Sand Plants Operations (in Years 1 through 8)

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) (Golder Associates, 2008b).

In the Minago water balance model, presented towards the end of this section, a dewatering rate of 40,000 m^3/day was assumed (32,000 m^3/day originating from the dewatering wells and 8,000 m^3/day from dewatering of the Open Pit).

- **Process Water and Dewatering Well Water (Flows Q2, Q3, Q4, Q5, Q6, and Q7):**

  Water from the dewatering wells will be used as process water (Q2) in the industrial complex (Q4), as input to the potable water treatment plant (Q5), as input to the Frac Sand Plant (Q6), and as fire water (Q7). Any excess dewatering well water not required for processing purposes (Q3) will be discharged to the Polishing Pond.

- **Potable Water / Grey Water / Sewage (Flows Q8, Q9, Q11, Q12, Q13, Q14, Q16, Q17, and Q23):**

  A water treatment plant to produce potable water will be operated at the Minago site to produce sufficient potable water (Q8) for the camp and offices (Q13), all other on-site personnel (Q11, Q12, and Q14), and any other processes that require potable water. Sludge from the potable water treatment plant (Q9) will be disposed of in the TWRMF.

  All on-site grey water and sewage (Q16 and Q17) will be collected and discharged to an extended aeration treatment system. Outflow from the sewage treatment system (Q23) will be discharged to the TWRMF.

  The sewage treatment system will be subject to the climatic effects of precipitation, sublimation, and evaporation.

- **Mill complex (Flows Q10, Q11, Q15, Q19, Q20, Q21, Q21x, and Q22):**

  Milling operations at Minago will be located on the north western side of the site and north of the access road (Figure 2.14-1). Schematically, the mill complex is illustrated with ‘Mill Operations’, ‘Concentrate Thickener in Mill’, and ‘Mill Thickener’ in Figure 2.14-3.

  The mill complex has the following inflows:

  1) Recycle water from the Polishing Pond (Q10);
  2) Potable water (Q11);
  3) primary crusher products and crushed ore from the Other Operations area (as well as water used for dust suppression) (Q15);
  4) recovered water from the concentrate thickener (Q19); and
  5) Recycle water from the mill thickener (Q21).

Outflows from the mill complex are nickel concentrate that will be shipped for sale and tailings slurry (Q22) that will be discharged to the Tailings and Ultramafic Waste Rock Management Facility (TWRMF). If the quality of the mill recycle water does not meet the process water quality standards for the mill, a portion of the recycle water from the Mill Thickener (Q21x) may also be discharged into the TWRMF. However, the redirection of the recycle water from the Mill Thickener is not expected under normal operating conditions.

- **Frac Sand Plant (Flows Q6, Q14, Q18, Q24 and Q25):**

The Frac Sand Plant will receive process water (Q6) consisting of dewatering well water and potable water (Q14). Liquid waste from the Frac Sand Plant (Q18) will be directed towards the thickener of the Frac Sand Plant.

Frac Sand Plant tailings (Q25) and related liquid waste (Q24) from the Frac Sand Plant will be discharged to the TWRMF.

- **Other Operations (Flow Q15):**

The term 'Other Operations' in the context of this site water management plan refers to the primary crusher, crushed ore tunnel, maintenance building, fueling area, and substation. The main outflow of the Other Operations Area (Q15) will be crushed ore that will be directed towards the mill complex. Grey water and sewage from the Other Operations Area will be discharged to the sewage treatment system. Hydrocarbons and other potentially deleterious substances in the Other Operations Area will be handled, stored and disposed of in an appropriate manner in compliance with all applicable regulations and guidelines and will not be discharged to the TWRMF.

- **Tailings and Ultramafic Waste Rock Management Facility (Flows Q9, Q21x, Q22, Q23, Q24, and Q25):**

The Tailings and Ultramafic Waste Rock Management Facility (TWRMF) is a key component of the water and waste management system at Minago for liquid waste, tailings and ultramafic waste rock management. The TWRMF will serve as repository for mill and Frac Sand Plant tailings and ultramafic waste rock.

Tailings and ultramafic waste rock will be disposed concurrently in the TWRMF and will be stored subaqueously. Key elements of the concurrent disposal of tailings and ultramafic waste rock are detailed in Section 2.13.

Submerging tailings containing sulphide minerals, or "subaqueous disposal", is practiced at many metal mines to keep oxidative rates at a minimum and to minimize metal leaching. Based on geochemical work done to date, Minago's mill tailings contain low sulphide levels and were deemed to be non acid generating (NAG) (URS, 2009i). Sulphide levels were less than or equal to 0.07 % in the Master tailings samples tested. However, the Precambiran ultamafic waste rock is potentially acid generating (URS, 2008i).

The TWRMF will remain in place after all operations have ceased at the site. The TWRMF inflow (Q26) will consist of:

- alternate flow from the mill thickener (only if warranted) (Q21x);
- mill tailings (Q22);
- sludge from the potable water treatment plant (Q9);
- liquid waste from the Frac Sand Plant (Q24);
- tailings from the Frac Sand Plant (Q25); and
- outflow from the sewage treatment system (Q23).

The TWRMF will also be subject to the climatic effects of precipitation, evaporation and sublimation.

Outflows from the TWRMF include the TWRMF Decant (Q27) and losses due to evaporation and sublimation, and seepage. Seepage will be captured by interceptor ditches surrounding the TWRMF and will be pumped back to the TWRMF. The flow volume of the TWRMF Decant will be regulated automatically by a control system.

During the operational phase, deposited waste will be kept under a nominal 0.5 m thick water cover. The design of the facility will include several baffles and/or barriers to encourage the settlement of suspended solids and to ensure that the TWRMF decant has a low suspended solids concentration.

The TWRMF will provide 38 million m^3 of storage with a maximum water surface area of approximately 219.7 ha (Wardrop, 2010).

- **Open Pit Dewatering (Flow Q28):**

During the mining phase, the open pit will be dewatered to ensure safe and dry working conditions in the pit. Open pit dewatering (Q28) will be subject to the climatic effects of precipitation and sublimation.

The excess open pit dewatering water will be pumped to the Polishing Pond.

- **Polishing Pond (PP) (Flows Q3, Q27, Q28, Q29, Q30, Q31, Q32, Q33 and Q36):**

The Polishing Pond will be used as water storage, final settling pond, and flood retention area. The Polishing Pond will be approximately 75 ha in area with a gross storage capacity of approximately 3.04 million m^3. This water containment structure will ensure that quality standards are met prior to discharge. Water contained in the Polishing Pond will be pumped to the Minago River watershed, the Oakley Creek watershed and to the process water tank as reclaim water.

The Polishing Pond will receive decant water from the TWRMF (Q27), dewatering water from the Open Pit (Q28), excess groundwater from the twelve (12) mine dewatering wells (Q3), and precipitation. Under normal operating conditions, when meeting water quality standards, water retained by the Polishing Pond (Q30) will either be recycled to the milling process (Q31 = Q10) or discharged to the receiving environment via a discharge pipeline system (Q32), which discharges water to the Minago River (Q33) and the Oakley Creek (Q36).

Storm water from the waste rock dumps, the TWRMF and the in-pit dewatering system will also be channelled into a Polishing Pond.

The Polishing Pond will also be subject to the climatic effects of precipitation, evaporation and sublimation.

- **Discharge System to Minago River (year round) (Flow Q33):**

  Discharge to the Minago River (Q33) will occur year round at rates that will be adjusted seasonally to ensure that the discharged flows will not impact the flow regime nor the flora and fauna in the Minago River negatively.

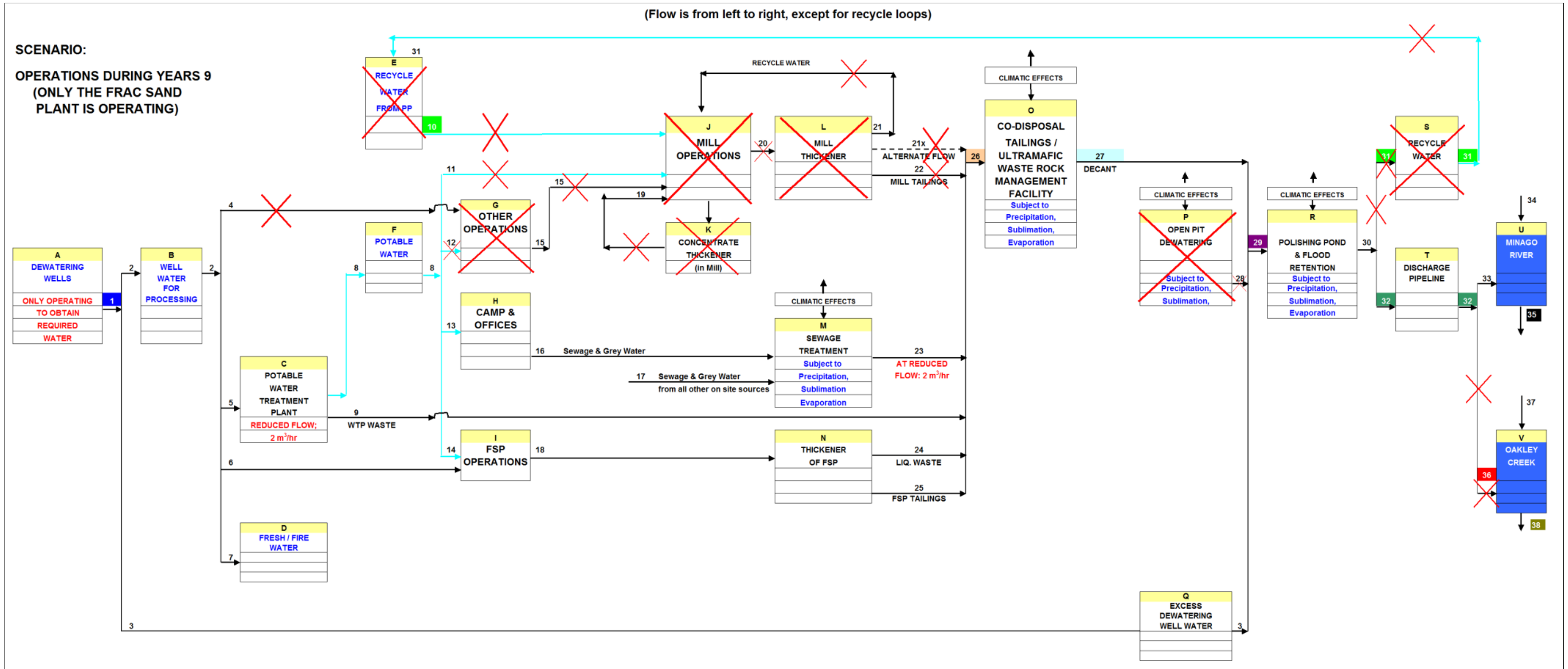
  In the water balance model, it was assumed that 70% of all excess Polishing Pond water will be directed towards the Minago River during the non-winter months (May to October) and that 65% of it will be discharged to the Minago River during the winter months (November to April).

- **Discharge System to Oakley Creek (Summer) (Flow Q36):**

  It was assumed that Oakley Creek will be completely frozen during the winter months and therefore no discharges are planned to Oakley Creek in the winter months (Nov. – Apr.). Discharge to the Oakley Creek (Q36) will occur from May to October. Discharges to the Oakley Creek will be adjusted seasonally to ensure that the discharged water will not impact the flow regime nor the flora and fauna in the Oakley Creek negatively. It was assumed that 30% of the excess Polishing Pond water will be discharged to the Oakley Creek during non-winter months (May – Oct.).

2.14.1.3 Water Management System during Frac Sand Plant Operations in Year 9 and 10

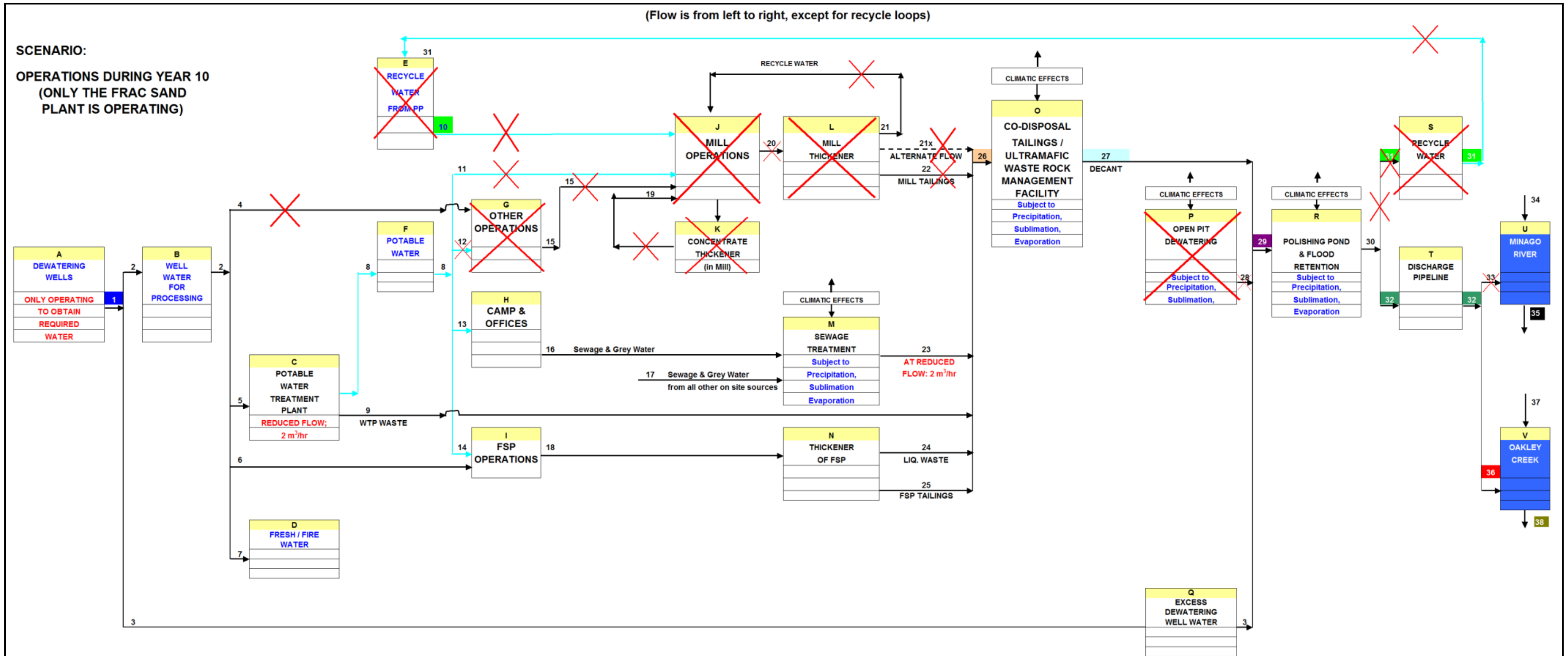
In Year 9 and Year 10, the Nickel Processing Plant will be decommissioned based on current projections of nickel resources, but the Frac Sand Plant will be operating as before. Accordingly, the extent of the water management system will be scaled back significantly. Less water will be needed for operations; and therefore, the mine dewatering program will be scaled down significantly. No water will be required nor discharged from the Nickel Processing Plant complex



SYSTEM OF LINEAR EQUATIONS (FOR FLOWS)

DEWATERING WELLS	Q1 = Q2+Q3
PROCESS WATER	Q2 = Q4 + Q5 + Q6 + Q7
RECYCLED WATER FROM FINAL POLISHING POND	Q10 = Q31
TMF INFLOW	Q26 = Q9+Q21x+Q22+Q23+Q24+Q25
TMF DECANT	Q27
POLISHING POND INFLOW	Q29 = Q3+Q27+Q28
DISCHARGED WATER	Q32
MINAGO RIVER DOWNSTREAM	Q35 = Q33+Q34
OAKLEY CREEK DOWNSTREAM	Q38 = Q36+Q37

Figure 2.14-4 Water Management System during Frac Sand Plant Operations in Year 9



SYSTEM OF LINEAR EQUATIONS (FOR FLOWS)

DEWATERING WELLS	Q1 = Q2+Q3
PROCESS WATER	Q2 = Q4 + Q5 + Q6 + Q7
RECYCLED WATER FROM FINAL POLISHING POND	Q10 = Q31
TMF INFLOW	Q26 = Q9+Q21x+Q22+Q23+Q24+Q25
TMF DECANT	Q27
POLISHING POND INFLOW	Q29 = Q3+Q27+Q28
DISCHARGED WATER	Q32
MINAGO RIVER DOWNSTREAM	Q35 = Q33+Q34
OAKLEY CREEK DOWNSTREAM	Q38 = Q36+Q37

Figure 2.14-5 Water Management System during Frac Sand Plant Operations in Year 10

during these years. The Open Pit dewatering will cease. These changes in the water management program compared to the Year 1 through Year 8 water management program are illustrated in Figures 2.14-4 and 2.14-5. Figure 2.14-4 shows conditions in Year 9 and Figure 2.14-5 illustrates conditions in Year 10.

In Figure 2.14-4, all components that will not be active in the water management system (i.e., for which flows will be zero) are shown as crossed out. All other flows and water balance components will remain the same as they will have been during the Year 1 through Year 8 operations.

Following is a short list of the flow conditions with respect to "zero" flows in Year 9 and Year 10:

- Dewatering Well Water (Flow Q1 => only one well will be operating);
- Process Water and Dewatering Well Water (Flows Q2, Q3, Q4=0, Q5, Q6, and Q7);
- Potable Water / Grey Water / Sewage (Flows Q8, Q9, Q11=0, Q12=0, Q14, Q16, Q17, Q23);
- Mill complex: It will be closed (Flows Q10=0, Q11=0, Q15=0, Q19=0, Q20=0, Q21=0, Q21x=0, and Q22=0);
- Frac Sand Plant (Flows Q6, Q14, Q18, Q24 and Q25);
- Other Operations (Flow Q15=0);
- Tailings and Ultramafic Waste Rock Management Facility (Flows Q9, Q21x=0, Q22=0, Q23, Q24, and Q25);
- Open Pit Dewatering (Flow Q28=0);
- Polishing Pond (PP) (Flows Q3, Q27, Q28=0, Q29, Q30, Q31=Q10=0, Q32, Q33 and Q36);
- Discharge System to the Minago River (year round) (Flow Q33):

  Year 9: In the Year 9 water balance model, it was assumed that 100% of all water to be discharged from the Polishing Pond will be directed towards Minago River (Q33) year round to achieve a staged reduction of discharges. The discharge will range from 1% to 5% of the average seasonal flows in the Minago River, as detailed lateron in this Section.

  Year 10: There will be no Polishing Pond discharges to Minago River (Q33=0) in Year 10.

- Discharge System to the Oakley Creek (Summer)(Flow Q36):

  It was assumed that Oakley Creek will be completely frozen during the winter months and therefore no discharges are planned to Oakley Creek in the winter months (Nov. to Apr.).

  Year 9: In the Year 9 water balance model, it was assumed that 0% of the Polishing Pond discharges will be directed towards the Oakley Creek (Q36).

Year 10: In Year 10, there will be no discharge to Oakley Creek in the winter months (Nov. to Apr.), but 100% of the Polishing Pond discharges will be directed towards Oakley Creek for the remainder of the year.

2.14.1.4 Water Management System during Closure

During the closure period, site and infrastructure decommissioning and site reclamation will take place and all processing facilities and appurtenances will be shut down. Water management during the closure period is illustrated in Figures 2.14-6 and 2.14-7. The first stage of the closure period is illustrated in Figure 2.14-6 and the second stage of the closure period is illustrated in Figure 2.14-7.

The following components will operate during the first stage of closure: dewatering wells, potable water treatment plant (at an appropriate rate based on on-site personnel), sewage treatment system, TWRMF, and the Polishing Pond. All of these components, with the exception of the dewatering wells, will be the same as was described for the Year 1 to Year 8 operational period. The dewatering wells will be used to install a 1.5 m high water cover on top of the TWRMF.

All water management components for the second stage of closure will be the same as for the first stage except for the dewatering wells. All dewatering wells will be decommissioned in the second stage of closure.

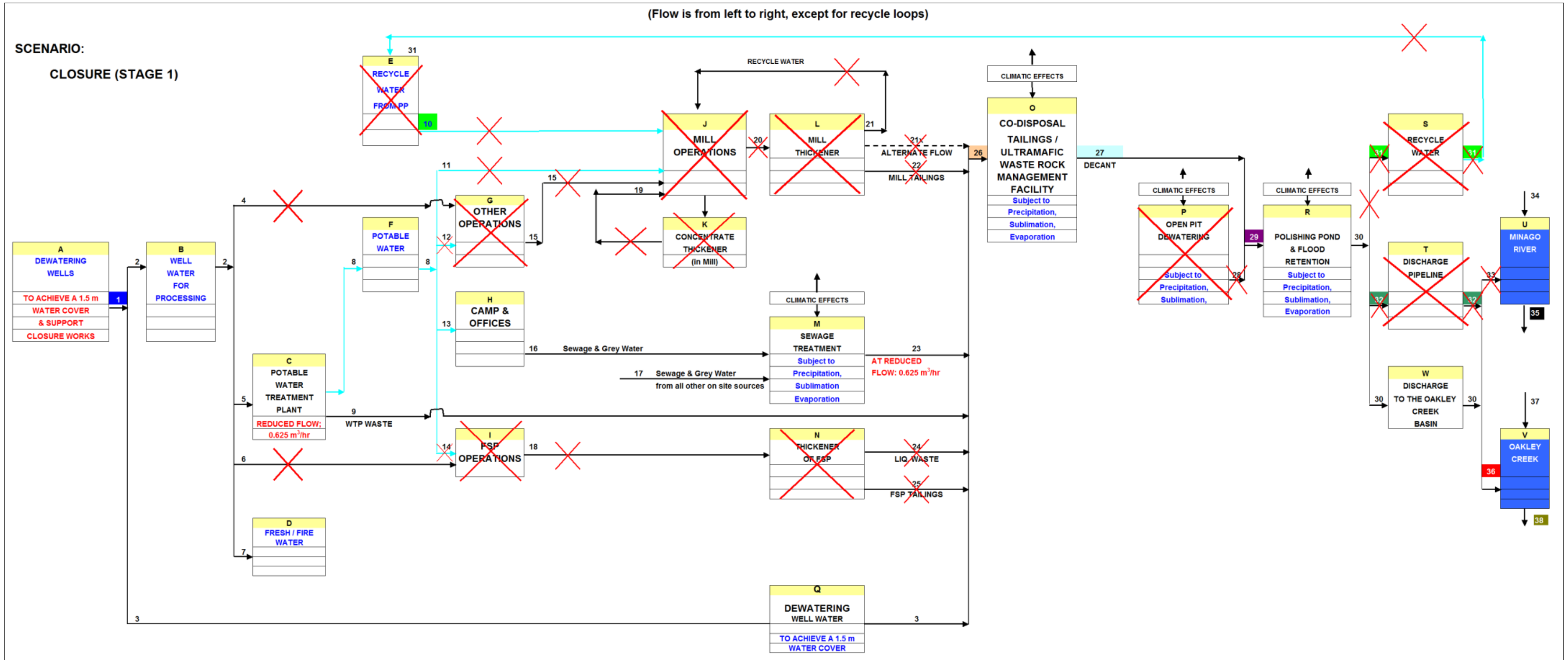
Water will be discharged from the Polishing Pond via a spillway to the Oakley Creek basin for ultimate discharge to Oakley Creek.

During the closure phase, the Tailings and Ultramafic Waste Rock Management Facility (TWRMF) will be reclaimed as a permanent pond. The access road will remain in place. Reclamation goals are a stabilized surface and a native plant community to provide wildlife habitat. The TWRMF embankments will be modified to ensure long-term saturation of the tailings and the ultramafic waste rock and to provide a spillway for ultimate passive decanting of the TWRMF at closure. The spillway will have been installed with an invert elevation approximately 1.5 m above the deposited tailings. The spillway will be installed before the closure phase and will allow controlled discharge of TWRMF supernatant (Q27) that is in excess of the 1.5 m high water cover.

2.14.1.5 Water Management System during Post Closure

Water management during the post closure period is illustrated in Figure 2.14-8. In the post closure period, all mining facilities and infrastructure will have been decommissioned with the exception of the TWRMF and the Polishing Pond.

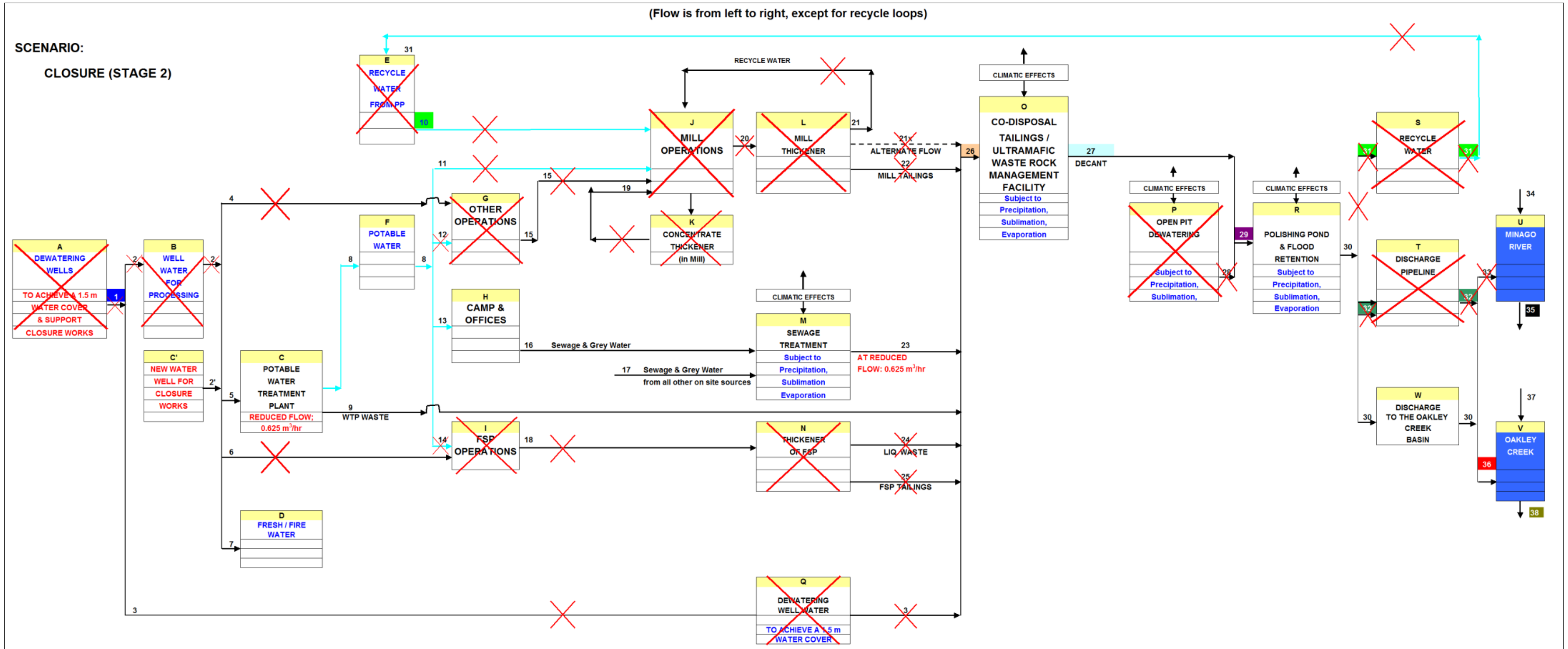
In the post closure phase, the TWRMF will have been decommissioned and reclaimed as much as possible.



SYSTEM OF LINEAR EQUATIONS (FOR FLOWS)

DEWATERING WELLS	Q1 = Q2+Q3
PROCESS WATER	Q2 = Q4 + Q5 + Q6 + Q7
RECYCLED WATER FROM FINAL POLISHING POND	Q10 = Q31
TMF INFLOW	Q26 = Q3+Q9+Q21x+Q22+Q23+Q24+Q25
TMF DECANT	Q27
POLISHING POND INFLOW	Q29 = Q27+Q28
DISCHARGED WATER	Q32
MINAGO RIVER DOWNSTREAM	Q35 = Q33+Q34
OAKLEY CREEK DOWNSTREAM	Q38 = Q36+Q37

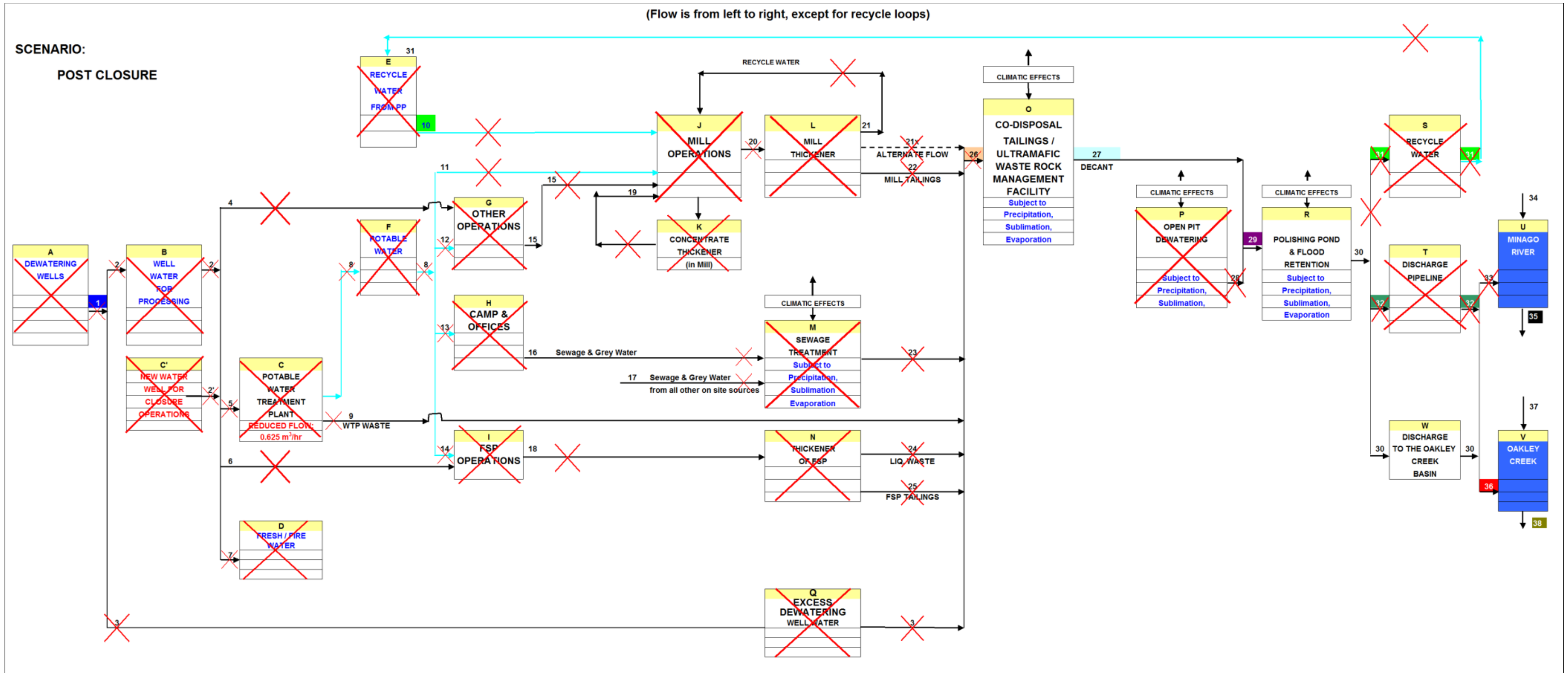
Figure 2.14-6 Water Management System during First Stage of Closure



SYSTEM OF LINEAR EQUATIONS (FOR FLOWS)

DEWATERING WELLS	Q1 = Q2+Q3
PROCESS WATER	Q2 = Q4 + Q5 + Q6 + Q7
RECYCLED WATER FROM FINAL POLISHING POND	Q10 = Q31
TMF INFLOW	Q26 = Q3+Q9+Q21x+Q22+Q23+Q24+Q25
TMF DECANT	Q27
POLISHING POND INFLOW	Q29 = Q27+Q28
DISCHARGED WATER	Q32
MINAGO RIVER DOWNSTREAM	Q35 = Q33+Q34
OAKLEY CREEK DOWNSTREAM	Q38 = Q36+Q37

Figure 2.14-7 Water Management System during Second Stage of Closure



SYSTEM OF LINEAR EQUATIONS (FOR FLOWS)

DEWATERING WELLS	Q1 = Q2+Q3
PROCESS WATER	Q2 = Q4 + Q5 + Q6 + Q7
RECYCLED WATER FROM FINAL POLISHING POND	Q10 = Q31
TMF INFLOW	Q26 = Q3+Q9+Q21x+Q22+Q23+Q24+Q25
TMF DECANT	Q27
POLISHING POND INFLOW	Q29 = Q27+Q28
DISCHARGED WATER	Q32
MINAGO RIVER DOWNSTREAM	Q35 = Q33+Q34
OAKLEY CREEK DOWNSTREAM	Q38 = Q36+Q37

Figure 2.14-8 Post Closure Water Management System

2.14.1.6 Water Management System during Temporary Suspension

A schematic of the site water management system during the temporary suspension (TS) of operations is given in Figure 2.14-9. As the name implies, the state of Temporary Suspension is typically temporary in nature. Temporary suspension does not occur under normal operating conditions. Due to the temporary nature of the state of Temporary Suspension, only production related facilities at the site such as the mill complex (mill operations, mill thickener, concentrate thickener in the mill), Frac Sand Plant, the thickener of the Frac Sand Plant, and Other Operations will be suspended. During Temporary Suspension, recycling of water from the Polishing Pond will also cease, but the mine site and open pit will still be dewatered as was done during site operations.

Continued dewatering of the site will permit a timely start-up after the temporary suspension of site operations is lifted and normal operations resume.

All other components of the water management system that will not be shut down will be as was described previously for the Year 1 to Year 8 operational period.

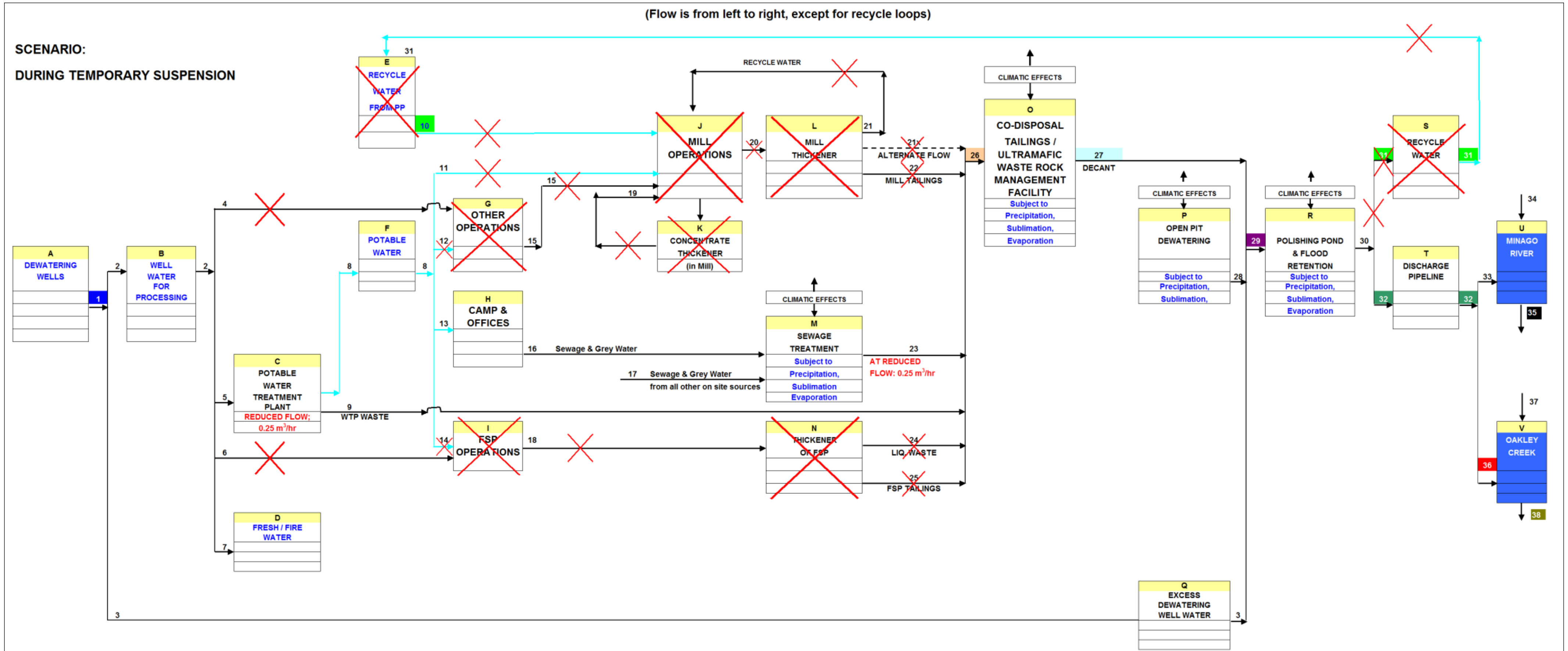
In the water balance model, it was assumed that the state of Temporary Suspension will occur at the end of Year 4.

2.14.1.7 Water Management System during a State of Inactivity

A schematic of the site water management system during a State of Inactivity (SI) is given in Figure 2.14-10. The State of Inactivity does not occur under normal operating conditions. During the State of Inactivity, all process related operations will cease and the mill complex (mill operations, mill thickener, concentrate thickener in the mill), Frac Sand Plant, the thickener of the Frac Sand Plant, and Other Operations will be shut down. Recycling of water from the Polishing Pond to the mill will also cease and dewatering of the open pit will be significantly reduced. As illustrated in Figure 2.14-10, only one out of the twelve dewatering wells will be operating to supply water for the remaining activities at Minago. Dewatering of the open pit mine will also cease.

All other components of the water management system that will not be shut down will be as was described for the Year 1 to Year 8 operational period.

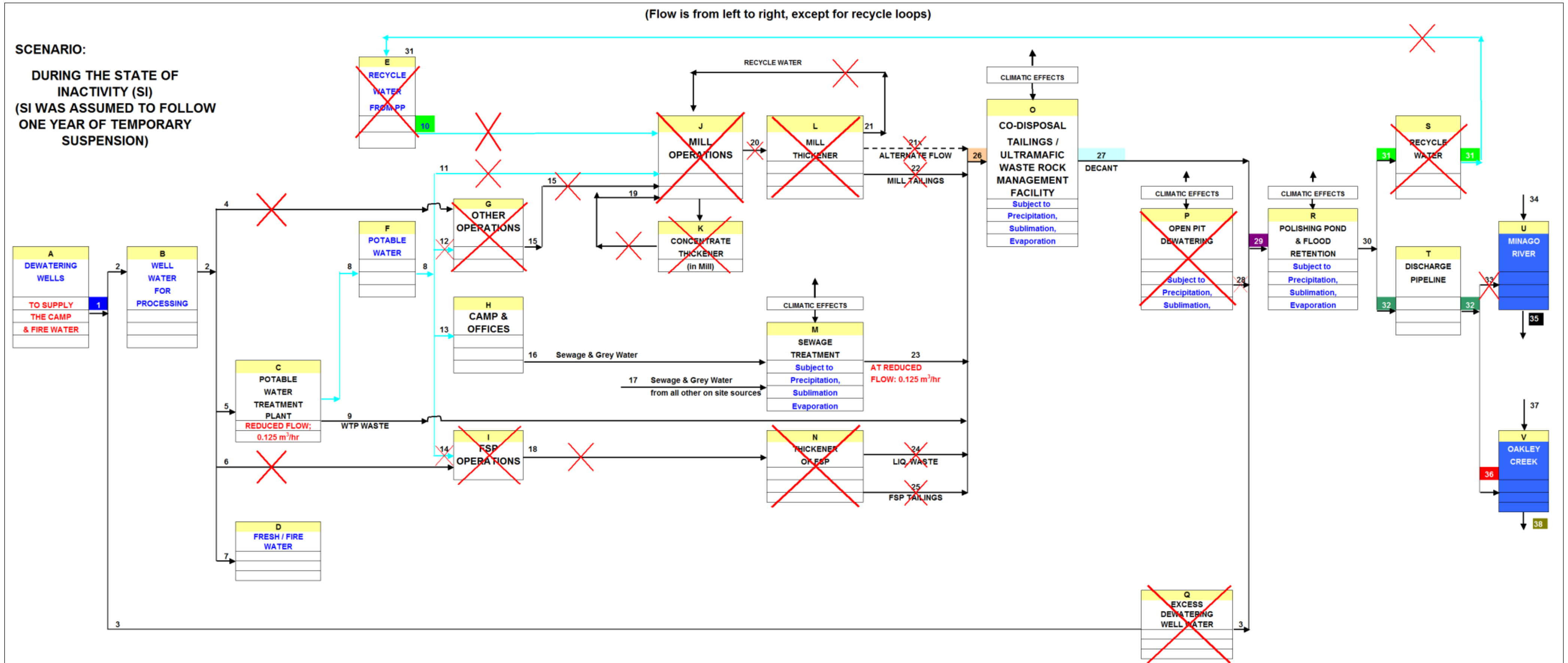
In the Minago water balance model, the State of Inactivity was assumed to have occurred after one year of Temporary Suspension at the end of Year 5.



SYSTEM OF LINEAR EQUATIONS (FOR FLOWS)

DEWATERING WELLS	Q1 = Q2+Q3
PROCESS WATER	Q2 = Q4 + Q5 + Q6 + Q7
RECYCLED WATER FROM FINAL POLISHING POND	Q10 = Q31
TMF INFLOW	Q26 = Q9+Q21x+Q22+Q23+Q24+Q25
TMF DECANT	Q27
POLISHING POND INFLOW	Q29 = Q3+Q27+Q28
DISCHARGED WATER	Q32
MINAGO RIVER DOWNSTREAM	Q35 = Q33+Q34
OAKLEY CREEK DOWNSTREAM	Q38 = Q36+Q37

Figure 2.14-9 Water Management System during Temporary Suspension



SYSTEM OF LINEAR EQUATIONS (FOR FLOWS)

DEWATERING WELLS	Q1 = Q2+Q3
PROCESS WATER	Q2 = Q4 + Q5 + Q6 + Q7
RECYCLED WATER FROM FINAL POLISHING POND	Q10 = Q31
TMF INFLOW	Q26 = Q9+Q21x+Q22+Q23+Q24+Q25
TMF DECANT	Q27
POLISHING POND INFLOW	Q29 = Q3+Q27+Q28
DISCHARGED WATER	Q32
MINAGO RIVER DOWNSTREAM	Q35 = Q33+Q34
OAKLEY CREEK DOWNSTREAM	Q38 = Q36+Q37

Figure 2.14-10 Water Management System during a State of Inactivity

2.14.2 Minago Water Balance Model

A Water Balance Model (WBM) was developed to estimate average elemental concentrations in flows that will be part of the working mine. The water balance was developed based on expected baseline inputs and outputs. Inputs and outputs are related to three main aspects including dewatering well water and its uses and discharges (chemistry and flow); mining and milling processes to produce concentrate and saleable products out of the ore (chemistry and flow); and climatic conditions (rainfall, snowfall, sublimation, and evaporation). Key input parameters and considerations of the water balance model are summarized below, first in general terms and then in detail.

As for the general description of the water management system, the water balance model is described for the following seven scenarios in this document:

- water balance during Construction (illustrated in Figure 2.14-2);
- water balance during Nickel and Frac Sand Plants Operations (in Years 1 through 8) (illustrated in Figure 2.14-3);
- water balance during Frac Sand Plant Operations (in Years 9 and 10) (illustrated in Figure 2.14-4);
- water balance during Closure (illustrated in Figures 2.14-6 and 2.14-7);
- water balance during Post Closure (illustrated in Figure 2.14-8).
- water balance during Temporary Suspension (illustrated in Figure 2.14-9); and
- water balance during the State of Inactivity (illustrated in Figure 2.14-10).

2.14.2.1 General Description of Inputs and Outputs of the Water Balance Model

The primary water inputs of the water balance model are due to dewatering wells that enable mining in the open pit by lowering the water table. In the water balance model, it was assumed that approximately 32,000 m^3/day will be pumped from 12 dewatering wells that surround the open pit and 8,000 m^3/day will be pumped from the Open Pit (Golder Associates, 2008b). Dewatering well water will be used for processing in the mill and Frac Sand Plant and to create potable water. However, the vast majority (approximately 84%) of the dewatering well water will be discharged unused to the Polishing Pond for subsequent discharge to the receiving environment (Minago River or Oakley Creek) during the mine operations as well as during the State of Inactivity and Temporary Suspension, should they occur.

Another major input into the water balance model are precipitation and associated climatic effects (evaporation, sublimation, etc). All large storage areas (including the waste rock dumps, the Tailings and Ultramafic Waste Rock Management Facility (TWRMF), the Open pit, the Polishing Pond, and the sewage treatment system) will be subject to climatic effects.

Input parameters and considerations used to characterize climatic effects for the Minago Project are as follows:

- **Precipitation**

  The precipitation at Minago was assumed to be 510 mm consisting of 369 mm (72%) of rain and 141 mm (28%) of snow (Golder Associates, 2009). It was assumed that 40 mm (10.8%) of the rain falls in the month of May and 329 mm (89.2%) in the period of June to October (Golder Associates, 2009).

- **Snow Storage**

  Snow sublimation and redistribution has a notable impact on the amount of water in the snowpack and therefore affects the water balance of site facilities and related watersheds. Sublimation can occur directly from snowpack surfaces or during blowing snow events with overall rates dependent on humidity and wind speed (Essery et al., 1999; Déry and Yau, 2002). Snow sublimation is highly dependent on the thermal balance of the snowpack. Golder Associates (2009) projected an average snow sublimation rate of 39% of the average annual snowfall for the Minago Project.

- **Snowmelt**

  In the water balance model, snowmelt was assumed to occur in the month of May.

- **Lake Evaporation and Evapotranspiration**

  Evaporation is the process by which water is transferred from land and water to the atmosphere. Transpiration is the evaporation of water from the vascular system of plants to the atmosphere. The combination of both processes is termed evapotranspiration and is a function of the type of surface (open water, leaf or leaf canopy, bare soil, etc.), the availability of water, and the net energy input into the system.

  The seasonal distribution of evaporation is affected primarily by solar radiation and vegetation cover (or lack of it). During the snowmelt period, evaporation is relatively small compared with the large supply of melt water within a thinly thawed active layer (Woo and Steer, 1983). Typically, evaporation is greatest following snowmelt and decreases through the summer period. Evaporation decreases as the latitude increases. Evaporation losses from lakes are greater than evapotranspiration losses from an equivalent terrestrial area.

  Lake evaporation in the vicinity of the proposed project site is expected to be 500 mm or more (EMRC, 1995), while evapotranspiration is estimated to range between 350 and 400 mm (EMRC, 1995). The majority of the water balance components at Minago will not be subjected to transpirational effects as they will be bare “brown” fields.

In the Minago water balance model, it was assumed that the evaporation from the Tailings and Ultramafic Waste Rock Management Facility (TWRMF), the Polishing Pond, and the sewage treatment system will be 50% of the lake evaporation estimated for large lakes in the vicinity of the Minago Project. Evaporation was assumed to be 56 mm in May, 218.35 mm in the period from June to October (over a period of 154 days), and 0 mm in the winter months (November to April). Evaporation losses were assumed to be negligible for the waste rock dumps (due to the coarseness of the material leading to negligible water storage on the surface) and the open pit due to the continuous removal (pumping) of water that infiltrates the open pit during operations.

- **Ice Regime**

The mean ice thickness in the vicinity of the Minago Project is expected to be between 0.75 and 1 m in lakes and rivers (Allen, 1977). The freeze-over window is expected to be early to mid November, while the ice-free date is typically in mid April (Allen, 1977).

Based on March, 2008, field measurements, Oakley Creek was found to be completely frozen near Highway #6 (at monitoring station OCW1) during the field monitoring program. As such, it is proposed not to discharge any water to Oakley Creek in the winter months.

Outputs

Discharges to Minago River and Oakley Creek watersheds are the major “output” of the water balance model. All other clean, potable, grey, and processing waters will be managed internally at the Minago Project.

2.14.2.2 Detailed Imput Parameters and Considerations of the Water Balance Model

Key input parameters and considerations of the Minago water balance model are presented below. These key input parameters and considerations include climatic conditions and the stages of Operations, Closure and Post Closure as well as Temporary Suspension and the State of Inactivity. Based on the stated input parameters and considerations, elemental concentrations and flowrates were estimated for combined flows that will have a bearing on the receiving environment.

- **Key Climatic Input Parameters and Considerations**

Key climatic parameters used for the water balance model are given in Table 2.14-1.

Table 2.14-1 Climatic Parameters and Considerations used for the Minago Water Balance Model

PRECIPITATION:		
Average annual precipitation:	**510 mm**	Source: Golder Associates (2009)
72% falls as rain:	369 mm	Source: Golder Associates (2009)
28% falls as snow:	141 mm	Source: Golder Associates (2009)
Snow Sublimation:		
39% of annual snow fall:	54.99 mm	Source: Golder Associates (2009)
Water equivalent remaining in the spring:	= 141-54.99 mm = 86.01 mm	Source: Golder Associates (2009)
Water Balance Model Assumptions:		
- It was assumed that 40 mm of rain falls in May (31 days).		Source: Golder Associates (2009)
- It was assumed that 141 mm of snow falls between November and April (180 days). It was assumed that 86.01 mm water equivalent remains of the snow precipitation in the spring.		Source: Golder Associates (2009)
- It was assumed that 329 mm of rain falls in June, July, August, September, October (2.1364 mm/day over 154 days)		Source: Golder Associates (2009)
LAKE EVAPORATION:		
Average annual lake evaporation:	566.0 mm	Source: Golder Associates (2009)
in April:	17.6 mm	Source: Golder Associates (2009)
in May:	112.0 mm	Source: Golder Associates (2009)
in period from June to October:	436.7 mm	Source: Golder Associates (2009)
Water Balance Model Assumptions:		
It was assumed that water evaporates from the sewage treatment system, TWRMF, and Polishing Pond at 50% of the lake evaporation measured for big lakes in the vicinity of the Minago Project. For the 50% evaporation model, it was assumed that 56 mm evaporate in the month of May (1.80645 mm/day over 31 days) and 218.35 mm (1.4179 mm/day over 154 days) evaporate in June, July, August, September and October.		

- **Key Input Parameters and Considerations for Nickel and Frac Sand Plant Operations (Year 1 through Year 8) (Figure 2.14-3):**

  1. The Nickel Processing Plant and the Frac Sand Plant and related appurtenances will be operating.
  2. All twelve dewatering wells will be running and the Open Pit will be dewatered.
  3. Tailings and ultramafic waste rock will be concurrently disposed in a Tailings and Waste Rock Management Facility (TWRMF).
  4. Only the deposited Ni tailings will leach at the maximum leaching rate measured during kinetic testing in the subaqueous leach column surface water.
  5. Voids in freshly deposited tailings will represent 22% of the tailings stream. Voids remaining in the ultramafic waste rock after concurrent disposal with tailings were assumed to represent 6.9% of the total volume of the waste rock and its voids (Wardop, 2010). All voids were assumed to be filled with water of the same quality as the supernatant of the TWRMF. This porewater was assumed to be unavailable for discharge from the TWRMF.
  6. On-site daily potable water consumption per person was assumed to be ~ 300 L.
  7. The TWRMF will have a water cover with a nominal thickness of 0.5 m during the operational phase.
  8. Excess groundwater from the dewatering wells will be discharged to the Polishing Pond all year round.
  9. In the winter months (Nov. to Apr.), 65% of the Polishing Pond water will be discharged to the Minago River and 35% will be stored in the Polishing Pond. During the remainder of the year (May to October), 70% of the Polishing Pond water will be discharged to the Minago River and 30% will be discharged to the Oakley Creek.

- **Key Input Parameters and Considerations for Frac Sand Plant Operation in Year 9 (Figure 2.14-4):**

  1. The Frac Sand Plant will operate and frac sand tailings will be deposited in the TWRMF.
  2. All operations will have ceased at the Nickel Processing Plant and related facilities and no more Ni tailings nor waste rock will be created or disposed.
  3. Only the deposited Ni tailings will leach at the maximum leaching rate measured during kinetic testing in the subaqueous leach column surface water.
  4. The TWRMF will have a water cover of a nominal thickness of 0.5 m.
  5. Dewatering pumps will be restricted to pump only sufficient water for frac sand processing and other site operations.

6. All of the Polishing Pond water will be discharged to the Minago River year round and discharge will be staged to prepare the aquatic habitat for complete withdrawal of discharges from the Polishing Pond.

- **Key Input Parameters and Considerations for Frac Sand Plant Operation in Year 10 (Figure 2.14-5):**

  All input parameters and considerations are as for Year 9 except for the discharge of Polishing Pond water. All of the Polishing Pond water will be stored in the winter months (Nov. to April) and discharged to the Oakley Creek watershed during the remainder of the year (May to October).

- **Key Input Parameters and Considerations for Closure:**

  The closure period was broken down into two stages (first and second) for which the input parameters and considerations are summarized below.

  Considerations for the First Stage of Closure (Figure 2.14-6):

1. All operations will have ceased at the Mill and Frac Sand Plant and related appurtenances.
2. Open pit dewatering will have ceased.
3. Water will be pumped from the dewatering wells to the TWRMF to provide a 1.5 m high water cover.
4. Only the deposited Ni tailings will leach at the maximum leaching rate measured during kinetic testing in the subaqueous leach column surface water.
5. On-site potable water consumption was assumed to be 15 m^3/day (~ 300 L/person/day for 30 people).
6. Polishing Pond supernatant will be discharged to the Oakley Creek basin via a spillway for ultimate discharge to Oakley Creek.

Considerations for the Second Stage of Closure (Figure 2.14-7):

All input parameters and considerations are as for first stage of closure except for the dewatering wells. The dewatering wells will be decommissioned, once a water cover of 1.5 m height will have been installed on top of the TWRMF.

- **Key Input Parameters and Considerations for Post Closure (Figure 2.14-8):**

1. All decommissioning activities of mining facilities and infrastructure will have been completed.

2. Only the deposited Ni tailings will leach at the maximum leaching rate measured during kinetic testing in the subaqueous leach column surface water.
3. TWRMF supernatant in excess of the 1.5 m water cover will be discharged to the Polishing Pond via a spillway.
4. Polishing Pond supernatant will be discharged to the Oakley Creek basin via a spillway for ultimate discharge to Oakley Creek.

- **Key Input Parameters and Considerations for Temporary Suspension (TS) at the end of Year 4:**

1. All operations will have ceased at the Mill and Frac Sand Plant and related appurtenances at the end of Year 4. TS means that advanced exploration, mining or mine production activities have been suspended due to factors such as low metal prices, or mine related factors such as ground control problems and labour disputes.
2. No more tailings will be deposited into the TWRMF.
3. Only deposited Ni tailings will leach at the maximum leaching rate measured during kinetic testing in the subaqueous leach column surface water.
4. Dewatering wells will be running as usual during regular operations.
5. On-site potable water consumption was assumed to be 6 m^3/day (~ 300 L/person/day for 20 people).
6. Excess groundwater from the dewatering wells will be discharged to the Polishing Pond all year round.
7. TWRMF will have a water cover of a nominal thickness of 0.5 m. Excess supernatant from the TWRMF will be discharged to the Polishing Pond.
8. During the winter months (Nov. to Apr.), 65% of the Polishing Pond water will be discharged to the Minago River and 35% will be stored in the Polishing Pond. During the remainder of the year (May to October), 70% of the Polishing Pond water will be discharged to the Minago River and 30% will be discharged to the Oakley Creek.

- **Key Input Parameters and Considerations for The State of Inactivity (SI)**

1. State of Inactivity was assumed to have occurred after one year of Temporary Suspension at the end of Year 5. SI means that mine production and mining operations on site have been suspended indefinitely.
2. No tailings will be deposited into the TWRMF.
3. Only deposited Ni tailings will leach at the maximum leaching rate measured during kinetic testing in the subaqueous leach column surface water.

4. Operations will have ceased at the Nickel Processing Plant and Frac Sand Plant and related appurtenances.
5. One dewatering well will be running, but only to supply the camp and site activities with water.
6. On-site potable water consumption was assumed to be 3 m^3/day (~ 300 L/person/day for 10 people).
9. TWRMF will have a water cover of a nominal thickness of 0.5 m. Excess supernatant from the TWRMF will be discharged to the Polishing Pond.
10. During the winter months (Nov. to Apr.), none of the Polishing Pond water will be discharged. During the remainder of the year (May to October), 100% of the Polishing Pond water will be discharged to the Oakley Creek.

- **Key Input Parameters and Considerations for the Calculation of Flowrates:**

  Key input parameters and considerations for flowrate calculations are detailed in Table 2.14-2. Efforts were made to use flowrates that are representative of anticipated site conditions. All flowrates not detailed in Table 2.14-2 were based on material flowsheets developed by Wardrop Engineering Inc. (Wardrop) and others and are presented as part of the presentation of modeling results.

- **Key Input Parameters and Considerations for the Calculation of Elemental Concentrations:**

  Key input parameters and considerations for contaminant loadings and element concentrations in the water balance flows are summarized in Table 2.14-3. Efforts were made to use concentrations that are representative of anticipated site and geochemical conditions.

- **Key Input Parameters and Considerations for Flowrates in Minago River and Oakley Creek:**

  Key input parameters and considerations for flowrates in Minago River and Oakley Creek are summarized in Table 2.14-4.

- **Assumed Weekly Metal Leaching Rates for the Minago Tailings**

  The metal leaching rates assumed for Minago tailings are detailed in Table 2.14-5 and correspond to 10% of surface water loadings measured for the subaqeous column in kinetic tests that were run for 54 weeks (URS, 2009). Steady State was assumed after week 11 (URS, 2008i).

- **Assumed Areas of Site Faciltiies:**

  The areas of site facilities that were used in the water balance model are detailed in Table 2.14-6.

- **Input Data – Material Flow Rates and Conditions for the TWRMF:**

  Assumed material flow rates and conditions for the TWRMF are detailed in Table 2.14-7.

Table 2.14-2 Key Input Parameters and Considerations for Flowrate Calculations in the Minago Water Balance Model

Flowrates Qi (i = 1 to 38)		Mathematical Formulae to determine Qi (i = 1 - 38)
UNIT EVAPORATION (1 Unit = 1 ha)	UNIT LAKE EVAPORATION	= Q-Unit-Evapo
UNIT PRECIPITATION (1 Unit = 1 ha)		
Q1	FLOW FROM DEWATERING WELLS	as per Feasibility Study
Q2	WELL WATER FOR PROCESSING	
Q3	EXCESS WATER FROM DEWATERING WELLS	
Q4	GROUNDWATER TO OTHER OPERATIONS	
Q5	GROUNDWATER TO WATER TREATMENT	
Q6	GROUNDWATER TO FRAC SAND PLANT	
Q7	GROUNDWATER FOR FIRE FIGHTING	
Q8	POTABLE WATER	
Q9	WATER TREATMENT PLANT WASTE	
Q10	RECYCLE WATER FROM POLISHING POND	= Q32
Q11	POTABLE WATER TO MILL	as per Feasibility Study
Q12	POTABLE WATER TO OTHER OPERATIONS	
Q13	POTABLE WATER TO OFFICES & CAMP	
Q14	POTABLE WATER TO FRAC SAND PLANT	
Q15	FLOW FROM OPERATIONS TO MILL	
Q16	SEWAGE & GREY WATER FROM CAMP AND OFFICES	
Q17	SEWAGE & GREY WATER FROM ALL OTHER ON SITE SOURCES	
Q18	FLOW FROM FSP OPERATIONS TO FSP THICKENER	
Q19	FLOW FROM CONCENTRATE THICKENER IN MILL TO MILL	
Q20	FLOW FROM MILL TO MILL THICKENER	
Q21	RECYCLE WATER FROM MILL THICKENER	
Q21x	ATERNATE FLOW FOR RECYCLE WATER FROM MILL THICKENER	
Q22	MILL TAILINGS SLURRY	
Q23	SEWAGE TREATMENT OUTFLOW	
Q24	LIQ. WASTE FROM FSP	
Q25	SLURRY FROM FRAC SAND PLANT (FSP)	
Q26	TWRMF INFLOW	= Q9 + Q21x + Q22 + Q23 + Q24 + Q25
Q - Liquid Precipitation on TWRMF	Available Precipitation on TWRMF	= AREA*Q-Unit-PPT
Q - Evaporation from TWRMF	Evaporation from TWRMF	= AREA*(Q-Evapo from TWRMF)
Q - Retained Water in Tailings Voids	Retained Water in Tailings Voids	= 22% Retained Water in Voids; assumed tailings density = 1.5 tonnes/m3
Q - TWRMF Supernatant	TWRMF Supernatant	= Q26+(Q-Remaining Supernatant)+Q-PPT on TWRMF-(Q-Evapo from TWRMF) - (Q-Retained Water in Voids)
Q27	TWRMF DECANT	= TWRMF Supernatant minus 0.5 m water during Operations
Q - Pit Dewatering	OPEN PIT DEWATERING	= 8000 m3/day during Operations;= 0 m^3/day thereafter
Q - Precipitation on Pit	Precipitation minus Sublimation on Open Pit	= AREA*Q-Unit-PPT
Q28	TOTAL OPEN PIT DEWATERING	= (Q-Pit Dewatering)+(Q-PPT on Pit)
Q29	POLISHING POND INFLOW	= (Q3+Q27+Q28) during Operations
Q - Precipitation on Polising Pond	Precipitation minus Sublimation ON POLISHING POND	= AREA*Q-Unit-PPT
Q - Evaporation from Polishing Pond	EVAPORATION FROM POLISHING POND	= AREA*Q-Unit-Evapo
Q30	POLISHING POND OUTFLOW	= Q29 + (Q-PPT on Polishing Pond) - (Q-Evapo from Polishing Pond)
Q31	RECYCLE FROM FINAL POLISHING POND	as per Feasibility Study
Q32	FLOW TO DISCHARGE PIPELINE	as per Feasibility Study
Q33	DISCHARGE TO MINAGO	= 65% of Q32 during winter and 70% of Q32 otherwise during Operations
Q34	MINAGO UPSTREAM	as per Hydrologic Study
Q35	MINAGO DOWNSTREAM	= Q33+Q34
Q36	DISCHARGE TO OAKLEY CREEK	= 0% of Q32 during winter; 30% of Q32 otherwise during Operations
Q37	OAKLEY CREEK UPSTREAM	as per Hydrologic Study
Q38	OAKLEY CREEK DOWNSTREAM	= Q36+Q37

Table 2.14-3 Key Input Parameters and Considerations for Calculations of Elemental Concentrations in the Minago Water Balance Model

Concentration Ci (in Flow Qi)	Mathematical Formulae to determine Ci (i = 1 to 38)
UNIT EVAPORATION	
UNIT PPT (U-PPT)	= CCME Mean Detection Limits
C1	= Aug-2008 Groundwater Quality (Dissolved Metals)
C2	= Aug-2008 Groundwater Quality (Dissolved Metals)
C3	= Aug-2008 Groundwater Quality (Dissolved Metals)
C4	= Aug-2008 Groundwater Quality (Dissolved Metals)
C5	= Aug-2008 Groundwater Quality (Dissolved Metals)
C6	= Aug-2008 Groundwater Quality (Dissolved Metals)
C7	= Aug-2008 Groundwater Quality (Dissolved Metals)
C8	= CCME Mean Detection Limits
C9	not assumed
C10	= C32
C11	= CCME Mean Detection Limits
C12	= CCME Mean Detection Limits
C13	= CCME Mean Detection Limits
C14	= CCME Mean Detection Limits
C15	Internal Nickel Processing Plant Water Quality
C16	not assumed
C17	not assumed
C18	Internal FSP Water Quality
C19	Internal Mill Water Quality
C20	
C21	
C21x	
C22	= Measured Concentration SGS Lakefield Nov. 7, 2008 Results
C23	= CCME Mean Detection Limits
C24	= Measured Dissolved Concentration for FSP Overflow
C25	= Measured Dissolved Concentration for FSP Underflow
C26	= {C9 +Q21x*C21x + Q22*C22 + Q23*C23 + Q24*C24 + Q25*C25} / Q26
C - PPT on TWRMF	= CCME Mean Detection Limits
C - Evapo from TWRMF	
C - Tailings Leachate	= {Mass of Tailings [tonnes]* Leaching Rate of Tailings [mg/kg/period]} / Q-TWRMF Supernatant [m^3/period]
C-TWRMF Supernatant	= {Q26*C26 + (Q-TWRMF Supernatant Remaining)*(C-TWRMF Supernatant Remaining) + (Q-PPT on TWRMF)*(C-PPT on TWRMF) + (Q-Tailings Leachate)*(C-Tailings Leachate) } / Q-TWRMF Supernatant
C27	= C-TWRMF Supernatant
C-Pit Dewatering	= Aug-2008 Groundwater Quality (Dissolved Metals)
C-PPT on Pit	= CCME Mean Detection Limits
C28	= {(Q-Pit Dewatering)*(C-Pit Dewatering) + (Q-PPT on Pit)*(C-PPT on Pit)} / Q28
C29	= {Q3*C3 + Q27*C27 + Q28*C28} / Q29 during Operations
C-PPT on PP	= CCME Mean Detection Limits
C-Evapo from PP	
C30	= {Q29*C29 + (Q-PPT on Polishing Pond)*(C-PPT on Polishing Pond)} / Q30
C31	= C30
C32	= C30
C33	= C30
C34	= AVERAGE 2006-2008 MINAGO RIVER WATER QUALITY (Dissolved Metals at MRW2)
C35	= {Q33*C33 + Q34*C34} / Q35
C36	= C30
C37	= AVERAGE 2006-2008 OAKLEY CK WATER QUALITY (Dissolved Metals at OCW2)
C38	= {Q36*C36 + Q37*C37} / Q38

Table 2.14-4 Estimated Flowrates in Minago River and Oakley Creek

Time Period / Stream	May m^3/s	June to October m^3/s	November to April m^3/s
Minago River	10	1.9	0.8
Oakley Creek	4	0.5	0

Table 2.14-5 Weekly Metal Leaching Rates Assumed for Minago Tailings

10% of Subaqueous Leach Column Surface Water Loading as given in URS Geochemical Memo, dated March 4, 2010				
ELEMENT	Unit	Minimum	Average	Maximum
Aluminum (Al)	mg/kg/wk	2.000E-06	2.120E-05	1.440E-04
Antimony (Sb)	mg/kg/wk	6.080E-07	9.290E-07	1.180E-06
Arsenic (As)	mg/kg/wk	2.000E-07	1.304E-06	6.400E-06
Cadmium (Cd)	mg/kg/wk	1.600E-08	7.450E-08	7.680E-07
Chromium (Cr)	mg/kg/wk	3.200E-07	1.210E-06	2.000E-06
Cobalt (Co)	mg/kg/wk	6.400E-08	6.030E-07	1.240E-06
Copper (Cu)	mg/kg/wk	1.800E-06	8.010E-06	2.240E-05
Iron (Fe)	mg/kg/wk	3.200E-06	1.570E-05	6.200E-05
Lead (Pb)	mg/kg/wk	9.280E-08	1.621E-06	1.630E-05
Molybdenum (Mo)	mg/kg/wk	6.000E-06	1.180E-05	1.960E-05
Nickel (Ni)	mg/kg/wk	1.800E-05	4.020E-05	8.420E-05
Selenium (Se)	mg/kg/wk	4.000E-07	8.720E-07	2.180E-06
Zinc (Zn)	mg/kg/wk	4.160E-06	1.300E-05	7.680E-05

Table 2.14-6 Area of Site Facilities

Designated Area	Area (ha)
Pit Area	190.0
Tailings and Ultramafic Waste Rock Management Facility (TWRMF)	219.7
Polishing Pond	75.0

Table 2.14-7 Input Data - Material Flow Rates and Conditions for the Tailings and Ultramafic Waste Rock Management Facility (TWRMF)

			Ultramafic WR in TWRMF (kT)	Ni Tailings in TWRMF (kT)	Water Cover Height	Discharge to Minago River from Discharge Pipeline	Discharge to Oakley Creek from Discharge Pipeline
Mill & Frac Sand Plant Operating	Year 1	Nov.-Apr.	8,802	1,806.364	0.5 m	65%	0%
		May	8,802	1,806.364	0.5 m	70%	30%
		Jun.-Oct.	8,802	1,806.364	0.5 m	70%	30%
	Year 2	Nov.-Apr.	14,326	5,360.918	0.5 m	65%	0%
		May	14,326	5,360.918	0.5 m	70%	30%
		Jun.-Oct.	14,326	5,360.918	0.5 m	70%	30%
	Year 3	Nov.-Apr.	19,993	8,915.472	0.5 m	65%	0%
		May	19,993	8,915.472	0.5 m	70%	30%
		Jun.-Oct.	19,993	8,915.472	0.5 m	70%	30%
	Year 4	Nov.-Apr.	25,725	12,470.026	0.5 m	65%	0%
		May	25,725	12,470.026	0.5 m	70%	30%
		Jun.-Oct.	25,725	12,470.026	0.5 m	70%	30%
	Year 5	Nov.-Apr.	30,107	16,024.580	0.5 m	65%	0%
		May	30,107	16,024.580	0.5 m	70%	30%
		Jun.-Oct.	30,107	16,024.580	0.5 m	70%	30%
	Year 6	Nov.-Apr.	33,133	19,579.134	0.5 m	65%	0%
		May	33,133	19,579.134	0.5 m	70%	30%
		Jun.-Oct.	33,133	19,579.134	0.5 m	70%	30%
	Year 7	Nov.-Apr.	35,430	23,133.688	0.5 m	65%	0%
		May	35,430	23,133.688	0.5 m	70%	30%
		Jun.-Oct.	35,430	23,133.688	0.5 m	70%	30%
	Year 8	Nov.-Apr.	35,659	24,847.808	0.5 m	65%	0%
		May	35,659	24,847.808	0.5 m	70%	30%
		Jun.-Oct.	35,659	24,847.808	0.5 m	70%	30%

Table 2.14-7 (Cont.'d) Input Data - Material Flow Rates and Conditions for the Tailings and Ultramafic Waste Rock Management Facility (TWRMF)

			Ultramafic WR in TWRMF (kT)	Ni Tailings in TWRMF (kT)	Water Cover Height	Discharge to Minago River from Discharge Pipeline	Discharge to Oakley Creek from Discharge Pipeline	Discharge to Oakley Creek via the Oakley Creek Basin	Comments
Frac Sand Plant Operating	Year 9	Nov.-Apr.	35,659	24,847.808	0.5 m	100%	0%	0%	Staging of Discharge to Minago River for Fisheries Habitat Conditioning
		May	35,659	24,847.808	0.5 m	100%	0%	0%	
		Jun.-Oct.	35,659	24,847.808	0.5 m	100%	0%	0%	
	Year 10	Nov.-Apr.	35,659	24,847.808	0.5 m	0%	0%	0%	No Discharge; Excess water will be stored in the Polishing Pond
		May	35,659	24,847.808	0.5 m	0%	100%	0%	
		Jun.-Oct.	35,659	24,847.808	0.5 m	0%	100%	0%	
Closure	Year 11	Nov.-Apr.	35,659	24,847.808	1.5 m	0%	0%	100%	Excess water from the Polishing Pond will be discharged to the Oakley Creek Basin
		May	35,659	24,847.808	1.5 m	0%	0%	100%	
		Jun.-Oct.	35,659	24,847.808	1.5 m	0%	0%	100%	
	Year 12	Nov.-Apr.	35,659	24,847.808	1.5 m	0%	0%	100%	Excess water from the Polishing Pond will be discharged to the Oakley Creek Basin
		May	35,659	24,847.808	1.5 m	0%	0%	100%	
		Jun.-Oct.	35,659	24,847.808	1.5 m	0%	0%	100%	
Post Closure	Year 13	Nov.-Apr.	35,659	24,847.808	1.5 m	0%	0%	100%	Excess water from the Polishing Pond will be discharged to the Oakley Creek Basin
		May	35,659	24,847.808	1.5 m	0%	0%	100%	
		Jun.-Oct.	35,659	24,847.808	1.5 m	0%	0%	100%	
Temporary Suspension (TS)	After Year 4	Nov.-Apr.	25,725	12,470.026	0.5 m	65%	0%	0%	
		May	25,725	12,470.026	0.5 m	70%	30%	0%	
		Jun.-Oct.	25,725	12,470.026	0.5 m	70%	30%	0%	
State of Inactivity (SI)	After one year of TS	Nov.-Apr.	25,725	12,470.026	0.5 m	0%	0%	0%	No Discharge; Excess water will be stored in the Polishing Pond
		May	25,725	12,470.026	0.5 m	0%	100%	0%	
		Jun.-Oct.	25,725	12,470.026	0.5 m	0%	100%	0%	

2.14.2.3 Results of the Minago Water Balance Model

Following are key results of the water balance model based on the assumptions outlined above. As for the general description of the water management, the water balance model results are presented for the following seven mine development phases: Construction, Operations, Closure, Post Closure, Temporary Suspension, and the State of Inactivity. Following the presentation of results, Contaminants of Concern respective to the water quality of the discharged water will be summarized.

Water balance models for all mine development phases were developed for three periods of the year: May, June to October, and November to April. These periods were chosen to represent average conditions during the freshet, summer, and winter.

Contaminant loadings and estimated elemental concentrations in the various flows of the Minago water balance model, presented below, are listed against the Metal Mining Effluent Regulations (Environment Canada, 2002a) and the Canadian Guidelines for the Protection of Aquatic Life (CCME, 2007). They are also summarized against the Manitoba Water Quality Standards, Objectives and Guidelines (Tier II and Tier III Freshwater Quality) (Williamson, 2002). These guideline limits are presented in Table 2.14-8. Parametric concentrations were estimated for aluminum (Al), antimony (Sb), arsenic (As), cadmium (Cd), chromium (Cr), cobalt (Co), copper (Cu), iron (Fe), lead (Pb), molybdenum (Mo), nickel (Ni), selenium (Se), and zinc (Zn).

The Metal Mining Effluent Regulations (MMER) were registered on June 6, 2002, under subsections 34(2), 36(5), and 38(9) of the *Fisheries Act* and replaced the MMLER and the associated *Metal Mining Liquid Effluent Guidelines* (Environment Canada, 2002a). The MMER prescribe authorized concentration limits for deleterious substances in mine effluents that discharge to waters frequented by fish. The MMER apply to all Canadian metal mines (except placer mines) that exceed an effluent flowrate of 50 m^3 per day. The MMER apply to effluent from all final discharge points (FDPs) at a mine site. A FDP is defined in the Regulations as a point beyond which the mine no longer exercises control over the quality of the effluent. The regulated MMER parameters are arsenic, copper, cyanide, lead, nickel, zinc, total suspended solids (TSS), Radium 226, and pH.

Canadian Water Quality Guidelines for the Protection of Aquatic Life define acceptable levels for substances or conditions that affect water quality such as toxic chemicals, temperature and acidity. As long as conditions are within the levels established by the guidelines, one would not expect to see negative effects in the environment (CCME, 2007). These guidelines are based on toxicity data for the most sensitive species of plants and animals found in Canadian waters and act as science-based benchmarks.

Table 2.14-8 Guideline Limits used for Interpreting Water Balance Results

Water Quality Parameter	Metal Mining Liquid Effluents (2002)		Manitoba Water Quality Standards, Objectives, and Guidelines (Williamson, 2002)		Canadian Water Quality Guidelines for the Protection of Aquatic Life
			TIER II Water Quality Objectives		
	Monthly Mean	Grab Sample	assuming hardness = 150 mg/L $CaCO_3$	Freshwater	(CCME, 2007)
Aluminum (Al)				0.005 - 0.1	0.005
Antimony (Sb)					
Arsenic (As)	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Cadmium (Cd)			0.00302 [B]	Tier II	0.000017 $10^{\{0.86[\log(hardness)]-3.2\}}$
Chromium (Cr)			0.10331 [C]	Tier II	
Cobalt (Co)					
Copper (Cu)	0.3	0.6	0.01266 [D]	Tier II	0.002
Iron (Fe)				0.3	0.3
Lead (Pb)	0.2	0.4	0.0039 [E]	Tier II	0.001
Molybdenum (Mo)				0.073	
Nickel (Ni)	0.5	1	0.07329 [F]	Tier II	0.025
Selenium (Se)				0.001	0.001
Zinc (Zn)	0.5	1	0.16657 [G]	Tier II	0.03

Tier II Water Quality Limits for arsenic, cadmium, chromium, copper, lead, nickel, and zinc are hardness dependent as follows:

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.

The Manitoba Tier II Water Quality Objectives are defined for a limited number of common pollutants (such as dissolved metals and nutrients) that are routinely controlled through licencing under the Manitoba Environment Act. Manitoba Tier II Water Quality Objectives typically form the basis for the water quality base approach when additional restrictions need to be developed to protect important uses of ground or surface waters (Williamson, 2002).

It should be noted that water quality guideline limits for heavy metals (such as cadmium, chromium, copper, lead, nickel and zinc) depend on hardness. Therefore, results presented below are listed in terms of applicable equations to determine the guideline limits based on hardness as well as for a hardness of 150 mg/L $CaCO_3$. The hardness level of 150 mg/L $CaCO_3$ was chosen as comparison for results obtained with the Minago water balance model based on water quality results obtained to date. For these results, listed in Table 2.14-9, the average hardness was 192.2 mg/L $CaCO_3$, the median hardness was 193 mg/L $CaCO_3$, and the weighted average hardness was 173.1 mg/L $CaCO_3$.

Table 2.14-9 Hardness Levels Measured at Minago

	Number of Samples	Minimum (mg/L$CaCO_3$)	Average (mg/L$CaCO_3$)	Maximum (mg/L$CaCO_3$)
Frac Sand Plant Overflow	2		171.5	194
Frac Sand Plant Underflow	2		167	192
Sub-aqueous Col. Pore Water	53	145	232	358
Sub-aqueous Col. Surface Water	53	71.2	102.8	138
Groundwater Limestone	3	242	267	287
Groundwater Sandstone	3	165	196	257
Upstream Minago (MRW2)	7	169	192	213
Downstream Minago (MRW1)	14	87.2	149	256
Upstream Oakley Cr. (OCW2)	13	169	204.8	265
Process Water (Nov. 2008 SGS Lakefield Results)	1		240	
Total	151			
Minimum		**71.2**		
Average			**192.2**	
Maximum				**358.0**
Weighted Average			**173.1**	

2.14.2.3.1 Water Balance Modeling Results during Construction (Year –3 to Year –1)

Estimated flowrates during construction prior to the dredging operations are listed in Table 2.14-10 and the corresponding water management plan is illustrated in Figure 2.14-2.

The Polishing Pond discharge to Minago River (Q19) in relation to the Minago River streamflow (Q20) will be 8% in May, 14% in the summer months (June to October) and 30% in the winter months (November to April). In absolute quantities, discharge to Minago River will range from 20,741 m^3/day to 69,360 m^3/day during construction. The Polishing Pond discharge to Oakley Creek (Q22) in relation to the Oakley Creek streamflow (Q23) will be 0% in the winter months (Nov. to Apr.), 9% in May, and 23% in the summer months (June to October). In absolute quantities, discharge to Oakley Creek will range from 0 m^3/day to 29,725 m^3/day during construction.

Table 2.14-11 presents projected parametric concentrations for the Polishing Pond outflow (Q17), Minago downstream (Q21), and Oakley Creek downstream (Q24). All projected Polishing Pond outflow concentrations meet the MMER levels and the projected water quality downstream of the mixing zones in the Minago River and the Oakley Creek meets the CCME (2007) and Manitoba Tier III Freshwater guidelines levels.

2.14.2.3.2 Water Balance Modeling Results during Operations

Year 1 through Year 8 Operations

Estimated flowrates during Year 1 through Year 8 operations are listed in Table 2.14-12 and the corresponding water management plan is illustrated in Figure 2.14-3.

The Polishing Pond discharge to Minago River (Q33) in relation to the Minago River streamflow (Q34) will be 10% in May, 19% in the summer months (June to October) and 31% to 36% in the winter months (November to April). In absolute quantities, discharge to Minago River will range from 21,160 m^3/day to 90,035 m^3/day during Year 1 to Year 8 operations. The Polishing Pond discharge to Oakley Creek (Q36) in relation to the Oakley Creek streamflow (Q37) will be 0% in the winter months (Nov. to Apr.), 10% to 11% in May, and 31% in the summer months (June to October). In absolute quantities, discharge to Oakley Creek will range from 0 m^3/day to 37,715 m^3/day during operations.

Table 2.14-13 and Table 2.14-14 present projected parametric concentrations for the Polishing Pond outflow (Q30), Minago downstream (Q35), and Oakley Creek downstream (Q38) for Year 1 through 4 and Year 5 through 8, respectively. Additional results for Q26 (TWRMF Inflow), Q27 (TWRMF Decant), and Q29 (Polishing Pond Inflow) and detailed flow estimates are provided in Appendix 2.14. All Polishing Pond outflow concentrations are projected to meet the MMER levels and the projected water quality downstream of the mixing zones in the Minago River and the Oakley Creek meets the CCME (2007) and Manitoba Tier III Freshwater guidelines levels.

Table 2.14-10 Projected Flow Rates during Construction

FLOW		During Construction prior to Dredging		
		NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER
		Minago River Flow 0.8 m^3/s; Oakley Creek Flow at 0 m^3/s	Minago River Flow 10 m^3/s; Oakley Creek Flow at 4 m^3/s	Minago River Flow 1.9 m^3/s; Oakley Creek Flow at 0.5 m^3/s
		m^3/day	m^3/day	m^3/day
UNIT EVAPORATION	UNIT LAKE EVAPORATION	0	18	14
UNIT PPT (U-PPT)	UNIT PRECIPITATION	0	41	21
Q1	FLOW FROM DEWATERING WELLS	31,999	31,999	31,999
Q2	WELL WATER FOR PROCESSING	90	90	90
Q3	EXCESS WATER FROM DEWATERING WELLS	31,909	31,909	31,909
Q4	GROUNDWATER TO MILL CONSTRUCTION	0	0	0
Q5	GROUNDWATER TO WATER TREATMENT	90	90	90
Q6	GROUNDWATER TO FSP CONSTRUCTION	0	0	0
Q7	GROUNDWATER FOR FIRE FIGHTING	0	0	0
Q9	POTABLE WATER	90	90	90
Q10	WATER TREATMENT PLANT WASTE	0	0	0
Q11	SEWAGE & GREY WATER FROM CAMP AND OFFICES	90	90	90
Q12	SEWAGE & GREY WATER FROM ALL OTHER ON SITE SOURCES	0	0	0
Q13	FLOW FROM THE ODF TO THE ODF SETTLING POND	0	0	0
Q14	DISCHARGE TO THE OAKLEY CREEK BASIN	0	0	0
Q15	SEWAGE TREATMENT OUTFLOW	0	635	97
Q16	POLISHING POND INFLOW	31,909	97,392	32,006
Q-PPT on Polishing Pond	PPT ON POLISHING POND	0	3,049	1,602
Q-Evapo from Polishing Pond	EVAPO FROM POLISHING POND	0	1,355	1,063
Q17	POLISHING POND OUTFLOW	31,909	99,086	32,545
Q18	DISCHARGE PIPELINE	20,741	99,086	32,545
Q19	DISCHARGE TO MINAGO	20,741	69,360	22,782
Q20	MINAGO UPSTREAM	69,120	864,000	164,160
Q21	MINAGO DOWNSTREAM	89,861	933,360	186,942
Q22	DISCHARGE TO OAKLEY CREEK	0	29,726	9,764
Q23	OAKLEY CREEK UPSTREAM	0	345,600	43,200
Q24	OAKLEY CREEK DOWNSTREAM	0	375,326	52,964
FLOW RATIOS:				
Q19 / Q20	RATIO OF DISCHARGE TO MINAGO TO FLOW IN MINAGO	30%	8%	14%
Q22 / Q23	RATIO OF DISCHARGE TO OAKLEY CK TO FLOW IN OAKLEY CK	0%	9%	23%

Table 2.14-11 Projected Effluent Concentrations in Site Flows during Construction prior to Dredging

SCENARIO:			WATER QUALITY	ESTIMATED AVERAGE CONCENTRATION During Construction			REGULATIONS				
				Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Metal Mining Liquid Effluents (2002)		Manitoba Water Quality Standards, Objectives, and Guidelines (Williamson, 2002)		Canadian Water Quality Guidelines for the Protection of Aquatic Life
				NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER			TIER II Water Quality Objectives		
FLOW			PARAM.	(mg/L)	(mg/L)	(mg/L)	Monthly Mean	Grab Sample	assuming hardness = 150 mg/L $CaCO_3$	Freshwater	(CCME, 2007)
Q17	POLISHING POND OUTFLOW	Al	RC17	0.009	0.009	0.009				0.005 - 0.1	0.005
Q17	POLISHING POND OUTFLOW	Sb	RC17	0.00003	0.00004	0.00005					
Q17	POLISHING POND OUTFLOW	As	RC17	0.001	0.001	0.001	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q17	POLISHING POND OUTFLOW	Cd	RC17	0.00001	0.00001	0.00001			0.00302 [B]	Tier II	0.000017 or $10^{\{0.86[\log(hardness)]-3.2\}}$
Q17	POLISHING POND OUTFLOW	Cr	RC17	0.0010	0.0010	0.0010			0.10331 [C]	Tier II	
Q17	POLISHING POND OUTFLOW	Co	RC17	0.00008	0.00009	0.00010					
Q17	POLISHING POND OUTFLOW	Cu	RC17	0.0005	0.0005	0.0006	0.3	0.6	0.01266 [D]	Tier II	0.002
Q17	POLISHING POND OUTFLOW	Fe	RC17	0.005	0.006	0.006				0.3	0.3
Q17	POLISHING POND OUTFLOW	Pb	RC17	0.00003	0.00005	0.00006	0.2	0.4	0.0039 [E]	Tier II	0.001
Q17	POLISHING POND OUTFLOW	Mo	RC17	0.0007	0.0007	0.0007				0.073	
Q17	POLISHING POND OUTFLOW	Ni	RC17	0.001	0.001	0.001	0.5	1	0.07329 [F]	Tier II	0.025
Q17	POLISHING POND OUTFLOW	Se	RC17	0.0002	0.0002	0.0002				0.001	0.001
Q17	POLISHING POND OUTFLOW	Zn	RC17	0.005	0.005	0.005	0.5	1	0.16657 [G]	Tier II	0.03
Q21	MINAGO DOWNSTREAM	Al	RC21	0.011	0.012	0.012				0.005 - 0.1	0.005
Q21	MINAGO DOWNSTREAM	Sb	RC21	0.00004	0.00005	0.00005					
Q21	MINAGO DOWNSTREAM	As	RC21	0.0007	0.0006	0.0006	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q21	MINAGO DOWNSTREAM	Cd	RC21	0.000014	0.000016	0.000015			0.00302 [B]	Tier II	0.000017 or $10^{\{0.86[\log(hardness)]-3.2\}}$
Q21	MINAGO DOWNSTREAM	Cr	RC21	0.00041	0.00029	0.00033			0.10331 [C]	Tier II	
Q21	MINAGO DOWNSTREAM	Co	RC21	0.00006	0.00005	0.00006					
Q21	MINAGO DOWNSTREAM	Cu	RC21	0.001	0.001	0.001	0.3	0.6	0.01266 [D]	Tier II	0.002
Q21	MINAGO DOWNSTREAM	Fe	RC21	0.054	0.065	0.062				0.3	0.3
Q21	MINAGO DOWNSTREAM	Pb	RC21	0.00005	0.00006	0.00006	0.2	0.4	0.0039 [E]	Tier II	0.001
Q21	MINAGO DOWNSTREAM	Mo	RC21	0.00025	0.00017	0.00020				0.073	
Q21	MINAGO DOWNSTREAM	Ni	RC21	0.001	0.001	0.001	0.5	1	0.07329 [F]	Tier II	0.025
Q21	MINAGO DOWNSTREAM	Se	RC21	0.00023	0.00024	0.00024				0.001	0.001
Q21	MINAGO DOWNSTREAM	Zn	RC21	0.002	0.001	0.001	0.5	1	0.16657 [G]	Tier II	0.03

Table 2.14-11 (Cont.'d) Projected Effluent Concentrations in Site Flows during Construction prior to Dredging

SCENARIO:				ESTIMATED AVERAGE CONCENTRATION			REGULATIONS				
				During Construction (see Figure WB-1)			Metal Mining Liquid Effluents (2002)		Manitoba Water Quality Standards, Objectives, and Guidelines (Williamson, 2002)		Canadian Water Quality Guidelines for the Protection of Aquatic Life
				Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate			TIER II Water Quality Objectives		
			WATER QUALITY	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER					
FLOW			PARAM.	(mg/L)	(mg/L)	(mg/L)	Monthly Mean	Grab Sample	assuming hardness = 150 mg/L $CaCO_3$	Freshwater	(CCME, 2007)
Q24	OAKLEY CREEK DOWNSTREAM	Al	RC24	N/A	0.0031	0.0031				0.005 - 0.1	0.005
Q24	OAKLEY CREEK DOWNSTREAM	Sb	RC24	N/A	0.000035	0.000035					
Q24	OAKLEY CREEK DOWNSTREAM	As	RC24	N/A	0.0004	0.0004	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q24	OAKLEY CREEK DOWNSTREAM	Cd	RC24	N/A	0.000012	0.000012			0.00302 [B]	Tier II	0.000017 or $10^{(0.86[\log(hardness)]-3.2)}$
Q24	OAKLEY CREEK DOWNSTREAM	Cr	RC24	N/A	0.0003	0.0003			0.10331 [C]	Tier II	
Q24	OAKLEY CREEK DOWNSTREAM	Co	RC24	N/A	0.0000	0.0000					
Q24	OAKLEY CREEK DOWNSTREAM	Cu	RC24	N/A	0.0002	0.0002	0.3	0.6	0.01266 [D]	Tier II	0.002
Q24	OAKLEY CREEK DOWNSTREAM	Fe	RC24	N/A	0.0470	0.0470				0.3	0.3
Q24	OAKLEY CREEK DOWNSTREAM	Pb	RC24	N/A	0.0000	0.0000	0.2	0.4	0.0039 [E]	Tier II	0.001
Q24	OAKLEY CREEK DOWNSTREAM	Mo	RC24	N/A	0.0001	0.0001				0.073	
Q24	OAKLEY CREEK DOWNSTREAM	Ni	RC24	N/A	0.0003	0.0003	0.5	1	0.07329 [F]	Tier II	0.025
Q24	OAKLEY CREEK DOWNSTREAM	Se	RC24	N/A	0.0002	0.0002				0.001	0.001
Q24	OAKLEY CREEK DOWNSTREAM	Zn	RC24	N/A	0.0009	0.0009	0.5	1	0.16657 [G]	Tier II	0.03

Notes: **NA not applicable**
August 2008 groundwater chemistry was assumed in the modeling.

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.

Table 2.14-12 Projected Flow Rates during Year 1 through 8 Operations

		Year 1			Year 2			...	Year 7			Year 8		
		Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate		Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate
		NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER		NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER
	FLOW	m^3/day	m^3/day	m^3/day	m^3/day	m^3/day	m^3/day		m^3/day	m^3/day	m^3/day	m^3/day	m^3/day	m^3/day
UNIT EVAPORATION	UNIT LAKE EVAPORATION	0	18	14	0	18	14		0	18	14	0	18	14
UNIT PPT (U-PPT)	UNIT PRECIPITATION	0	41	21	0	41	21		0	41	21	0	41	21
Q1	FLOW FROM DEWATERING WELLS	31,999	31,999	31,999	31,999	31,999	31,999		31,999	31,999	31,999	31,999	31,999	31,999
Q2	WELL WATER FOR PROCESSING	5,724	5,724	5,724	5,724	5,724	5,724		5,724	5,724	5,724	5,724	5,724	5,724
Q3	EXCESS WATER FROM DEWATERING WELLS	26,276	26,276	26,276	26,276	26,276	26,276		26,276	26,276	26,276	26,276	26,276	26,276
Q4	GROUNDWATER TO OTHER OPERATIONS	1,440	1,440	1,440	1,440	1,440	1,440		1,440	1,440	1,440	1,440	1,440	1,440
Q5	GROUNDWATER TO WATER TREATMENT	96	96	96	96	96	96		96	96	96	96	96	96
Q6	GROUNDWATER TO FRAC SAND PLANT	4,188	4,188	4,188	4,188	4,188	4,188		4,188	4,188	4,188	4,188	4,188	4,188
Q7	GROUNDWATER FOR FIRE FIGHTING	0	0	0	0	0	0		0	0	0	0	0	0
Q8	POTABLE WATER	96	96	96	96	96	96		96	96	96	96	96	96
Q9	WATER TREATMENT PLANT WASTE	0	0	0	0	0	0		0	0	0	0	0	0
Q10	RECYCLE WATER FROM FPP	10,632	10,632	10,632	10,632	10,632	10,632		10,632	10,632	10,632	10,632	10,632	10,632
Q11	POTABLE WATER TO MILL	6	6	6	6	6	6		6	6	6	6	6	6
Q12	POTABLE WATER TO OTHER OPERATIONS	5	5	5	5	5	5		5	5	5	5	5	5
Q13	POTABLE WATER TO OFFICES & CAMP	72	72	72	72	72	72		72	72	72	72	72	72
Q14	POTABLE WATER TO FRAC SAND PLANT	12	12	12	12	12	12		12	12	12	12	12	12
Q15	FLOW FROM OPERATIONS TO MILL	1,440	1,440	1,440	1,440	1,440	1,440		1,440	1,440	1,440	1,440	1,440	1,440
Q16	SEWAGE & GREY WATER FROM CAMP AND OFFICES	72	72	72	72	72	72		72	72	72	72	72	72
Q17	SEWAGE & GREY WATER FROM ALL OTHER ON SITE SOURCES	24	24	24	24	24	24		24	24	24	24	24	24
Q19	FLOW FROM CONCENTRATE THICKENER IN MILL TO MILL	1,080	1,080	1,080	1,080	1,080	1,080		1,080	1,080	1,080	1,080	1,080	1,080
Q20	FLOW FROM MILL TO MILL THICKENER	32,928	32,928	32,928	32,928	32,928	32,928		32,928	32,928	32,928	32,928	32,928	32,928
Q21	RECYCLE WATER FROM MILL THICKENER	20,856	20,856	20,856	20,856	20,856	20,856		20,856	20,856	20,856	20,856	20,856	20,856
Q21x	ATERNATE FLOW FOR RECYCLE WATER FROM MILL THICKENER	0	0	0	0	0	0		0	0	0	0	0	0
Q22	MILL TAILINGS SLURRY	12,072	12,072	12,072	12,072	12,072	12,072		12,072	12,072	12,072	12,072	12,072	12,072
Q23	SEWAGE TREATMENT OUTFLOW	0	676	103	0	676	103		0	676	103	0	676	103
Q24	LIQ. WASTE FROM FSP	2,892	2,892	2,892	2,892	2,892	2,892		2,892	2,892	2,892	2,892	2,892	2,892
Q25	SLURRY FROM FSP	772	772	772	772	772	772		772	772	772	772	772	772
Q26	TWRMF INFLOW	15,736	16,412	15,839	15,736	16,412	15,839		15,736	16,412	15,839	15,736	16,412	15,839
Q - Liquid PPT on TWRMF	PPT on TWRMF	0	8,930	4,694	0	8,930	4,694		0	8,930	4,694	0	8,930	4,694
Q - Retained Water in Tailings Voids	Q - Retained Water in Tailings Voids	726	1,467	1,467	1,467	1,467	1,467		1,467	1,467	1,467	1,467	1,467	1,467
Q - TWRMF Supernatant	TWRMF Supernatant	15,010	55,342	23,084	20,372	55,342	23,084		20,372	55,342	23,084	20,372	55,342	23,084
Q27	TWRMF Decant	8,907	19,907	15,951	14,269	19,907	15,951		14,269	19,907	15,951	14,269	19,907	15,951
Q - Pit Dewatering	OPEN PIT DEWATERING	8,000	8,000	8,000	8,000	8,000	8,000		8,000	8,000	8,000	8,000	8,000	8,000
Q - Precipitation on Pit	Precipitation minus Sublimation on Open Pit	0	7,723	4,059	0	7,723	4,059		0	7,723	4,059	0	7,723	4,059
Q28	TOT. OPEN PIT DEWATERING	8,000	15,723	12,059	8,000	15,723	12,059		8,000	15,723	12,059	8,000	15,723	12,059
Q29	POLISHING POND INFLOW	43,183	128,057	54,285	48,545	137,557	54,285		48,545	123,043	54,285	48,545	121,643	54,285
Q - Precipitation on Polishing Pond	Precipitation minus Sublimation ON POLISHING POND	0	3,049	1,602	0	3,049	1,602		0	3,049	1,602	0	3,049	1,602
Q - Evaporation from Polishing Pond	EVAPORATION FROM POLISHING POND	0	1,355	1,063	0	1,355	1,063		0	1,355	1,063	0	1,355	1,063
Q30	POLISHING POND OUTFLOW	43,183	129,751	54,824	48,545	139,250	54,824		48,545	124,737	54,824	48,545	123,337	54,824
Q31	RECYCLE FROM FINAL POLISHING POND	10,632	10,632	10,632	10,632	10,632	10,632		10,632	10,632	10,632	10,632	10,632	10,632
Q32	DISCHARGE PIPELINE	21,158	119,119	44,192	24,643	128,618	44,192		24,643	114,105	44,192	24,643	112,705	44,192
Q33	DISCHARGE TO MINAGO	21,158	83,383	30,935	24,643	90,033	30,935		24,643	79,873	30,935	24,643	78,894	30,935
Q34	MINAGO UPSTREAM	69,120	864,000	164,160	69,120	864,000	164,160		69,120	864,000	164,160	69,120	864,000	164,160
Q35	MINAGO DOWNSTREAM	90,278	947,383	195,095	93,763	954,033	195,095		93,763	943,873	195,095	93,763	942,894	195,095
Q36	DISCHARGE TO OAKLEY CREEK	0	35,736	13,258	0	38,585	13,258		0	34,231	13,258	0	33,812	13,258
Q37	OAKLEY CREEK UPSTREAM	0	345,600	43,200	0	345,600	43,200		0	345,600	43,200	0	345,600	43,200
Q38	OAKLEY CREEK DOWNSTREAM	0	381,336	56,458	0	384,185	56,458		0	379,831	56,458	0	379,412	56,458
FLOW RATIOS:														
Q33 / Q34	RATIO OF DISCHARGE TO MINAGO TO FLOW IN MINAGO	31%	10%	19%	36%	10%	19%		36%	9%	19%	36%	9%	19%
Q36 / Q37	RATIO OF DISCHARGE TO OAKLEY CK TO FLOW IN OAKLEY CK	0%	10%	31%	0%	11%	31%		0%	10%	31%	0%	10%	31%

Note: A complete listing of projected flowrates during the Year 1 to Year 8 Operations are given in Appendix 2.14.

Table 2.14-13 Projected Effluent Concentrations in Site Flows during Year 1 through Year 4 Operations

SCENARIO:			ESTIMATED AVERAGE CONCENTRATION												REGULATIONS				
			Year 1			Year 2			Year 3			Year 4			Metal Mining Liquid Effluents (2002)		Manitoba Water Quality Standards, Objectives, and Guidelines (Williamson, 2002)		Canadian Water Quality Guidelines for the Protection of Aquatic Life (CCME, 2007)
			Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate					
		WATER QUALITY	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER			TIER II Water Quality Objectives		
FLOW		PARAM.	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	Monthly Mean	Grab Sample	assuming hardness = 150 mg/L $CaCO_3$	Freshwater	
Q30	POLISHING POND OUTFLOW	Al	0.119	0.139	0.154	0.169	0.169	0.156	0.171	0.174	0.160	0.174	0.181	0.165				0.005 - 0.1	0.005
Q30	POLISHING POND OUTFLOW	Sb	0.00092	0.00115	0.00130	0.00135	0.00139	0.00132	0.00136	0.00144	0.00135	0.00138	0.00149	0.00139					
Q30	POLISHING POND OUTFLOW	As	0.001	0.001	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q30	POLISHING POND OUTFLOW	Cd	0.00018	0.00021	0.00024	0.00026	0.00026	0.00024	0.00027	0.00027	0.00025	0.00028	0.00029	0.00027			0.00302 [B]	Tier II	0.000017 or $10^{\{0.86[\log(hardness)]-3.2}$
Q30	POLISHING POND OUTFLOW	Cr	0.0038	0.0043	0.0048	0.0051	0.0051	0.0048	0.0051	0.0052	0.0049	0.0052	0.0054	0.0050			0.10331 [C]	Tier II	
Q30	POLISHING POND OUTFLOW	Co	0.00245	0.00292	0.00324	0.00353	0.00353	0.00329	0.00355	0.00364	0.00336	0.00358	0.00375	0.00344					
Q30	POLISHING POND OUTFLOW	Cu	0.0084	0.0100	0.0110	0.0122	0.0122	0.0114	0.0124	0.01272	0.0118	0.0127	0.0133	0.0122	0.3	0.6	0.01266 [D]	Tier II	0.002
Q30	POLISHING POND OUTFLOW	Fe	0.527	0.628	0.696	0.762	0.761	0.704	0.765	0.781	0.717	0.769	0.804	0.733				0.3	0.3
Q30	POLISHING POND OUTFLOW	Pb	0.00141	0.00172	0.00193	0.00216	0.00220	0.00209	0.00233	0.00242	0.00228	0.00250	0.00266	0.00249	0.2	0.4	0.0039 [E]	Tier II	0.001
Q30	POLISHING POND OUTFLOW	Mo	0.0032	0.0037	0.0041	0.0045	0.0046	0.0043	0.0047	0.0049	0.0046	0.0049	0.0052	0.0049				0.073	
Q30	POLISHING POND OUTFLOW	Ni	0.120	0.142	0.158	0.174	0.173	0.160	0.175	0.178	0.164	0.177	0.184	0.168	0.5	1	0.07329 [F]	Tier II	0.025
Q30	POLISHING POND OUTFLOW	Se	0.0018	0.0022	0.0025	0.0026	0.0027	0.0026	0.0027	0.0028	0.0026	0.0027	0.0029	0.0027				0.001	0.001
Q30	POLISHING POND OUTFLOW	Zn	0.011	0.012	0.014	0.015	0.015	0.014	0.015	0.016	0.015	0.016	0.017	0.016	0.5	1	0.16657 [G]	Tier II	0.03
Q35	MINAGO DOWNSTREAM	Al	0.037	0.023	0.034	0.053	0.027	0.035	0.054	0.027	0.036	0.055	0.027	0.036				0.005 - 0.1	0.005
Q35	MINAGO DOWNSTREAM	Sb	0.00025	0.00015	0.00025	0.00039	0.00018	0.00025	0.00039	0.00018	0.00026	0.00040	0.00018	0.00026					
Q35	MINAGO DOWNSTREAM	As	0.0008	0.0007	0.0008	0.0009	0.0007	0.0008	0.0009	0.0007	0.0008	0.0009	0.0007	0.0008	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q35	MINAGO DOWNSTREAM	Cd	0.000054	0.000034	0.000052	0.000081	0.000040	0.000053	0.000083	0.000041	0.000054	0.000085	0.000041	0.000056			0.00302 [B]	Tier II	0.000017 or $10^{\{0.86[\log(hardness)]-3.2}$
Q35	MINAGO DOWNSTREAM	Cr	0.00107	0.00059	0.00095	0.00151	0.00069	0.00096	0.00152	0.00069	0.00097	0.00153	0.00070	0.00099			0.10331 [C]	Tier II	
Q35	MINAGO DOWNSTREAM	Co	0.00061	0.00030	0.00056	0.00096	0.00038	0.00056	0.00097	0.00038	0.00057	0.00098	0.00039	0.00059					
Q35	MINAGO DOWNSTREAM	Cu	0.002	0.001	0.002	0.004	0.002	0.002	0.004	0.002	0.002	0.004	0.002	0.002	0.3	0.6	0.01266 [D]	Tier II	0.002
Q35	MINAGO DOWNSTREAM	Fe	0.177	0.118	0.169	0.251	0.135	0.170	0.252	0.135	0.172	0.253	0.136	0.175				0.3	0.3
Q35	MINAGO DOWNSTREAM	Pb	0.00037	0.00020	0.00035	0.00061	0.00026	0.00038	0.00065	0.00028	0.00041	0.00070	0.00029	0.00044	0.2	0.4	0.0039 [E]	Tier II	0.001
Q35	MINAGO DOWNSTREAM	Mo	0.00085	0.00045	0.00076	0.00128	0.00055	0.00079	0.00134	0.00057	0.00083	0.00139	0.00059	0.00088				0.073	
Q35	MINAGO DOWNSTREAM	Ni	0.029	0.013	0.026	0.046	0.017	0.026	0.047	0.017	0.027	0.047	0.018	0.028	0.5	1	0.07329 [F]	Tier II	0.025
Q35	MINAGO DOWNSTREAM	Se	0.00062	0.00042	0.00061	0.00087	0.00048	0.00061	0.00088	0.00048	0.00062	0.00089	0.00048	0.00063				0.001	0.001
Q35	MINAGO DOWNSTREAM	Zn	0.003	0.002	0.003	0.005	0.002	0.003	0.005	0.002	0.003	0.005	0.002	0.003	0.5	1	0.16657 [G]	Tier II	0.03

Table 2.14-13 (Cont.'d) Projected Effluent Concentrations in Site Flows during Year 1 through Year 4 Operations

SCENARIO:			ESTIMATED AVERAGE CONCENTRATION												REGULATIONS				
			Year 1			Year 2			Year 3			Year 4			Metal Mining Liquid Effluents (2002)		Manitoba Water Quality Standards, Objectives, and Guidelines (Williamson, 2002)		Canadian Water Quality Guidelines for the Protection of Aquatic Life
			Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate			TIER II Water Quality Objectives		
FLOW		WATER QUALITY PARAM.	NOVEMBER TO APRIL (mg/L)	MAY (mg/L)	JUNE TO OCTOBER (mg/L)	NOVEMBER TO APRIL (mg/L)	MAY (mg/L)	JUNE TO OCTOBER (mg/L)	NOVEMBER TO APRIL (mg/L)	MAY (mg/L)	JUNE TO OCTOBER (mg/L)	NOVEMBER TO APRIL (mg/L)	MAY (mg/L)	JUNE TO OCTOBER (mg/L)	Monthly Mean	Grab Sample	assuming hardness = 150 mg/L $CaCO_3$	Freshwater	(CCME, 2007)
Q38	OAKLEY CREEK DOWNSTREAM	Al	N/A	0.0155	0.0381	N/A	0.0193	0.0388	N/A	0.0195	0.0397	N/A	0.0198	0.0408				0.005 - 0.1	0.005
Q38	OAKLEY CREEK DOWNSTREAM	Sb	N/A	0.000138	0.000330	N/A	0.000170	0.000336	N/A	0.000172	0.000344	N/A	0.000174	0.000353					
Q38	OAKLEY CREEK DOWNSTREAM	As	N/A	0.0005	0.0007	N/A	0.0005	0.0007	N/A	0.0005	0.0007	N/A	0.0005	0.0007	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q38	OAKLEY CREEK DOWNSTREAM	Cd	N/A	0.000031	0.000065	N/A	0.000038	0.000067	N/A	0.000038	0.000069	N/A	0.000039	0.000072			0.00302 [B]	Tier II	0.000017 or $10^{\{0.86[\log(\text{hardness})]-3.2}$
Q38	OAKLEY CREEK DOWNSTREAM	Cr	N/A	0.0007	0.0013	N/A	0.0008	0.0014	N/A	0.0008	0.0014	N/A	0.0008	0.0014			0.10331 [C]	Tier II	
Q38	OAKLEY CREEK DOWNSTREAM	Co	N/A	0.0003	0.0008	N/A	0.0004	0.0008	N/A	0.0004	0.0008	N/A	0.0004	0.0008					
Q38	OAKLEY CREEK DOWNSTREAM	Cu	N/A	0.0011	0.0027	N/A	0.0014	0.0028	N/A	0.0014	0.0029	N/A	0.0014	0.0030	0.3	0.6	0.01266 [D]	Tier II	0.002
Q38	OAKLEY CREEK DOWNSTREAM	Fe	N/A	0.1046	0.2021	N/A	0.1218	0.2040	N/A	0.1224	0.2071	N/A	0.1231	0.2108				0.3	0.3
Q38	OAKLEY CREEK DOWNSTREAM	Pb	N/A	0.0002	0.0005	N/A	0.0002	0.0005	N/A	0.0003	0.0006	N/A	0.0003	0.0006	0.2	0.4	0.0039 [E]	Tier II	0.001
Q38	OAKLEY CREEK DOWNSTREAM	Mo	N/A	0.0004	0.0010	N/A	0.0005	0.0011	N/A	0.0006	0.0012	N/A	0.0006	0.0012				0.073	
Q38	OAKLEY CREEK DOWNSTREAM	Ni	N/A	0.0135	0.0372	N/A	0.0176	0.0377	N/A	0.0178	0.0386	N/A	0.0180	0.0396	0.5	1	0.07329 [F]	Tier II	0.025
Q38	OAKLEY CREEK DOWNSTREAM	Se	N/A	0.0004	0.0008	N/A	0.0005	0.0008	N/A	0.0005	0.0008	N/A	0.0005	0.0008				0.001	0.001
Q38	OAKLEY CREEK DOWNSTREAM	Zn	N/A	0.0017	0.0036	N/A	0.0020	0.0038	N/A	0.0021	0.0040	N/A	0.0022	0.0043	0.5	1	0.16657 [G]	Tier II	0.03

Notes: **N/A not applicable**

August 2008 groundwater chemistry was assumed in the modeling.

A Arsenic li 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 [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 Chromiun 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 li [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 limit [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 limi [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 limit: [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.

Table 2.14-14 Projected Effluent Concentrations in Site Flows during Year 5 through Year 8 Operations

SCENARIO:			ESTIMATED AVERAGE CONCENTRATION												REGULATIONS				
			Year 5			Year 6			Year 7			Year 8			Metal Mining Liquid Effluents (2002)		Manitoba Water Quality Standards, Objectives, and Guidelines (Williamson, 2002)		Canadian Water Quality Guideline for the Protection of Aquatic Life
			Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate					
		WATER QUALITY	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER			TIER II Water Quality Objectives		
FLOW		PARAM.	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	Monthly Mean	Grab Sample	assuming hardness = 150 mg/L $CaCO_3$	Freshwater	(CCME, 2007)
Q30	POLISHING POND OUTFLOW	Al	0.176	0.188	0.170	0.179	0.196	0.177	0.182	0.206	0.184	0.185	0.212	0.189				0.005 - 0.1	0.005
Q30	POLISHING POND OUTFLOW	Sb	0.00140	0.00155	0.00143	0.00142	0.00162	0.00149	0.00145	0.00170	0.00155	0.00147	0.00175	0.00159					
Q30	POLISHING POND OUTFLOW	As	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q30	POLISHING POND OUTFLOW	Cd	0.00029	0.00031	0.00028	0.00030	0.00033	0.00030	0.00031	0.00035	0.00031	0.00031	0.00036	0.00033			0.00302 [B]	Tier II	0.000017 or $10^{\{0.86[\log(hardness)]-3.}$
Q30	POLISHING POND OUTFLOW	Cr	0.0052	0.0056	0.0051	0.0053	0.0058	0.0053	0.0053	0.0060	0.0055	0.0054	0.0062	0.0056			0.10331 [C]	Tier II	
Q30	POLISHING POND OUTFLOW	Co	0.00361	0.00389	0.00353	0.00365	0.00404	0.00365	0.00370	0.00421	0.00379	0.00374	0.00433	0.00387					
Q30	POLISHING POND OUTFLOW	Cu	0.0130	0.0140	0.0127	0.0133	0.0147	0.0133	0.0137	0.0156	0.0140	0.0140	0.0161	0.0144	0.3	0.6	0.01266 [D]	Tier II	0.002
Q30	POLISHING POND OUTFLOW	Fe	0.774	0.830	0.752	0.781	0.860	0.774	0.788	0.896	0.802	0.797	0.920	0.820				0.3	0.3
Q30	POLISHING POND OUTFLOW	Pb	0.00269	0.00293	0.00271	0.00287	0.00321	0.00295	0.00306	0.00353	0.00322	0.00320	0.00375	0.00338	0.2	0.4	0.0039 [E]	Tier II	0.001
Q30	POLISHING POND OUTFLOW	Mo	0.0052	0.0056	0.0052	0.0054	0.0060	0.0055	0.0056	0.0064	0.0059	0.0058	0.0068	0.0061				0.073	
Q30	POLISHING POND OUTFLOW	Ni	0.179	0.191	0.173	0.181	0.199	0.179	0.183	0.208	0.186	0.186	0.214	0.191	0.5	1	0.07329 [F]	Tier II	0.025
Q30	POLISHING POND OUTFLOW	Se	0.0027	0.0030	0.0028	0.0028	0.0031	0.0029	0.0028	0.0033	0.0030	0.0029	0.0034	0.0031				0.001	0.001
Q30	POLISHING POND OUTFLOW	Zn	0.017	0.018	0.017	0.018	0.020	0.018	0.019	0.022	0.020	0.020	0.023	0.021	0.5	1	0.16657 [G]	Tier II	0.03
Q35	MINAGO DOWNSTREAM	Al	0.055	0.028	0.037	0.056	0.028	0.038	0.057	0.028	0.039	0.057	0.029	0.040				0.005 - 0.1	0.005
Q35	MINAGO DOWNSTREAM	Sb	0.00040	0.00018	0.00027	0.00041	0.00019	0.00028	0.00042	0.00019	0.00029	0.00042	0.00019	0.00029					
Q35	MINAGO DOWNSTREAM	As	0.0009	0.0007	0.0008	0.0010	0.0007	0.0008	0.0010	0.0007	0.0008	0.0010	0.0008	0.0009	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q35	MINAGO DOWNSTREAM	Cd	0.000088	0.000042	0.000058	0.000090	0.000044	0.000061	0.000093	0.000045	0.000064	0.000095	0.000046	0.000066			0.00302 [B]	Tier II	0.000017 or $10^{\{0.86[\log(hardness)]-3.2}$
Q35	MINAGO DOWNSTREAM	Cr	0.00154	0.00070	0.00101	0.00155	0.00071	0.00103	0.00157	0.00072	0.00106	0.00159	0.00073	0.00108			0.10331 [C]	Tier II	
Q35	MINAGO DOWNSTREAM	Co	0.00099	0.00039	0.00060	0.00100	0.00040	0.00062	0.00101	0.00040	0.00064	0.00102	0.00041	0.00066					
Q35	MINAGO DOWNSTREAM	Cu	0.004	0.002	0.003	0.004	0.002	0.003	0.004	0.002	0.003	0.004	0.002	0.003	0.3	0.6	0.01266 [D]	Tier II	0.002
Q35	MINAGO DOWNSTREAM	Fe	0.255	0.137	0.177	0.256	0.138	0.181	0.258	0.139	0.186	0.260	0.140	0.188				0.3	0.3
Q35	MINAGO DOWNSTREAM	Pb	0.00075	0.00031	0.00048	0.00080	0.00033	0.00052	0.00085	0.00035	0.00056	0.00088	0.00037	0.00058	0.2	0.4	0.0039 [E]	Tier II	0.001
Q35	MINAGO DOWNSTREAM	Mo	0.00145	0.00061	0.00092	0.00151	0.00063	0.00098	0.00158	0.00066	0.00104	0.00163	0.00068	0.00107				0.073	
Q35	MINAGO DOWNSTREAM	Ni	0.048	0.018	0.028	0.048	0.018	0.029	0.049	0.019	0.030	0.050	0.019	0.031	0.5	1	0.07329 [F]	Tier II	0.025
Q35	MINAGO DOWNSTREAM	Se	0.00090	0.00049	0.00065	0.00091	0.00049	0.00066	0.00092	0.00050	0.00068	0.00093	0.00051	0.00069				0.001	0.001
Q35	MINAGO DOWNSTREAM	Zn	0.005	0.003	0.004	0.005	0.003	0.004	0.006	0.003	0.004	0.006	0.003	0.004	0.5	1	0.16657 [G]	Tier II	0.03

Table 2.14-14 (Cont.'d) Projected Effluent Concentrations in Site Flows during Year 5 through Year 8 Operations

SCENARIO:		WATER QUALITY	ESTIMATED AVERAGE CONCENTRATION												REGULATIONS				
			Year 5			Year 6			Year 7			Year 8			Metal Mining Liquid Effluents (2002)		Manitoba Water Quality Standards, Objectives, and Guidelines (Williamson, 2002)		Canadian Water Quality Guidelines for the Protection of Aquatic Life
			Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate			TIER II Water Quality Objectives		
			NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER					
FLOW		PARAM.	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	Monthly Mean	Grab Sample	assuming hardness = 150 mg/L $CaCO_3$	Freshwater	(CCME, 2007)
Q38	OAKLEY CREEK DOWNSTREAM	Al	N/A	0.0201	0.0421	N/A	0.0205	0.0436	N/A	0.0210	0.0453	N/A	0.0213	0.0464				0.005 - 0.1	0.005
Q38	OAKLEY CREEK DOWNSTREAM	Sb	N/A	0.000177	0.000363	N/A	0.000180	0.000375	N/A	0.000184	0.000389	N/A	0.000187	0.000398					
Q38	OAKLEY CREEK DOWNSTREAM	As	N/A	0.0005	0.0007	N/A	0.0005	0.0008	N/A	0.0005	0.0008	N/A	0.0006	0.0008	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q38	OAKLEY CREEK DOWNSTREAM	Cd	N/A	0.000040	0.000075	N/A	0.000042	0.000079	N/A	0.000043	0.000083	N/A	0.000044	0.000086			0.00302 [B]	Tier II	0.000017 or $10^{\{0.86[\log(hardness)]-3.2}$
Q38	OAKLEY CREEK DOWNSTREAM	Cr	N/A	0.0008	0.0014	N/A	0.0008	0.0015	N/A	0.0008	0.0015	N/A	0.0008	0.0015			0.10331 [C]	Tier II	
Q38	OAKLEY CREEK DOWNSTREAM	Co	N/A	0.0004	0.0009	N/A	0.0004	0.0009	N/A	0.0004	0.0009	N/A	0.0004	0.0009					
Q38	OAKLEY CREEK DOWNSTREAM	Cu	N/A	0.0015	0.0031	N/A	0.0015	0.0032	N/A	0.0015	0.0034	N/A	0.0016	0.0035	0.3	0.6	0.01266 [D]	Tier II	0.002
Q38	OAKLEY CREEK DOWNSTREAM	Fe	N/A	0.1240	0.2151	N/A	0.1252	0.2204	N/A	0.1267	0.2270	N/A	0.1280	0.2312				0.3	0.3
Q38	OAKLEY CREEK DOWNSTREAM	Pb	N/A	0.0003	0.0007	N/A	0.0003	0.0007	N/A	0.0003	0.0008	N/A	0.0004	0.0008	0.2	0.4	0.0039 [E]	Tier II	0.001
Q38	OAKLEY CREEK DOWNSTREAM	Mo	N/A	0.0006	0.0013	N/A	0.0006	0.0014	N/A	0.0007	0.0015	N/A	0.0007	0.0015				0.073	
Q38	OAKLEY CREEK DOWNSTREAM	Ni	N/A	0.0182	0.0408	N/A	0.0186	0.0422	N/A	0.0190	0.0439	N/A	0.0193	0.0449	0.5	1	0.07329 [F]	Tier II	0.025
Q38	OAKLEY CREEK DOWNSTREAM	Se	N/A	0.0005	0.0008	N/A	0.0005	0.0009	N/A	0.0005	0.0009	N/A	0.0005	0.0009				0.001	0.001
Q38	OAKLEY CREEK DOWNSTREAM	Zn	N/A	0.0023	0.0045	N/A	0.0024	0.0048	N/A	0.0025	0.0051	N/A	0.0026	0.0053	0.5	1	0.16657 [G]	Tier II	0.03

Notes: **N/A not applicable**

August 2008 groundwater chemistry was assumed in the modeling.

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.

The projected outflow from the Polishing Pond meets MMER requirements at all times. Projected results range from 0.17 to 0.21 mg/L for Al, from 0.013 to 0.016 mg/L for Cu, from 0.75 to 0.92 mg/L for Fe, from 0.003 to 0.004 mg/L for Pb, from 0.17 to 0.21 mg/L for Ni, and from 0.003 to 0.003 mg/L for Se.

Year 9 and Year 10 Operations

Estimated flowrates during Year 9 and Year 10 are listed in Table 2.14-15 and the corresponding water management plan is illustrated in Figure 2.14-4.

Year 9

The Polishing Pond discharge to Minago River (Q33) in relation to the Minago River streamflow (Q34) will be 1% in May, 4% in the summer months (June to October) and 5% in the winter months (November to April). In absolute quantities, discharge to Minago River will range from 3,665 m^3/day to 10,670 m^3/day during Year 9 operations. The Polishing Pond discharge to Oakley Creek (Q36) in relation to the Oakley Creek streamflow (Q37) will be 0% year round.

Table 2.14-16 presents projected parametric concentrations for the Polishing Pond outflow (Q30), Minago downstream (Q35), and Oakley Creek downstream (Q38). Additional results for Q26 (TWRMF Inflow), Q27 (TWRMF Decant), and Q29 (Polishing Pond Inflow) are provided in Appendix 2.14. All Polishing Pond outflow concentrations are projected to meet the MMER levels and the projected water quality downstream of the mixing zones in the Minago River and the Oakley Creek meets the CCME (2007) and Manitoba Tier III Freshwater guidelines levels.

Year 10

The Polishing Pond discharge to Minago River (Q33) in relation to the Minago River streamflow (Q34) will be 0% year round. The Polishing Pond discharge to Oakley Creek (Q36) in relation to the Oakley Creek streamflow (Q37) will be 9% in May, 14% in the summer months (June to October) and 0% in the winter months (November to April).

Table 2.14-16 presents projected parametric concentrations for the Polishing Pond outflow (Q30), Minago downstream (Q35), and Oakley Creek downstream (Q38). Additional results for Q26 (TWRMF Inflow), Q27 (TWRMF Decant), and Q29 (Polishing Pond Inflow) are provided in Appendix 2.14. All Polishing Pond outflow concentrations are projected to meet the MMER levels and the projected water quality downstream of the mixing zones in the Minago River and the Oakley Creek meets the CCME (2007) and Manitoba Tier III Freshwater guidelines levels.

Table 2.14-15 Projected Flow Rates during Year 9 and Year 10 Operations

		Year 9			Year 10		
		Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate
		NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER
	FLOW	m^3/day	m^3/day	m^3/day	m^3/day	m^3/day	m^3/day
UNIT EVAPORATION	UNIT LAKE EVAPORATION	0	18	14	0	18	14
UNIT PPT (U-PPT)	UNIT PRECIPITATION	0	41	21	0	41	21
Q1	FLOW FROM DEWATERING WELLS	4,236	4,236	4,236	4,236	4,236	4,236
Q2	WELL WATER FOR PROCESSING	4,236	4,236	4,236	4,236	4,236	4,236
Q3	EXCESS WATER FROM DEWATERING WELLS	0	0	0	0	0	0
Q4	GROUNDWATER TO OTHER OPERATIONS	0	0	0	0	0	0
Q5	GROUNDWATER TO WATER TREATMENT	48	48	48	48	48	48
Q6	GROUNDWATER TO FRAC SAND PLANT	4,188	4,188	4,188	4,188	4,188	4,188
Q7	GROUNDWATER FOR FIRE FIGHTING	0	0	0	0	0	0
Q8	POTABLE WATER	48	48	48	48	48	48
Q9	WATER TREATMENT PLANT WASTE	0	0	0	0	0	0
Q10	RECYCLE WATER FROM FPP	0	0	0	0	0	0
Q11	POTABLE WATER TO MILL	0	0	0	0	0	0
Q12	POTABLE WATER TO OTHER OPERATIONS	0	0	0	0	0	0
Q13	POTABLE WATER TO OFFICES & CAMP	36	36	36	36	36	36
Q14	POTABLE WATER TO FRAC SAND PLANT	12	12	12	12	12	12
Q15	FLOW FROM OPERATIONS TO MILL	0	0	0	0	0	0
Q16	SEWAGE & GREY WATER FROM CAMP AND OFFICES	36	36	36	36	36	36
Q17	SEWAGE & GREY WATER FROM ALL OTHER ON SITE SOURCES	12	12	12	12	12	12
Q19	FLOW FROM CONCENTRATE THICKENER IN MILL TO MILL	0	0	0	0	0	0
Q20	FLOW FROM MILL TO MILL THICKENER	0	0	0	0	0	0
Q21	RECYCLE WATER FROM MILL THICKENER	0	0	0	0	0	0
Q21x	ATERNATE FLOW FOR RECYCLE WATER FROM MILL THICKENER	0	0	0	0	0	0
Q22	MILL TAILINGS SLURRY	0	0	0	0	0	0
Q23	SEWAGE TREATMENT OUTFLOW	0	349	55	0	349	55
Q24	LIQ. WASTE FROM FSP	2,892	2,892	2,892	2,892	2,892	2,892
Q25	SLURRY FROM FSP	772	772	772	772	772	772
Q26	TWRMF INFLOW	3,664	4,013	3,719	3,664	4,013	3,719
Q - Liquid PPT on TWRMF	PPT on TWRMF	0	8,930	4,694	0	8,930	4,694
Q - Retained Water in Tailings Voids	Q - Retained Water in Tailings Voids	0	0	0	0	0	0
Q - TWRMF Supernatant	TWRMF Supernatant	9,766	44,410	12,430	9,766	44,410	12,430
Q27	TWRMF Decant	3,664	8,975	5,297	3,664	8,975	5,297
Q - Pit Dewatering	OPEN PIT DEWATERING	0	0	0	0	0	0
Q - Precipitation on Pit	Precipitation minus Sublimation on Open Pit	0	7,723	4,059	0	7,723	4,059
Q28	TOT. OPEN PIT DEWATERING	0	0	0	0	0	0
Q29	POLISHING POND INFLOW	3,664	8,975	5,297	3,664	30,247	5,297
Q - Precipitation on Polishing Pond	Precipitation minus Sublimation ON POLISHING POND	0	3,049	1,602	0	3,049	1,602
Q - Evaporation from Polishing Pond	EVAPORATION FROM POLISHING POND	0	1,355	1,063	0	1,355	1,063
Q30	POLISHING POND OUTFLOW	3,664	10,668	5,836	3,664	31,941	5,836
Q31	RECYCLE FROM FINAL POLISHING POND	0	0	0	0	0	0
Q32	DISCHARGE PIPELINE	3,664	10,668	5,836	0	31,941	5,836
Q33	DISCHARGE TO MINAGO	3,664	10,668	5,836	0	0	0
Q34	MINAGO UPSTREAM	69,120	864,000	164,160	69,120	864,000	164,160
Q35	MINAGO DOWNSTREAM	72,784	874,668	169,996	69,120	864,000	164,160
Q36	DISCHARGE TO OAKLEY CREEK	0	0	0	0	31,941	5,836
Q37	OAKLEY CREEK UPSTREAM	0	345,600	43,200	0	345,600	43,200
Q38	OAKLEY CREEK DOWNSTREAM	0	345,600	43,200	0	377,541	49,036
FLOW RATIOS:							
Q33 / Q34	RATIO OF DISCHARGE TO MINAGO TO FLOW IN MINAGO	5%	1%	4%	0%	0%	0%
Q36 / Q37	RATIO OF DISCHARGE TO OAKLEY CK TO FLOW IN OAKLEY CK	0%	0%	0%	0%	9%	14%

Table 2.14-16 Projected Effluent Concentrations in Site Flows Year 9 and Year 10 Operations

SCENARIO:			ESTIMATED AVERAGE CONCENTRATION						REGULATIONS				
			Year 9			Year 10			(2002)		TIER II Water Quality Objectives		the Protection of Aquatic Life
			Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate					
		WATER QUALITY	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	Monthly Mean	Grab Sample	assuming hardness = 150 mg/L $CaCO_3$	Freshwater	(CCME, 2007)
FLOW		PARAM.	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)					
Q30	POLISHING POND OUTFLOW	Al	0.448	0.313	0.235	0.215	0.196	0.138				0.005 - 0.1	0.005
Q30	POLISHING POND OUTFLOW	Sb	0.00422	0.00324	0.00291	0.00289	0.00271	0.00236					
Q30	POLISHING POND OUTFLOW	As	0.006	0.005	0.005	0.006	0.006	0.005	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q30	POLISHING POND OUTFLOW	Cd	0.00099	0.00073	0.00068	0.00074	0.00067	0.00056			0.00302 [B]	Tier II	0.000017 or
Q30	POLISHING POND OUTFLOW	Cr	0.0113	0.0082	0.0061	0.0048	0.0045	0.0034			0.10331 [C]	Tier II	
Q30	POLISHING POND OUTFLOW	Co	0.00901	0.00635	0.00469	0.00406	0.00373	0.00265					
Q30	POLISHING POND OUTFLOW	Cu	0.0393	0.0286	0.0247	0.0261	0.0240	0.0192	0.3	0.6	0.01266 [D]	Tier II	0.002
Q30	POLISHING POND OUTFLOW	Fe	1.772	1.208	0.779	0.555	0.500	0.273				0.3	0.3
Q30	POLISHING POND OUTFLOW	Pb	0.01304	0.01008	0.01070	0.01318	0.01218	0.01076	0.2	0.4	0.0039 [E]	Tier II	0.001
Q30	POLISHING POND OUTFLOW	Mo	0.0249	0.0196	0.0218	0.0278	0.0257	0.0230				0.073	
Q30	POLISHING POND OUTFLOW	Ni	0.447	0.308	0.219	0.190	0.172	0.113	0.5	1	0.07329 [F]	Tier II	0.025
Q30	POLISHING POND OUTFLOW	Se	0.0070	0.0054	0.0044	0.0038	0.0036	0.0031				0.001	0.001
Q30	POLISHING POND OUTFLOW	Zn	0.068	0.054	0.058	0.071	0.066	0.059	0.5	1	0.16657 [G]	Tier II	0.03
Q35	MINAGO DOWNSTREAM	Al	0.034	0.016	0.020	N/A	N/A	N/A				0.005 - 0.1	0.005
Q35	MINAGO DOWNSTREAM	Sb	0.00026	0.00009	0.00015	N/A	N/A	N/A					
Q35	MINAGO DOWNSTREAM	As	0.0009	0.0007	0.0008	N/A	N/A	N/A	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q35	MINAGO DOWNSTREAM	Cd	0.000066	0.000025	0.000039	N/A	N/A	N/A			0.00302 [B]	Tier II	0.000017 or $10^{\{0.86[\log(hardness)]-3.2\}}$
Q35	MINAGO DOWNSTREAM	Cr	0.00078	0.00033	0.00043	N/A	N/A	N/A			0.10331 [C]	Tier II	
Q35	MINAGO DOWNSTREAM	Co	0.00050	0.00013	0.00021	N/A	N/A	N/A					
Q35	MINAGO DOWNSTREAM	Cu	0.003	0.001	0.001	N/A	N/A	N/A	0.3	0.6	0.01266 [D]	Tier II	0.002
Q35	MINAGO DOWNSTREAM	Fe	0.155	0.083	0.094	N/A	N/A	N/A				0.3	0.3
Q35	MINAGO DOWNSTREAM	Pb	0.00071	0.00018	0.00042	N/A	N/A	N/A	0.2	0.4	0.0039 [E]	Tier II	0.001
Q35	MINAGO DOWNSTREAM	Mo	0.00138	0.00036	0.00087	N/A	N/A	N/A				0.073	
Q35	MINAGO DOWNSTREAM	Ni	0.024	0.005	0.009	N/A	N/A	N/A	0.5	1	0.07329 [F]	Tier II	0.025
Q35	MINAGO DOWNSTREAM	Se	0.00059	0.00031	0.00039	N/A	N/A	N/A				0.001	0.001
Q35	MINAGO DOWNSTREAM	Zn	0.004	0.002	0.003	N/A	N/A	N/A	0.5	1	0.16657 [G]	Tier II	0.03

Table 2.14-16 (Cont.'d) Projected Effluent Concentrations in Site Flows during Year 9 and Year 10 Operations

SCENARIO: FLOW		WATER QUALITY PARAM.	ESTIMATED AVERAGE CONCENTRATION: Year 9, Tailings only; max.tailings leaching rate, NOVEMBER TO APRIL (mg/L)	Year 9, Tailings only; max.tailings leaching rate, MAY (mg/L)	Year 9, Tailings only; max.tailings leaching rate, JUNE TO OCTOBER (mg/L)	Year 10, Tailings only; max.tailings leaching rate, NOVEMBER TO APRIL (mg/L)	Year 10, Tailings only; max.tailings leaching rate, MAY (mg/L)	Year 10, Tailings only; max.tailings leaching rate, JUNE TO OCTOBER (mg/L)	REGULATIONS: (2002) Monthly Mean	(2002) Grab Sample	TIER II Water Quality Objectives assuming hardness = 150 mg/L $CaCO_3$	Freshwater	the Protection of Aquatic Life (CCME, 2007)
Q38	OAKLEY CREEK DOWNSTREAM	Al	N/A	N/A	N/A	N/A	0.0190	0.0188				0.005 - 0.1	0.005
Q38	OAKLEY CREEK DOWNSTREAM	Sb	N/A	N/A	N/A	N/A	0.000261	0.000311					
Q38	OAKLEY CREEK DOWNSTREAM	As	N/A	N/A	N/A	N/A	0.0008	0.0009	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q38	OAKLEY CREEK DOWNSTREAM	Cd	N/A	N/A	N/A	N/A	0.000069	0.000077			0.00302 [B]	Tier II	0.000017 or $10^{\{0.86[\log(hardness)]-3.2\}}$
Q38	OAKLEY CREEK DOWNSTREAM	Cr	N/A	N/A	N/A	N/A	0.0006	0.0007			0.10331 [C]	Tier II	
Q38	OAKLEY CREEK DOWNSTREAM	Co	N/A	N/A	N/A	N/A	0.0003	0.0003					
Q38	OAKLEY CREEK DOWNSTREAM	Cu	N/A	N/A	N/A	N/A	0.0022	0.0024	0.3	0.6	0.01266 [D]	Tier II	0.002
Q38	OAKLEY CREEK DOWNSTREAM	Fe	N/A	N/A	N/A	N/A	0.0885	0.0769				0.3	0.3
Q38	OAKLEY CREEK DOWNSTREAM	Pb	N/A	N/A	N/A	N/A	0.0010	0.0013	0.2	0.4	0.0039 [E]	Tier II	0.001
Q38	OAKLEY CREEK DOWNSTREAM	Mo	N/A	N/A	N/A	N/A	0.0023	0.0028				0.073	
Q38	OAKLEY CREEK DOWNSTREAM	Ni	N/A	N/A	N/A	N/A	0.0148	0.0136	0.5	1	0.07329 [F]	Tier II	0.025
Q38	OAKLEY CREEK DOWNSTREAM	Se	N/A	N/A	N/A	N/A	0.0005	0.0006				0.001	0.001
Q38	OAKLEY CREEK DOWNSTREAM	Zn	N/A	N/A	N/A	N/A	0.0061	0.0076	0.5	1	0.16657 [G]	Tier II	0.03

Notes: **N/A not applicable**

August 2008 groundwater chemistry was assumed in the modeling.

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 limit [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 limi 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.

2.14.2.3.3 Water Balance Results during Closure

Estimated flowrates during the first and second stages of the closure period are listed in Table 2.14-17. The water balance during the first stage of Closure is illustrated in Figure 2.14-6 and the second stage of Closure is illustrated in Figure 2.14-7.

During the first stage of Closure, a water cover will be installed on top of the TWRMF and no discharges to the receiving environment will occur from the TWRMF nor from the pipeline discharge system. After closure, water from the Polishing Pond will be discharged into a cross-ditch to report to the Oakley Creek. The major cross-ditch will report to the ditch at Highway 6 and to the Oakley Creek through the low lying marsh on the east side of Highway 6.

The Polishing Pond discharge to Minago River (Q33) in relation to the Minago River streamflow (Q34) will be 0% during the second stage of Closure (after the installation of a water cover on top of the tailings).

The Polishing Pond discharge to Oakley Creek (Q36) in relation to the Oakley Creek streamflow (Q37) will be 0% in the winter months (Nov. to Apr.), 2% in May, and 5% in the summer months (June to October). In absolute quantities, discharge to Oakley Creek will range from 0 m^3/day to 5,500 m^3/day during the second stage of Closure.

Table 2.14-18 presents projected parametric concentrations during the two stages of Closure for the Polishing Pond outflow (Q30), Minago downstream (Q35), and Oakley Creek downstream (Q38). Additional results for Q26 (TWRMF Inflow), Q27 (TWRMF Decant), Q29 (Polishing Pond Inflow) are given in Appendix 2.14.

During the first and second stages of Closure, the projected outflow from the Polishing Pond will meet MMER requirements at all times. During both stages of Closure, the projected water quality in Minago River and Oakley Creek downstream of the mixing zones meets the Manitoba Freshwater guidelines for the protection of aquatic life for all parameters.

2.14.2.3.4 Water Balance Results during Post Closure

During the Post Closure period, all discharge pipeline systems to Minago River and Oakley Creek will have been dismantled and excess water from the TWRMF (Q27 = TWRMF Decant) will be discharged via a spillway to the Polishing Pond for subsequent discharge to the receiving environment – the Oakley Creek basin and ultimately Oakley Creek. The active and inactive water balance components during the Post Closure period are illustrated in Figure 2.14-8.

Projected flowrates during the post closure period are listed in Table 2.14-19. Projected Polishing Pond outflow rates range from 0 m^3/day in the winter months (Nov. to Apr.) to 2,117 m^3/day in the period from June to October to 5,375 m^3/day in May.

Table 2.14-17 Projected Concentrations in Flows around the Minago Site during Closure

			Year 11			Year 12		
			Closure (Stage 1)	Closure (Stage 2)	Closure (Stage 2)	Closure (Stage 2)		
			Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate
			NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER
		FLOW	m^3/day	m^3/day	m^3/day	m^3/day	m^3/day	m^3/day
UNIT EVAPORATION		UNIT LAKE EVAPORATION	0	18	14	0	18	14
UNIT PPT (U-PPT)		UNIT PRECIPITATION	0	41	21	0	41	21
Q1		FLOW FROM DEWATERING WELLS	12,000	0	0	0	0	0
Q2		WELL WATER FOR PROCESSING	15	15	15	15	15	15
Q3		EXCESS WATER FROM DEWATERING WELLS	11,985	0	0	0	0	0
Q4		GROUNDWATER TO OTHER OPERATIONS	0	0	0	0	0	0
Q5		GROUNDWATER TO WATER TREATMENT	15	15	15	15	15	15
Q6		GROUNDWATER TO FRAC SAND PLANT	0	0	0	0	0	0
Q7		GROUNDWATER FOR FIRE FIGHTING	0	0	0	0	0	0
Q8		POTABLE WATER	15	15	15	15	15	15
Q9		WATER TREATMENT PLANT WASTE	0	0	0	0	0	0
Q10		RECYCLE WATER FROM FPP	0	0	0	0	0	0
Q11		POTABLE WATER TO MILL	0	0	0	0	0	0
Q12		POTABLE WATER TO OTHER OPERATIONS	0	0	0	0	0	0
Q13		POTABLE WATER TO OFFICES & CAMP	15	15	15	15	15	15
Q14		POTABLE WATER TO FRAC SAND PLANT	0	0	0	0	0	0
Q15		FLOW FROM OPERATIONS TO MILL	0	0	0	0	0	0
Q16		SEWAGE & GREY WATER FROM CAMP AND OFFICES	15	15	15	15	15	15
Q17		SEWAGE & GREY WATER FROM ALL OTHER ON SITE SOURCES	0	0	0	0	0	0
Q19		FLOW FROM CONCENTRATE THICKENER IN MILL TO MILL	0	0	0	0	0	0
Q20		FLOW FROM MILL TO MILL THICKENER	0	0	0	0	0	0
Q21		RECYCLE WATER FROM MILL THICKENER	0	0	0	0	0	0
Q21x		ATERNATE FLOW FOR RECYCLE WATER FROM MILL THICKENER	0	0	0	0	0	0
Q22		MILL TAILINGS SLURRY	0	0	0	0	0	0
Q23		SEWAGE TREATMENT OUTFLOW	0	125	22	0	125	22
Q24		LIQ. WASTE FROM FSP	0	0	0	0	0	0
Q25		SLURRY FROM FSP	0	0	0	0	0	0
Q26		TWRMF INFLOW	11,985	125	22	0	125	22
	Q - Liquid PPT on TWRMF	PPT on TWRMF	0	8,930	4,694	0	8,930	4,694
	Q - Retained Water in Tailings Voids	Q - Retained Water in Tailings Voids	0	0	0	0	0	0
	Q - TWRMF Supernatant	TWRMF Supernatant	18,088	110,112	23,000	18,308	110,112	23,000
Q27		TWRMF Decant	0	3,806	1,601	0	3,806	1,601
	Q - Pit Dewatering	OPEN PIT DEWATERING	0	0	0	0	0	0
	Q - Precipitation on Pit	Precipitation minus Sublimation on Open Pit	0	7,723	4,059	0	7,723	4,059
Q28		TOT. OPEN PIT DEWATERING	0	0	0	0	0	0
Q29		POLISHING POND INFLOW	0	3,806	1,601	0	3,806	1,601
	Q - Precipitation on Polishing Pond	Precipitation minus Sublimation ON POLISHING POND	0	3,049	1,602	0	3,049	1,602
	Q - Evaporation from Polishing Pond	EVAPORATION FROM POLISHING POND	0	1,355	1,063	0	1,355	1,063
Q30		POLISHING POND OUTFLOW	0	5,499	2,140	0	5,499	2,140
Q31		RECYCLE FROM FINAL POLISHING POND						
Q32		DISCHARGE PIPELINE						
Q33		DISCHARGE TO MINAGO						
Q34		MINAGO UPSTREAM						
Q35		MINAGO DOWNSTREAM						
Q36		DISCHARGE TO OAKLEY CREEK	0	5,499	2,140	0	5,499	2,140
Q37		OAKLEY CREEK UPSTREAM	0	345,600	43,200	0	345,600	43,200
Q38		OAKLEY CREEK DOWNSTREAM	0	351,099	45,340	0	351,099	45,340
FLOW RATIOS:								
Q33 / Q34		RATIO OF DISCHARGE TO MINAGO TO FLOW IN MINAGO	0%	0%	0%	0%	0%	0%
Q36 / Q37		RATIO OF DISCHARGE TO OAKLEY CK TO FLOW IN OAKLEY CK	0%	2%	5%	0%	2%	5%

Table 2.14-18 Projected Concentrations in Flows around the Minago Site during Closure

SCENARIO:			ESTIMATED AVERAGE CONCENTRATION						REGULATIONS				
			Closure (Stage 1)	Closure (Stage 2)	Closure (Stage 2)	Year 12 - Closure (Stage 2)			(2002)				the Protection of Aquatic Life
			Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate			TIER II Water Quality Objectives		
FLOW		WATER QUALITY PARAM.	NOVEMBER TO APRIL (mg/L)	MAY (mg/L)	JUNE TO OCTOBER (mg/L)	NOVEMBER TO APRIL (mg/L)	MAY (mg/L)	JUNE TO OCTOBER (mg/L)	Monthly Mean	Grab Sample	assuming hardness = 150 mg/L $CaCO_3$	Freshwater	(CCME, 2007)
Q30	POLISHING POND OUTFLOW	Al	0.000	0.062	0.081	0.000	0.093	0.112				0.005 - 0.1	0.005
Q30	POLISHING POND OUTFLOW	Sb	0.00000	0.00104	0.00135	0.00000	0.00134	0.00166					
Q30	POLISHING POND OUTFLOW	As	0.000	0.003	0.004	0.000	0.004	0.005	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q30	POLISHING POND OUTFLOW	Cd	0.00000	0.00027	0.00038	0.00000	0.00046	0.00055			0.00302 [B]	Tier II	0.000017 or $10^{0.86[\log(hardness)]-3.2}$
Q30	POLISHING POND OUTFLOW	Cr	0.0000	0.0021	0.0027	0.0000	0.0027	0.0033			0.10331 [C]	Tier II	
Q30	POLISHING POND OUTFLOW	Co	0.00000	0.00103	0.00132	0.00000	0.00136	0.00161					
Q30	POLISHING POND OUTFLOW	Cu	0.0000	0.0089	0.0118	0.0000	0.0137	0.0168	0.3	0.6	0.01266 [D]	Tier II	0.002
Q30	POLISHING POND OUTFLOW	Fe	0.000	0.095	0.113	0.000	0.107	0.126				0.3	0.3
Q30	POLISHING POND OUTFLOW	Pb	0.00000	0.00540	0.00748	0.00000	0.00903	0.01113	0.2	0.4	0.0039 [E]	Tier II	0.001
Q30	POLISHING POND OUTFLOW	Mo	0.0000	0.0095	0.0121	0.0000	0.0136	0.0163				0.073	
Q30	POLISHING POND OUTFLOW	Ni	0.000	0.041	0.052	0.000	0.058	0.069	0.5	1	0.07329 [F]	Tier II	0.025
Q30	POLISHING POND OUTFLOW	Se	0.0000	0.0017	0.0023	0.0000	0.0023	0.0029				0.001	0.001
Q30	POLISHING POND OUTFLOW	Zn	0.000	0.031	0.042	0.000	0.048	0.059	0.5	1	0.16657 [G]	Tier II	0.03
Q35	MINAGO DOWNSTREAM	Al	N/A	N/A	N/A	N/A	N/A	N/A				0.005 - 0.1	0.005
Q35	MINAGO DOWNSTREAM	Sb	N/A	N/A	N/A	N/A	N/A	N/A					
Q35	MINAGO DOWNSTREAM	As	N/A	N/A	N/A	N/A	N/A	N/A	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q35	MINAGO DOWNSTREAM	Cd	N/A	N/A	N/A	N/A	N/A	N/A			0.00302 [B]	Tier II	0.000017 or $10^{0.86[\log(hardness)]-3.2}$
Q35	MINAGO DOWNSTREAM	Cr	N/A	N/A	N/A	N/A	N/A	N/A			0.10331 [C]	Tier II	
Q35	MINAGO DOWNSTREAM	Co	N/A	N/A	N/A	N/A	N/A	N/A					
Q35	MINAGO DOWNSTREAM	Cu	N/A	N/A	N/A	N/A	N/A	N/A	0.3	0.6	0.01266 [D]	Tier II	0.002
Q35	MINAGO DOWNSTREAM	Fe	N/A	N/A	N/A	N/A	N/A	N/A				0.3	0.3
Q35	MINAGO DOWNSTREAM	Pb	N/A	N/A	N/A	N/A	N/A	N/A	0.2	0.4	0.0039 [E]	Tier II	0.001
Q35	MINAGO DOWNSTREAM	Mo	N/A	N/A	N/A	N/A	N/A	N/A				0.073	
Q35	MINAGO DOWNSTREAM	Ni	N/A	N/A	N/A	N/A	N/A	N/A	0.5	1	0.07329 [F]	Tier II	0.025
Q35	MINAGO DOWNSTREAM	Se	N/A	N/A	N/A	N/A	N/A	N/A				0.001	0.001
Q35	MINAGO DOWNSTREAM	Zn	N/A	N/A	N/A	N/A	N/A	N/A	0.5	1	0.16657 [G]	Tier II	0.03

Table 2.14-18 (Cont.'d) Projected Concentrations in Flows around the Minago Site during Closure

SCENARIO:			ESTIMATED AVERAGE CONCENTRATION						REGULATIONS				
			Closure (Stage 1)	Closure (Stage 2)	Closure (Stage 2)	Year 12 - Closure (Stage 2)			(2002)				the Protection of Aquatic Life
			Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate			TIER II Water Quality Objectives		
		WATER QUALITY	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	Monthly Mean	Grab Sample	assuming hardness = 150 mg/L $CaCO_3$	Freshwater	(CCME, 2007)
FLOW		PARAM.	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)					
Q38	OAKLEY CREEK DOWNSTREAM	Al	N/A	0.0036	0.0064	N/A	0.0041	0.0078				0.005 - 0.1	0.005
Q38	OAKLEY CREEK DOWNSTREAM	Sb	N/A	0.000050	0.000096	N/A	0.000054	0.000111					
Q38	OAKLEY CREEK DOWNSTREAM	As	N/A	0.0004	0.0005	N/A	0.0004	0.0006	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q38	OAKLEY CREEK DOWNSTREAM	Cd	N/A	0.000017	0.000030	N/A	0.000020	0.000038			0.00302 [B]	Tier II	0.000017 or $10^{\{0.86[\log(\text{hardness})]-3.2\}}$
Q38	OAKLEY CREEK DOWNSTREAM	Cr	N/A	0.0003	0.0004	N/A	0.0003	0.0004			0.10331 [C]	Tier II	
Q38	OAKLEY CREEK DOWNSTREAM	Co	N/A	0.0000	0.0001	N/A	0.0001	0.0001					
Q38	OAKLEY CREEK DOWNSTREAM	Cu	N/A	0.0003	0.0007	N/A	0.0004	0.0009	0.3	0.6	0.01266 [D]	Tier II	0.002
Q38	OAKLEY CREEK DOWNSTREAM	Fe	N/A	0.0512	0.0534	N/A	0.0514	0.0540				0.3	0.3
Q38	OAKLEY CREEK DOWNSTREAM	Pb	N/A	0.0001	0.0004	N/A	0.0002	0.0005	0.2	0.4	0.0039 [E]	Tier II	0.001
Q38	OAKLEY CREEK DOWNSTREAM	Mo	N/A	0.0002	0.0007	N/A	0.0003	0.0009				0.073	
Q38	OAKLEY CREEK DOWNSTREAM	Ni	N/A	0.0009	0.0026	N/A	0.0011	0.0034	0.5	1	0.07329 [F]	Tier II	0.025
Q38	OAKLEY CREEK DOWNSTREAM	Se	N/A	0.0003	0.0003	N/A	0.0003	0.0004				0.001	0.001
Q38	OAKLEY CREEK DOWNSTREAM	Zn	N/A	0.0011	0.0025	N/A	0.0014	0.0034	0.5	1	0.16657 [G]	Tier II	0.03

Notes: **N/A not applicable**

August 2008 groundwater chemistry was assumed in the modeling.

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.

Table 2.14-19 Projected Flow Rates during Post Closure

FLOW		Year 13 - Post Closure		
		Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate
		NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER
		m³/day	m³/day	m³/day
UNIT EVAPORATION	UNIT LAKE EVAPORATION	0	18	14
UNIT PPT (U-PPT)	UNIT PRECIPITATION	0	41	21
Q1	FLOW FROM DEWATERING WELLS	0	0	0
Q2	WELL WATER FOR PROCESSING	0	0	0
Q3	EXCESS WATER FROM DEWATERING WELLS	0	0	0
Q4	GROUNDWATER TO OTHER OPERATIONS	0	0	0
Q5	GROUNDWATER TO WATER TREATMENT	0	0	0
Q6	GROUNDWATER TO FRAC SAND PLANT	0	0	0
Q7	GROUNDWATER FOR FIRE FIGHTING	0	0	0
Q8	POTABLE WATER	0	0	0
Q9	WATER TREATMENT PLANT WASTE	0	0	0
Q10	RECYCLE WATER FROM FPP	0	0	0
Q11	POTABLE WATER TO MILL	0	0	0
Q12	POTABLE WATER TO OTHER OPERATIONS	0	0	0
Q13	POTABLE WATER TO OFFICES & CAMP	0	0	0
Q14	POTABLE WATER TO FRAC SAND PLANT	0	0	0
Q15	FLOW FROM OPERATIONS TO MILL	0	0	0
Q16	SEWAGE & GREY WATER FROM CAMP AND OFFICES	0	0	0
Q17	SEWAGE & GREY WATER FROM ALL OTHER ON SITE SOURCES	0	0	0
Q19	FLOW FROM CONCENTRATE THICKENER IN MILL TO MILL	0	0	0
Q20	FLOW FROM MILL TO MILL THICKENER	0	0	0
Q21	RECYCLE WATER FROM MILL THICKENER	0	0	0
Q21x	ATERNATE FLOW FOR RECYCLE WATER FROM MILL THICKENER	0	0	0
Q22	MILL TAILINGS SLURRY	0	0	0
Q23	SEWAGE TREATMENT OUTFLOW	0	0	0
Q24	LIQ. WASTE FROM FSP	0	0	0
Q25	SLURRY FROM FSP	0	0	0
Q26	TWRMF INFLOW	0	0	0
Q - Liquid PPT on TWRMF	PPT on TWRMF	0	8,930	4,694
Q - Retained Water in Tailings Voids	Q - Retained Water in Tailings Voids	0	0	0
Q - TWRMF Supernatant	TWRMF Supernatant	18,308	109,987	22,978
Q27	TWRMF Decant	0	3,681	1,579
Q - Pit Dewatering	OPEN PIT DEWATERING	0	0	0
Q - Precipitation on Pit	Precipitation minus Sublimation on Open Pit	0	7,723	4,059
Q28	TOT. OPEN PIT DEWATERING	0	0	0
Q29	POLISHING POND INFLOW	0	3,681	1,579
Q - Precipitation on Polishing Pond	Precipitation minus Sublimation ON POLISHING POND	0	3,049	1,602
Q - Evaporation from Polishing Pond	EVAPORATION FROM POLISHING POND	0	1,355	1,063
Q30	POLISHING POND OUTFLOW	0	5,375	2,117
Q31	RECYCLE FROM FINAL POLISHING POND			
Q32	DISCHARGE PIPELINE			
Q33	DISCHARGE TO MINAGO			
Q34	MINAGO UPSTREAM			
Q35	MINAGO DOWNSTREAM			
Q36	DISCHARGE TO OAKLEY CREEK	0	5,375	2,117
Q37	OAKLEY CREEK UPSTREAM	0	345,600	43,200
Q38	OAKLEY CREEK DOWNSTREAM	0	350,975	45,317
FLOW RATIOS:				
Q33 / Q34	RATIO OF DISCHARGE TO MINAGO TO FLOW IN MINAGO	0%	0%	0%
Q36 / Q37	RATIO OF DISCHARGE TO OAKLEY CK TO FLOW IN OAKLEY CK	0%	2%	5%

The projected parametric concentrations for the Polishing Pond outflow (Q30), Minago downstream (Q35), and Oakley Creek downstream (Q38) are given in Table 2.14-20. Additional results for Q26 (TWRMF Inflow), Q27 (TWRMF Decant), Q29 (Polishing Pond Inflow) are given in Appendix 2.14.

During the Post Closure, the projected outflow from the Polishing Pond will meet MMER requirements at all times. During the Post Closure period, the projected water quality in Oakley Creek downstream of the mixing zones will meet the Manitoba Freshwater guidelines for the protection of aquatic life for all parameters.

2.14.2.3.5 Water Balance Modeling Results during Temporary Suspension and a State of Inactivity

Estimated flowrates during Temporary Suspension and the State of Inactivity are listed in Table 2.14-21 and the corresponding water management diagrams are shown in Figure 2.14-9 and Figure 2.14-10, respectively.

During the Temporary Suspension of operations, the Polishing Pond discharge to Minago River (Q33) in relation to the Minago River streamflow (Q34) will be 11% in May, 20% in the summer months (June to October) and 38% in the winter months (November to April). In absolute quantities, discharge to Minago River will range from 26,000 m^3/day to 95,000 m^3/day during the Temporary Suspension of operations.

During the Temporary Suspension of operations, the projected Polishing Pond discharge to Oakley Creek (Q36) to the Oakley Creek streamflow (Q37) will be 0% in the winter months (Nov. to Apr.), 12% in May, and 32% in the summer months (June to October). In absolute quantities, discharge to Oakley Creek will range from 0 m^3/day to 40,715 m^3/day during the Temporary Suspension of operations.

During the State of Inactivity, the projected Polishing Pond discharge to Minago River (Q33) in relation to the Minago River streamflow (Q34) will be 0% year round. During the State of Inactivity, the projected Polishing Pond discharge to Oakley Creek (Q36) in relation to the Oakley Creek streamflow (Q37) will be 0% in the winter months (Nov. to Apr.), 7% in May, and 11% in the summer months (June to October). In absolute quantities, discharge to Oakley Creek will range from 0 m^3/day to 24,830 m^3/day during the State of Inactivity.

Table 2.14-22 presents projected parametric concentrations for the Polishing Pond outflow (Q30), Minago downstream (Q35), and Oakley Creek downstream (Q38) during Temporary Suspension and the State of Inactivity. Additional results for Q26 (TWRMF Inflow), Q27 (TWRMF Decant), and Q29 (Polishing Pond Inflow) are given in Appendix 2.14.

During Temporary Suspension, the projected outflow from the Polishing Pond will meet MMER requirements at all times. During Temporary Suspension, the projected water quality in Minago

Table 2.14-20 Projected Concentrations in Flows around the Minago Site during Post Closure

SCENARIO:		WATER QUALITY	ESTIMATED AVERAGE CONCENTRATION Year 13 - POST CLOSURE			REGULATIONS (2002)				the Protection of Aquatic Life
			Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate			TIER II Water Quality Objectives		
			NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	Monthly Mean	Grab Sample	assuming hardness = 150 mg/L $CaCO_3$	Freshwater	(CCME, 2007)
FLOW		PARAM.	(mg/L)	(mg/L)	(mg/L)					
Q30	POLISHING POND OUTFLOW	Al	0.000	0.119	0.114				0.005 - 0.1	0.005
Q30	POLISHING POND OUTFLOW	Sb	0.00000	0.00161	0.00222					
Q30	POLISHING POND OUTFLOW	As	0.000	0.005	0.006	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q30	POLISHING POND OUTFLOW	Cd	0.00000	0.00060	0.00054			0.00302 [B]	Tier II	0.000017 or $10^{\{0.86[\log(hardness)]-3.2\}}$
Q30	POLISHING POND OUTFLOW	Cr	0.0000	0.0031	0.0040			0.10331 [C]	Tier II	
Q30	POLISHING POND OUTFLOW	Co	0.00000	0.00159	0.00191					
Q30	POLISHING POND OUTFLOW	Cu	0.0000	0.0180	0.0203	0.3	0.6	0.01266 [D]	Tier II	0.002
Q30	POLISHING POND OUTFLOW	Fe	0.000	0.118	0.131				0.3	0.3
Q30	POLISHING POND OUTFLOW	Pb	0.00000	0.01214	0.01107	0.2	0.4	0.0039 [E]	Tier II	0.001
Q30	POLISHING POND OUTFLOW	Mo	0.0000	0.0171	0.0224				0.073	
Q30	POLISHING POND OUTFLOW	Ni	0.000	0.072	0.087	0.5	1	0.07329 [F]	Tier II	0.025
Q30	POLISHING POND OUTFLOW	Se	0.0000	0.0029	0.0035				0.001	0.001
Q30	POLISHING POND OUTFLOW	Zn	0.000	0.063	0.062	0.5	1	0.16657 [G]	Tier II	0.03
Q35	MINAGO DOWNSTREAM	Al	N/A	N/A	N/A				0.005 - 0.1	0.005
Q35	MINAGO DOWNSTREAM	Sb	N/A	N/A	N/A					
Q35	MINAGO DOWNSTREAM	As	N/A	N/A	N/A	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q35	MINAGO DOWNSTREAM	Cd	N/A	N/A	N/A			0.00302 [B]	Tier II	0.000017 or $10^{\{0.86[\log(hardness)]-3.2\}}$
Q35	MINAGO DOWNSTREAM	Cr	N/A	N/A	N/A			0.10331 [C]	Tier II	
Q35	MINAGO DOWNSTREAM	Co	N/A	N/A	N/A					
Q35	MINAGO DOWNSTREAM	Cu	N/A	N/A	N/A	0.3	0.6	0.01266 [D]	Tier II	0.002
Q35	MINAGO DOWNSTREAM	Fe	N/A	N/A	N/A				0.3	0.3
Q35	MINAGO DOWNSTREAM	Pb	N/A	N/A	N/A	0.2	0.4	0.0039 [E]	Tier II	0.001
Q35	MINAGO DOWNSTREAM	Mo	N/A	N/A	N/A				0.073	
Q35	MINAGO DOWNSTREAM	Ni	N/A	N/A	N/A	0.5	1	0.07329 [F]	Tier II	0.025
Q35	MINAGO DOWNSTREAM	Se	N/A	N/A	N/A				0.001	0.001
Q35	MINAGO DOWNSTREAM	Zn	N/A	N/A	N/A	0.5	1	0.16657 [G]	Tier II	0.03

Table 2.14-20 (Cont.'d) Projected Concentrations in Flows around the Minago Site during Post Closure

SCENARIO:			ESTIMATED AVERAGE CONCENTRATION			REGULATIONS				
			Year 13 - POST CLOSURE			(2002)		TIER II Water Quality Objectives		the Protection of Aquatic Life
			Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate					
		WATER QUALITY	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	Monthly Mean	Grab Sample	assuming hardness = 150 mg/L $CaCO_3$	Freshwater	(CCME, 2007)
FLOW		PARAM.	(mg/L)	(mg/L)	(mg/L)					
Q38	OAKLEY CREEK DOWNSTREAM	Al	N/A	0.0044	0.0079				0.005 - 0.1	0.005
Q38	OAKLEY CREEK DOWNSTREAM	Sb	N/A	0.000058	0.000136					
Q38	OAKLEY CREEK DOWNSTREAM	As	N/A	0.0005	0.0006	0.5	1	0.15 mg/L (4-Day, 3-Year) A	Tier II	0.005
Q38	OAKLEY CREEK DOWNSTREAM	Cd	N/A	0.000022	0.000037			0.00302 B	Tier II	0.000017 or $10^{\{0.86[\log(hardness)]-3.2\}}$
Q38	OAKLEY CREEK DOWNSTREAM	Cr	N/A	0.0003	0.0005			0.10331 C	Tier II	
Q38	OAKLEY CREEK DOWNSTREAM	Co	N/A	0.0001	0.0001					
Q38	OAKLEY CREEK DOWNSTREAM	Cu	N/A	0.0004	0.0011	0.3	0.6	0.01266 D	Tier II	0.002
Q38	OAKLEY CREEK DOWNSTREAM	Fe	N/A	0.0515	0.0542				0.3	0.3
Q38	OAKLEY CREEK DOWNSTREAM	Pb	N/A	0.0002	0.0005	0.2	0.4	0.0039 E	Tier II	0.001
Q38	OAKLEY CREEK DOWNSTREAM	Mo	N/A	0.0004	0.0011				0.073	
Q38	OAKLEY CREEK DOWNSTREAM	Ni	N/A	0.0013	0.0043	0.5	1	0.07329 F	Tier II	0.025
Q38	OAKLEY CREEK DOWNSTREAM	Se	N/A	0.0003	0.0004				0.001	0.001
Q38	OAKLEY CREEK DOWNSTREAM	Zn	N/A	0.0016	0.0035	0.5	1	0.16657 G	Tier II	0.03

Notes: **N/A not applicable**

August 2008 groundwater chemistry was assumed in the modeling.

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.

Table 2.14-21 Projected Flow Rates during Temporary Suspension and State of Inactivity

		TS after Year 4			SI after one year TS		
		Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate
		NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER
FLOW		m^3/day	m^3/day	m^3/day	m^3/day	m^3/day	m^3/day
UNIT EVAPORATION	UNIT LAKE EVAPORATION	0	18	14	0	18	14
UNIT PPT (U-PPT)	UNIT PRECIPITATION	0	41	21	0	41	21
Q1	FLOW FROM DEWATERING WELLS	31,999	31,999	31,999	2,666	2,667	2,667
Q2	WELL WATER FOR PROCESSING	6	6	6	3	3	3
Q3	EXCESS WATER FROM DEWATERING WELLS	31,993	31,993	31,993	2,663	2,664	2,664
Q4	GROUNDWATER TO OTHER OPERATIONS	0	0	0	0	0	0
Q5	GROUNDWATER TO WATER TREATMENT	6	6	6	3	3	3
Q6	GROUNDWATER TO FRAC SAND PLANT	0	0	0	0	0	0
Q7	GROUNDWATER FOR FIRE FIGHTING	0	0	0	0	0	0
Q8	POTABLE WATER	6	6	6	3	3	3
Q9	WATER TREATMENT PLANT WASTE	0	0	0	0	0	0
Q10	RECYCLE WATER FROM FPP	0	0	0	0	0	0
Q11	POTABLE WATER TO MILL	0	0	0	0	0	0
Q12	POTABLE WATER TO OTHER OPERATIONS	0	0	0	0	0	0
Q13	POTABLE WATER TO OFFICES & CAMP	6	6	6	3	3	3
Q14	POTABLE WATER TO FRAC SAND PLANT	0	0	0	0	0	0
Q15	FLOW FROM OPERATIONS TO MILL	0	0	0	0	0	0
Q16	SEWAGE & GREY WATER FROM CAMP AND OFFICES	6	6	6	3	3	3
Q17	SEWAGE & GREY WATER FROM ALL OTHER ON SITE SOURCES	0	0	0	0	0	0
Q19	FLOW FROM CONCENTRATE THICKENER IN MILL TO MILL	0	0	0	0	0	0
Q20	FLOW FROM MILL TO MILL THICKENER	0	0	0	0	0	0
Q21	RECYCLE WATER FROM MILL THICKENER	0	0	0	0	0	0
Q21x	ATERNATE FLOW FOR RECYCLE WATER FROM MILL THICKENER	0	0	0	0	0	0
Q22	MILL TAILINGS SLURRY	0	0	0	0	0	0
Q23	SEWAGE TREATMENT OUTFLOW	0	63	13	0	43	10
Q24	LIQ. WASTE FROM FSP	0	0	0	0	0	0
Q25	SLURRY FROM FSP	0	0	0	0	0	0
Q26	TWRMF INFLOW	0	63	13	0	43	10
Q - Liquid PPT on TWRMF	PPT on TWRMF	0	8,930	4,694	0	8,930	4,694
Q - Retained Water in Tailings Voids	Q - Retained Water in Tailings Voids	0	0	0	0	0	0
Q - TWRMF Supernatant	TWRMF Supernatant	6,103	40,461	8,725	6,103	40,440	8,722
Q27	TWRMF Decant	0	5,025	1,592	0	5,005	1,589
Q - Pit Dewatering	OPEN PIT DEWATERING	8,000	8,000	8,000	0	0	0
Q - Precipitation on Pit	Precipitation minus Sublimation on Open Pit	0	7,723	4,059	0	7,723	4,059
Q28	TOT. OPEN PIT DEWATERING	8,000	15,723	12,059	0	0	0
Q29	POLISHING POND INFLOW	39,993	134,018	45,644	2,663	23,133	4,252
Q - Precipitation on Polishing Pond	Precipitation minus Sublimation ON POLISHING POND	0	3,049	1,602	0	3,049	1,602
Q - Evaporation from Polishing Pond	EVAPORATION FROM POLISHING POND	0	1,355	1,063	0	1,355	1,063
Q30	POLISHING POND OUTFLOW	39,993	135,712	46,183	2,663	24,827	4,791
Q31	RECYCLE FROM FINAL POLISHING POND	0	0	0	0	0	0
Q32	DISCHARGE PIPELINE	25,996	135,712	46,183	0	24,827	4,791
Q33	DISCHARGE TO MINAGO	25,996	94,998	32,328	0	0	0
Q34	MINAGO UPSTREAM	69,120	864,000	164,160	69,120	864,000	164,160
Q35	MINAGO DOWNSTREAM	95,116	958,998	196,488	69,120	864,000	164,160
Q36	DISCHARGE TO OAKLEY CREEK	0	40,713	13,855	0	24,827	4,791
Q37	OAKLEY CREEK UPSTREAM	0	345,600	43,200	0	345,600	43,200
Q38	OAKLEY CREEK DOWNSTREAM	0	386,313	57,055	0	370,427	47,991
FLOW RATIOS:							
Q33 / Q34	RATIO OF DISCHARGE TO MINAGO TO FLOW IN MINAGO	38%	11%	20%	0%	0%	0%
Q36 / Q37	RATIO OF DISCHARGE TO OAKLEY CK TO FLOW IN OAKLEY CK	0%	12%	32%	0%	7%	11%

Table 2.14-22 Projected Effluent Concentrations in Flows during Temporary Suspension and the State of Inactivity

SCENARIO:			WATER QUALITY	ESTIMATED AVERAGE CONCENTRATION						REGULATIONS				
				TS after Year 4			SI after one year TS			Metal Mining Liquid Effluents (2002)		Manitoba Water Quality Standards, Objectives, and Guidelines (Williamson, 2002)		Canadian Water Quality Guidelines for the Protection of Aquatic Life
				Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate			TIER II Water Quality Objectives		
				NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER					
FLOW			PARAM.	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	Monthly Mean	Grab Sample	assuming hardness = 150 mg/L $CaCO_3$	Freshwater	(CCME, 2007)
Q30	POLISHING POND OUTFLOW	RC30	Al	0.009	0.028	0.024	0.009	0.097	0.138				0.005 - 0.1	0.005
Q30	POLISHING POND OUTFLOW	RC30	Sb	0.00003	0.00023	0.00023	0.00003	0.00091	0.00146					
Q30	POLISHING POND OUTFLOW	RC30	As	0.001	0.001	0.001	0.001	0.002	0.003	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q30	POLISHING POND OUTFLOW	RC30	Cd	0.00001	0.00004	0.00004	0.00001	0.00023	0.00036			0.00302 [B]	Tier II	0.000017 or $10^{\{0.86[\log(hardness)]-3.2\}}$
Q30	POLISHING POND OUTFLOW	RC30	Cr	0.0010	0.0015	0.0014	0.0010	0.0032	0.0043			0.10331 [C]	Tier II	
Q30	POLISHING POND OUTFLOW	RC30	Co	0.00008	0.00049	0.00042	0.00008	0.00178	0.00255					
Q30	POLISHING POND OUTFLOW	RC30	Cu	0.0005	0.0021	0.0019	0.0005	0.0089	0.0137	0.3	0.6	0.01266 [D]	Tier II	0.002
Q30	POLISHING POND OUTFLOW	RC30	Fe	0.005	0.089	0.071	0.005	0.337	0.466				0.3	0.3
Q30	POLISHING POND OUTFLOW	RC30	Pb	0.00003	0.00052	0.00056	0.00003	0.00350	0.00597	0.2	0.4	0.0039 [E]	Tier II	0.001
Q30	POLISHING POND OUTFLOW	RC30	Mo	0.0007	0.0014	0.0014	0.0007	0.0054	0.0087				0.073	
Q30	POLISHING POND OUTFLOW	RC30	Ni	0.001	0.020	0.016	0.001	0.085	0.121	0.5	1	0.07329 [F]	Tier II	0.025
Q30	POLISHING POND OUTFLOW	RC30	Se	0.0002	0.0006	0.0006	0.0002	0.0018	0.0029				0.001	0.001
Q30	POLISHING POND OUTFLOW	RC30	Zn	0.005	0.007	0.007	0.005	0.021	0.033	0.5	1	0.16657 [G]	Tier II	0.03
Q35	MINAGO DOWNSTREAM	RC35	Al	0.011	0.014	0.014	N/A	N/A	N/A				0.005 - 0.1	0.005
Q35	MINAGO DOWNSTREAM	RC35	Sb	0.00004	0.00007	0.00008	N/A	N/A	N/A					
Q35	MINAGO DOWNSTREAM	RC35	As	0.0007	0.0006	0.0007	N/A	N/A	N/A	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q35	MINAGO DOWNSTREAM	RC35	Cd	0.000014	0.000019	0.000021	N/A	N/A	N/A			0.00302 [B]	Tier II	0.000017 or $10^{\{0.86[\log(hardness)]-3.2\}}$
Q35	MINAGO DOWNSTREAM	RC35	Cr	0.00044	0.00035	0.00042	N/A	N/A	N/A			0.10331 [C]	Tier II	
Q35	MINAGO DOWNSTREAM	RC35	Co	0.00006	0.00009	0.00011	N/A	N/A	N/A					
Q35	MINAGO DOWNSTREAM	RC35	Cu	0.001	0.001	0.001	N/A	N/A	N/A	0.3	0.6	0.01266 [D]	Tier II	0.002
Q35	MINAGO DOWNSTREAM	RC35	Fe	0.052	0.071	0.070	N/A	N/A	N/A				0.3	0.3
Q35	MINAGO DOWNSTREAM	RC35	Pb	0.00005	0.00010	0.00014	N/A	N/A	N/A	0.2	0.4	0.0039 [E]	Tier II	0.001
Q35	MINAGO DOWNSTREAM	RC35	Mo	0.00028	0.00025	0.00034	N/A	N/A	N/A				0.073	
Q35	MINAGO DOWNSTREAM	RC35	Ni	0.001	0.003	0.004	N/A	N/A	N/A	0.5	1	0.07329 [F]	Tier II	0.025
Q35	MINAGO DOWNSTREAM	RC35	Se	0.00023	0.00028	0.00030	N/A	N/A	N/A				0.001	0.001
Q35	MINAGO DOWNSTREAM	RC35	Zn	0.002	0.002	0.002	N/A	N/A	N/A	0.5	1	0.16657 [G]	Tier II	0.03

Table 2.14-22 (Cond.'d) Projected Effluent Concentrations in Flows during Temporary Suspension and the State of Inactivity

SCENARIO:				ESTIMATED AVERAGE CONCENTRATION						REGULATIONS				
				TS after Year 4			SI after one year TS			Metal Mining Liquid Effluents (2002)		Manitoba Water Quality Standards, Objectives, and Guidelines (Williamson, 2002)		Canadian Water Quality Guidelines for the Protection of Aquatic Life
				Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate	Tailings only; max.tailings leaching rate			TIER II Water Quality Objectives		
			WATER QUALITY	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	NOVEMBER TO APRIL	MAY	JUNE TO OCTOBER	Monthly Mean	Grab Sample	assuming hardness = 150 mg/L $CaCO_3$	Freshwater	
FLOW			PARAM.	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)					(CCME, 2007)
Q38	OAKLEY CREEK DOWNSTREAM	RC38	Al	N/A	0.0053	0.0079	N/A	0.0090	0.0162				0.005 - 0.1	0.005
Q38	OAKLEY CREEK DOWNSTREAM	RC38	Sb	N/A	0.000054	0.000081	N/A	0.000092	0.000177					
Q38	OAKLEY CREEK DOWNSTREAM	RC38	As	N/A	0.0004	0.0005	N/A	0.0005	0.0007	0.5	1	0.15 mg/L (4-Day, 3-Year) [A]	Tier II	0.005
Q38	OAKLEY CREEK DOWNSTREAM	RC38	Cd	N/A	0.000016	0.000020	N/A	0.000027	0.000047			0.00302 [B]	Tier II	0.000017 or $10^{\{0.86[\log(\text{hardness})]-3.2\}}$
Q38	OAKLEY CREEK DOWNSTREAM	RC38	Cr	N/A	0.0004	0.0006	N/A	0.0005	0.0007			0.10331 [C]	Tier II	
Q38	OAKLEY CREEK DOWNSTREAM	RC38	Co	N/A	0.0001	0.0001	N/A	0.0002	0.0003					
Q38	OAKLEY CREEK DOWNSTREAM	RC38	Cu	N/A	0.0004	0.0006	N/A	0.0007	0.0015	0.3	0.6	0.01266 [D]	Tier II	0.002
Q38	OAKLEY CREEK DOWNSTREAM	RC38	Fe	N/A	0.0545	0.0556	N/A	0.0697	0.0920				0.3	0.3
Q38	OAKLEY CREEK DOWNSTREAM	RC38	Pb	N/A	0.0001	0.0002	N/A	0.0003	0.0006	0.2	0.4	0.0039 [E]	Tier II	0.001
Q38	OAKLEY CREEK DOWNSTREAM	RC38	Mo	N/A	0.0002	0.0004	N/A	0.0005	0.0010				0.073	
Q38	OAKLEY CREEK DOWNSTREAM	RC38	Ni	N/A	0.0024	0.0041	N/A	0.0059	0.0122	0.5	1	0.07329 [F]	Tier II	0.025
Q38	OAKLEY CREEK DOWNSTREAM	RC38	Se	N/A	0.0003	0.0003	N/A	0.0003	0.0005				0.001	0.001
Q38	OAKLEY CREEK DOWNSTREAM	RC38	Zn	N/A	0.0013	0.0022	N/A	0.0020	0.0039	0.5	1	0.16657 [G]	Tier II	0.03

Notes: **N/A not applicable**

August 2008 groundwater chemistry was assumed in the modeling.

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.

River and Oakley Creek downstream of the mixing zone will meet the Manitoba Tier III Freshwater guidelines for all parameters.

During the State of Inactivity, projected outflow from the Polishing Pond meets MMER requirements at all times. During the State of Inactivity, the projected water quality in Oakley Creek downstream of the mixing zone meets the Manitoba Tier III Freshwater guidelines for all parameters.

2.14.2.3.6 Storm Water Management

The site storm water management at the Minago Project is designed to accommodate a 1-in-20 year storm event over a 5-day period (120 mm) (Wardrop, 2009b).

Site water will be pumped to designated area settling ponds and sumps, or discharged to the local watersheds via runoff. Surface runoff from the industrial area, Overburden Disposal Facility, Dolomite Waste Rock Dump (WRD) and Country Rock WRD will be benign and is not expected to require treatment. The storm water falling on no-process areas including the Dolomite WRD and the Country Rock WRD will report to the natural environment. Settling ponds will nonetheless be built to control major events in the Overburden Disposal Facility areas. Seepage from the Tailings and Ultramafic Waste Rock Management Facility (TWRMF) will be collected in a perimeter ditch around the exterior of the facility and will be pumped back into TWRMF. The Polishing Pond and flood retention area will contain the storm water from the TWRMF, mine dewatering and site runoff. During operations, this water will be pumped to the Minago River and the Oakley Creek watersheds, and a portion will be diverted back to the process water tank (Wardrop, 2009b). Storm water release from the Polishing Pond will be staged over several days as needed to condition the Minago River and the Oakley Creek watersheds. After closure, water from the Polishing Pond will be discharged into the cross-ditch to report to the Oakley Creek. The major cross-site ditch will report to the ditch at Highway 6 and to the Oakley Creek through the low lying marsh on the east side of Highway 6.

2.14.2.3.7 Contaminants of Concern (CoC)

All discharges to the receiving environment are expected to meet the MMER guidelines during all stages of the mine development, closure and post closure periods. Table 2.14-23 summarizes the projected Polishing Pond water quality for the different mine development and closure stages against the MMER guideline limits (Environment Canada, 2002a). On the basis of the projected discharge water quantity for all phases of operation, there will be no contaminant of concern (CoC) for this project as all contaminants meet MMER guidelines.

2.14.3 Seepage Control

Seepage from the TWRMF will be collected with interceptor ditches surrounding the TWRMF. To ensure good capture of seepage from the tailings dam, the interceptor channel will be deep

Table 2.14-23 Water Quality of Polishing Pond Discharges

SCENARIO:		ESTIMATED AVERAGE CONCENTRATION																		REGULATIONS	
		DURING CONSTRUCTION	DURING OPERATIONS							Year 11			Year 12			Year 13				Metal Mining Liquid Effluents (2002)	
			Year 1 through Year 8	Year 9			Year 10			Closure (Stage 1)	Closure (Stage 2)	Closure (Stage 2)	Closure (Stage 2)			POST CLOSURE					
	WATER QUALITY	Maximum	Maximum	NOV. TO APRIL	MAY	JUNE TO OCTOBER	NOV. TO APRIL	MAY	JUNE TO OCTOBER	NOV. TO APRIL	MAY	JUNE TO OCTOBER	NOV. TO APRIL	MAY	JUNE TO OCTOBER	NOV. TO APRIL	MAY	JUNE TO OCTOBER	OVERALL MAXIMUM		
FLOW	PARAM.	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	(mg/L)	Monthly Mean	Grab Sample
POLISHING POND OUTFLOW	As	0.001	0.002	0.006	0.005	0.005	0.006	0.006	0.005	0.000	0.003	0.004	0.000	0.004	0.005	0.000	0.005	0.006	0.006	0.5	1
POLISHING POND OUTFLOW	Cu	0.0006	0.0161	0.0393	0.0286	0.0247	0.0261	0.0240	0.0192	0.0000	0.0089	0.0118	0.0000	0.0137	0.0168	0.0000	0.0180	0.0203	0.0393	0.3	0.6
POLISHING POND OUTFLOW	Pb	0.00006	0.00375	0.01304	0.01008	0.01070	0.01318	0.01218	0.01076	0.00000	0.00540	0.00748	0.00000	0.00903	0.01113	0.00000	0.01214	0.01107	0.01318	0.2	0.4
POLISHING POND OUTFLOW	Ni	0.001	0.214	0.447	0.308	0.219	0.190	0.172	0.113	0.000	0.041	0.052	0.000	0.058	0.069	0.000	0.072	0.087	0.447	0.5	1
POLISHING POND OUTFLOW	Zn	0.005	0.023	0.068	0.054	0.058	0.071	0.066	0.059	0.000	0.031	0.042	0.000	0.048	0.059	0.000	0.063	0.062	0.071	0.5	1

enough to drain the local groundwater around it and to capture the seepage from the TWRMF. A good level of maintenance of this channel will be provided as any sustained channel blockage, local infilling or pump malfunction will reduce the effectiveness of the channel.

Horizontal seepage through the deposited tailings will be captured by a filter drain system to be constructed within the perimeter embankment of the TWRMF. The filter drain system will discharge to the interceptor channel close to the base of the embankment. The collected water in the interceptor channel will be pumped back into the TWRMF.

2.14.4 Control Systems

Automatic gauging stations will be installed upstream and downstream on Minago River and Oakley Creek. These gauging stations will provide a continuous record of water levels and flows in Minago River and Oakley Creek.

2.14.5 Effluent Monitoring

Monitoring programs will be implemented to assess project effects. Potential project effects on water quality in local watersheds during the operational and closure phases may be caused by the following:

- discharge from the Polishing Pond into the Oakley Creek and the Minago River; and
- introduction of sediments (total suspended solids) to receiving waters due to runoff from areas disturbed during mine facility construction.

Baseline and proposed monitoring programs during opeations and closure are summarized below.

2.14.5.1 Baseline Monitoring Program

Surface water quality in watercourses surrounding the Minago Project was assessed by Wardrop (2007) from May to October 2006, URS (2008g) from May to August 2007, and KR Design Inc. from September 2007 to May 2008. Wardrop (2007) monitored water quality in Oakley Creek and Minago Project River while URS (2008g) and KR Design Inc. regularly monitored water quality in Oakley Creek, Minago River, William River, and Hargrave River. One-time assessments of surface water quality were also completed for William Lake, Little Limestone Lake, Russell Lake, and two locations near the confluence of William River and Limestone Bay on Lake Winnipeg. The selected locations for surface water sampling stations were based on:

- a review of topographic maps, orthophoto and drainage features at and surrounding the Minago site;

- consideration of the simultaneous collection of hydrological data, stream sediment and benthic samples during one or more of the surface water sampling events;
- consideration of the selection of representative stations both upstream and downstream of the Project site for the development of long-term sampling stations to monitor long-term trends in surface water quality during the exploration, development, operation and post-closure phases of the Project's mine life.

Water samples were analyzed for field parameters (pH, temperature, conductivity, oxidation-reduction potential (ORP), and dissolved oxygen (DO)), nutrients, major ions, metals, Radium-226, and other physicochemical parameters. Collection methods conformed to the guidelines outlined in the federal Metal Mining Guidance Document for Aquatic Environmental Effects Monitoring (MMER-EEM; Environment Canada, 2002b). Details are provided in Section 7.5: Surface Water Quality.

2.14.5.2 Chemical Monitoring

Chemical monitoring will be undertaken during the operational and closure phases, in accordance with permit and MMER requirements. An application for amendment setting out a revised program for approval will be submitted to the respective agency. In addition to meeting permit requirements of the day, monitoring will be limited in scope to those parameters given in Schedule 4 of the MMER. In accordance with MMER, monitoring will continue as per the proposed program for three additional years. During the closure phase, chemical monitoring data will be reviewed for continual improvement.

2.14.5.3 Biological Monitoring

Biological monitoring will be undertaken to meet permit and MMER related requirements. Toxicity testing will be part of the biological monitoring program and will continue as required. In accordance with MMER, monitoring will continue as per the proposed program for seven additional years.

2.14.5.4 Physical Monitoring

Monitoring programs to assess physical parameters will be undertaken during the operational, closure, and post closure phases. In the event of any significant improvement or deficiency during the post closure monitoring phase (expected to last 4-6 years after closure), Victory Nickel will apply for an amendment setting out a revised program for approval.

2.14.5.5 Operational and Closure Water Quality Monitoring Programs

Victory Nickel intends to design its environmental protection programs in an environmentally sensitive manner to ensure that the above effects do not occur. However, in order to assess impacts, Victory Nickel will undertake a regional study during the operations and after closure. This regional study area will include water bodies and watersheds beyond the local project area that reflect the general region to be considered for cumulative effects and that provide suitable reference areas for sampling. The regional study will encompass water sampling in:

- Minago River downstream and upstream of the Polishing Pond discharge;
- Hargrave River;
- upstream and downstream of the Oakley Creek and William River confluence;
- William River;
- Limestone Bay; and
- Cross Lake.

Monitoring sites have already been established as outlined in Table 2.14-24 and Figure 2.14-11. These sampling sites will also be used during the operations, TS, SI and closure stages.

2.14.5.6 Proposed Water Quality Characterization

The proposed water quality monitoring parameters and associated minimum detection limits are given in Table 2.14-25. The respective QA/QC criteria and procedures for closure will be similar to the ones used during operations.

A water quality monitoring program was established as part of the environmental baseline studies. These streams will continue to be sampled during the operational and closure phases to determine potential impact(s) over time. The stations that will be sampled during the closure phase are provided in Table 2.14-26.

Table 2.14-24 Sampling Locations

Victory Nickel Sample Location (as of Sept. 15, 2007)	UTM (NAD 83)		UTM (NAD 83)		Description
	Northing	Easting	Latitude	Longitude	
HRW1	6028072	495606	54°24.041' N	99°04.051' W	Hargrave River immediately west of Highway 6
MRW1	6005277	488671	54°11.721' N	99°10.420' W	Minago Project River immediately west of Highway 6
MRW2x	6001166	472571	54°09.470' N	99°25.206' W	Minago Project River near Habiluk Lake (~ 100 m downstream of MRW2)
MRW3	6007895	494274			Minago Project River downstream of Highway 6 near powerline cut
OCW1	5990510	489322	54°03.762' N	99°09.786' W	Oakley Creek immediately east of Highway 6
OCW2	5990961	487463	54°04.002' N	99°11.492' W	Oakley Creek immediately downstream of north tributary
OCW3	5990892	487230	54°03.965' N	99°11.707' W	Oakley Creek immediately upstream of north tributary
WRW2x	5987162	495416	54°01.963' N	99°04.199' W	William River approx. 6 km upstream of the Oakley Creek confluence
WRW1x	5986554	498523	54°01.637' N	99°01.350' W	William River approx. 100 m downstream of the Oakley Creek confluence
WRAOC	5986647	498452	54°01.685' N	99°01.416' W	William River approx. 50 m upstream of the Oakley Creek
OCAWR	5986744	498457	54°01.738' N	99°01.414' W	Oakley Creek approx. 50 m above William River
WRALSB	5969206	503935	53°52.278' N	98°56.410' W	William River approx. 100 m above Limestone Bay
LSBBWR	5968889	504092	53°52.107' N	98°56.262' W	Limestone Bay approx. 250 m below William River
Little Limestone Lake	5954922	478725			Little Limestone Lake (at end of road)
Russell Lake	5967117	482571			Russell Lake (at end of road)
William River (Winter)	5973774	485141	53°54.730' N	99°13.574' W	William River east of Highway 6
William River at Road	5973791	485078			William River west of Highway 6
William Lake	5973831	479083			William Lake at end of access road
Polishing Pond	TBA*	TBA*	TBA*	TBA*	Polishing Pond Outflow to Receiving Environment during Closure

Note: * TBA To Be Announced

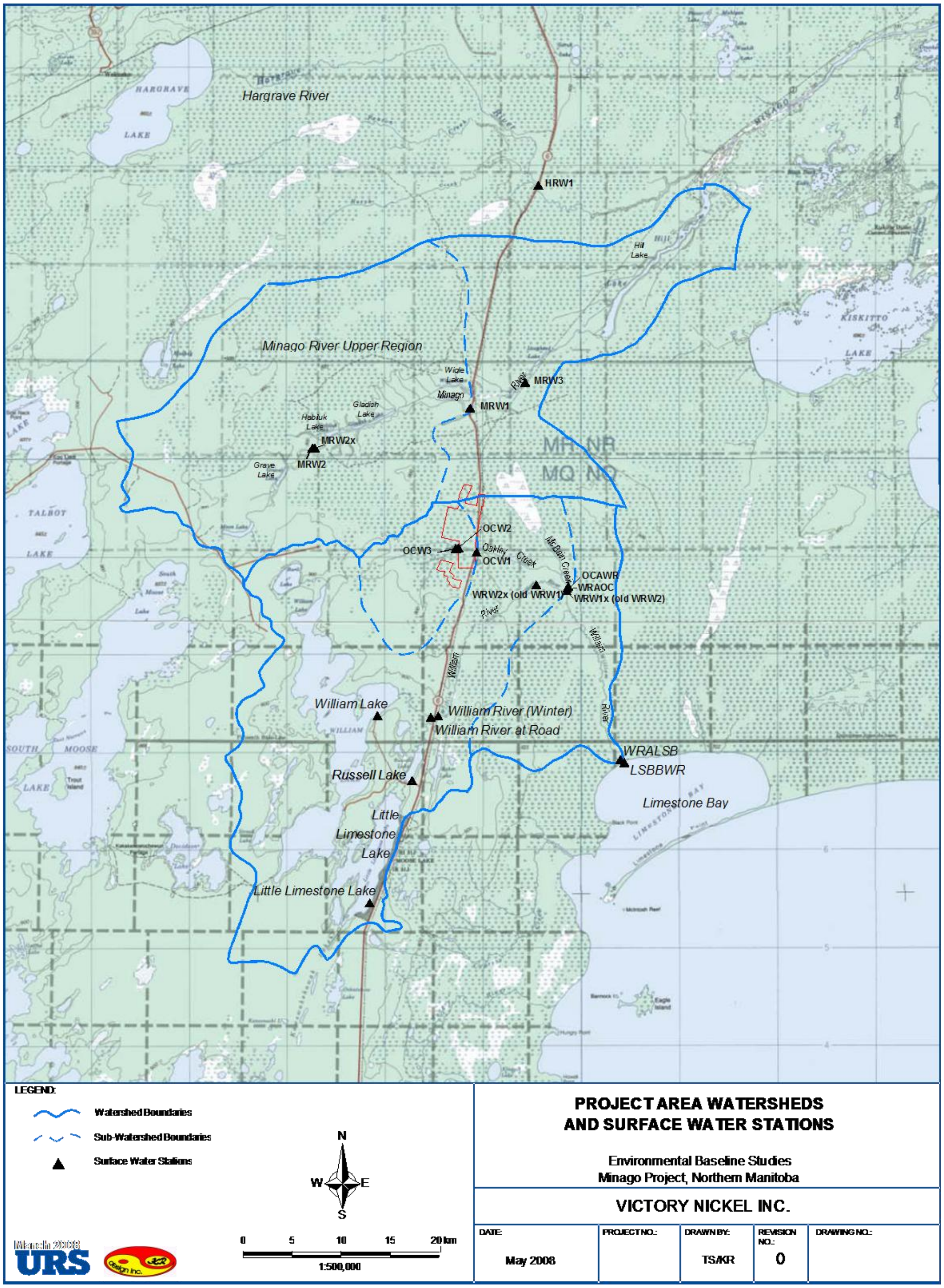


Figure 2.14-11 Minago Project – Site Watersheds

Table 2.14-25 Water Quality Monitoring Parameters and Detection Limits

Parameter		Detection limit (mg/L)	Analytical Method
Aluminum, total and dissolved	Al	0.001	ICP / ICP MS
Antimony, total and dissolved	Sb	0.00005	ICP / ICP MS
Arsenic, total and dissolved	As	0.00005	ICP / ICP MS
Barium, total and dissolved	Ba	0.00005	ICP / ICP MS
Beryllium, total and dissolved	Be	0.0005	ICP / ICP MS
Bismuth, total and dissolved	Bi	0.0005	ICP / ICP MS
Boron, total and dissolved	B	0.001	ICP / ICP MS
Cadmium, total and dissolved	Cd	0.00005 to 0.02	ICP / ICP MS
Calcium, total and dissolved	Ca	0.05	ICP / ICP MS
Chromium, total and dissolved	Cr	0.0001	ICP / ICP MS
Cobalt, total and dissolved	Co	0.0001	ICP / ICP MS
Copper, total and dissolved	Cu	0.0001	ICP / ICP MS
Iron, total and dissolved	Fe	0.01	ICP / ICP MS
Lead, total and dissolved	Pb	0.00005	ICP / ICP MS
Lithium, total and dissolved	Li	0.001	ICP / ICP MS
Magnesium, total and dissolved	Mg	0.05	ICP / ICP MS
Manganese, total and dissolved	Mn	0.00005	ICP / ICP MS
Mercury (total) , total and dissolved	Hg	0.00005	Cold Oxidation (CVAAS)
Molybdenum, total and dissolved	Mo	0.00005	ICP / ICP MS
Nickel, total and dissolved	Ni	0.0001	ICP / ICP MS
Phosphorus, total and dissolved	P	0.05	ICP / ICP MS
Potassium, total and dissolved	K	0.2	ICP / ICP MS
Selenium, total and dissolved	Se	0.0005	ICP / ICP MS
Silicon, total and dissolved	Si	0.05	ICP / ICP MS
Silver, total and dissolved	Ag	0.00001	ICP / ICP MS
Sodium, total and dissolved	Na	2	ICP / ICP MS
Strontium, total and dissolved	Sr	0.0001	ICP / ICP MS
Thallium, total and dissolved	Tl	0.00005	ICP / ICP MS

Table 2.14-25 (Cont.'d) Water Quality Monitoring Parameters and Detection Limits

Parameter	Detection limit (mg/L)		Analytical Method
Tin, total and dissolved	Sn	0.0001	ICP / ICP MS
Titanium, total and dissolved	Ti	0.01	ICP / ICP MS
Vanadium, total and dissolved	V	0.001	ICP / ICP MS
Zinc, total and dissolved	Zn	0.001	ICP / ICP MS
Total alkalinity	$CaCO_3$	1	Titration to pH=4.5
Ammonia	N	0.005	Colorimetry
Nitrate	N	0.005	Ion Exchange Chromatography
Nitrite	N	0.001	Colorimetry
Nitrite + nitrate	N	0.005	Ion Exchange Chromatography
Sulphate	SO_4	0.03	Ion Exchange Chromatography
Total dissolved solids		1 to 5	Filtration/Gravimetric
Total suspended solids		1 to 5	Filtration/Gravimetric
Turbidity		1.0 (NTU)	Nephelometric
Conductivity		1.0 (µS)	Conductivity cell
pH (RelU)		0.1 (RelU)	Potentiometric
Cyanide (total)	CN	0.005	Distillation/UV Detection
Fluoride	F	0.02	Colorimetry
Chloride	Cl	0.5	Colorimetry

Table 2.14-26 Sediment and Surface Water Monitoring Stations

VICTORY NICKEL Water Quality Monitoring Stations	Description	Monitoring Frequency – Water Quality – during Operational Phase	Monitoring Frequency – Water Quality – during Closure and Post Closure Phases	Monitoring Frequency – Flow – during Operational Phase	Monitoring Frequency – Flow – during Closure and Post Closure Phases	Duration – No. of Years	Applicable Regulations
HRW1	Hargrave River immediately west of Highway 6	M	Q	Q	Q	6	IP
MRW1	Minago River immediately west of Highway 6	M	Q	M	Q	6	IP
MRW2	Minago River near Habiluk Lake	SA	A	SA	A	6	IP
MRW2X	Minago River near Habiluk Lake (100 m downstream of MRW2)	Q	A	Q	A	6	IP
MRW3	Minago River downstream of Highway 6 near powerline cut	M	Q	M	Q	6	CCME / MB Tier II
OCW1	Oakley Creek immediately east of Highway 6	M	Q	M	Q	6	CCME / MB Tier II
OCW2	Oakley Creek immediately downstream of north tributary	M	A	M	A	6	IP
OCW3	Oakley Creek immediately upstream of north tributary	M	A	M	A	6	IP
WRW2X	William River approx. 6 km upstream of the Oakley Creek confluence	SA	A	SA	A	6	IP
WRW1X	William River approx. 100m downstream of the Oakley Creek confluence	M	A	M	A	6	IP
WRAOC	William River approx. 50 m upstream of the Oakley Creek	Q	Q	Q	Q	6	IP
OCAWR	Oakley Creek approx. 50 m above William River	Q	Q	Q	A	6	IP
WRALSB	William River approx. 100 m above Limestone Bay	Q	Q	Q	Q	3	IP
LSBBWR	Limestone Bay approx. 250 m below William River	Q	Q	Q	Q	1	IP
Little Limestone Lk	Little Limestone Lake (at end of road)	A	A	A	A	1	IP
Russell Lake	Russell Lake (at end of road)	A	A	A	A	1	IP
William River (Winter)	William River east of Highway 6	A	A	A	A	1	IP
William River at Road	William River east of Highway 6	A	Q	Q	Q	6	IP
William Lake	William Lake at end of access road	A	A	A	A	1	IP
Polishing Pond	Polishing Pond Outflow	M	M	M	Q	6	MMER

Note: A= Annually, Q= Quarterly, SA= Semi Annually; IP= Internal Programs; MMER= Metal Mines Effluent Regulation Monitoring Point (Polishing Point Outflow); CCME/MB Tier II Monitoring Station (OCW1 and MRW3).

2.15 Site Facilities and Infrastructure

The proposed project will be comprised of an open pit mine, an ore concentrating plant, a frac sand plant, and supporting infrastructure. The Ore Concentrating Plant will process 10,000 tonnes per day of ore through crushing, grinding, flotation, and gravity operations to produce nickel concentrate. The Frac Sand Plant will produce 1,500,000 t/a of various sand products including 20/40 and 40/70 frac sand, glass sand, and foundry sand products. The general site layout is illustrated in Figure 2.1-2.

The proposed infrastructure for the Project will include (adapted from Wardrop, 2009b):

- site haul and access roads and laydown areas;
- open pit (described in Section 2.9);
- Mill Process and Frac Sand Plant (described in Section 2.10);
- Crusher and Concentrator Facilities;
- a Tailings and Ultramafic Waste Rock Management Facility (TWRMF);
- waste rock and overburden disposal dumps;
- water and wastewater facilities, including an open pit dewatering system, site de-watering systems with associated pipelines and pumping stations, a sewage treatment system, a potable water treatment plant, a Polishing Pond and site infrastructure piping;
- a fuelling storage and dispensing facility for mobile equipment;
- equipment repair and maintenance facilities;
- miscellaneous service buildings including emergency services building, cold storage building, process and fresh water pump house, security guardhouse and scale house;
- an explosives storage;
- electrical power supply, transformation and distribution;
- modular facilities, including mine site staff dormitories, wash/laundry facilities, staff kitchen/cafeteria, mine dry, a modular office complex and a recreational facility;
- storm water management systems;
- life safety and security systems;
- data and communication systems; and
- other refuse disposal.

The modular camp, which is designed to accommodate 300 people, will form the basis of the accommodation for the construction workforce.

All infrastructure facilities will be located at least 300 m from Highway 6, to provide a visual tree-line barrier from traffic to the Minago operation. Only the guard house and scale house will initially be visible from Highway 6. Since the tailings dam will be of limited height and will be set back from the road, the tree lined barrier will limit visibility (Wardrop, 2009b).

The major infrastructure facilities such as the mill, crushing facility and truck garage will be located in the northwest corner of the site where the overburden thickness is minimal. This area has the highest site elevation therefore eliminates concerns on site drainage and flooding (Wardrop, 2009b).

The minimum distance requirements to separate the explosive plant operations from other operations and the necessary minimum clearances to the 230 kV and other electrical lines have been taken into account in the site layout.

2.15.1 Site Roads

Site roads will be located throughout the site to provide access to all operational areas of the mine (Wardrop, 2009b). Roads will vary in width and general cross section depending on the location, staging and ultimate use of the roadway. Initial road widths of 6 m, 8 m, 12 m, 20 m and 30 m will be used throughout the mine site and will be constructed so that the finished roadway surface is a minimum 0.8 m – 1.0 m above the surrounding ground elevation.

Haul roads will facilitate movement of the 218 tonne trucks with the required clearances. The roads carrying highway truck traffic for incoming supplies and materials and outgoing ore concentrate will be designed to accommodate a Super B-Train loading (GVW 62,500kg) and roads carrying mining ore will accommodate GVW 324,000kg haul trucks (Wardrop, 2009b).

A number of the roads will have elevated berm sections to accommodate utilities/pipelines. The elevated berms will prevent vehicles from wandering across the roadway and into the utilities themselves.

Parking areas will be illuminated and equipped with electrical plugs where necessary.

The intersection of the mine site access road with Highway 6 will require improvements to accommodate turning and slow moving truck traffic entering and exiting the site. The improvements will include pavement widening to create auxiliary acceleration/deceleration lanes.

2.15.2 Crushing and Concentrator Facilities

A crusher building was designed with a footprint of 19 m x 12 m. The crusher will be 51 m high and has a truck dumping area on one side of the building at a relative height of 30 m. The crusher will be contained in a fully enclosed building and has been designed to accommodate a 45 tonne bridge crane. The crusher foundation has been designed as a thick slab, assumed to be sitting on or near the bedrock layer.

The concentrator building is designed as a main building (27 m wide x 150 m long x 29 m high). This main area will house a ball mill, sag mill, pebble crusher, conditioner, and flotation units. Four separate lean-to buildings are also included in the design of the concentrator. The first lean-to building will be 9 m wide x 22.5 m long x 9.5 m high and houses the switch rooms and motor control centers (MCCs). The second lean-to building will be a 5.5 m wide x 4.5 m long x 2.44 m high and is designed as an unloading area. The third lean-to building will be 16 m wide x 60 m long x 23 m high and houses the reagent area. The final lean-to building associated with the concentrator will be 14 m wide x 90 m long x 26 m high and houses the stock tank, concentrate thickener, and a storage area.

2.15.3 Tailings and Ultramafic Waste Rock Management Facility

The Tailings and Ultramafic Waste Rock Management Facility (TWRMF) will be a key component of the water and waste management system at Minago for liquid waste, nickel mill and Frac Sand Plant tailings, und ultramafic waste rock. Mine waste contained in the TWRMF will be stored subaqueously.

Submerging mine waste containing sulphide minerals, or “subaqueous disposal”, is practiced at many metal mines to keep oxidative rates at a minimum and to minimize metal leaching. Based on geochemical work done to date, Minago’s nickel tailings will contain low sulphide levels and were deemed to not become acid generating (URS, 2008a). Sulphide levels were less than or equal to 0.07 % in the Master tailings samples tested. However, Minago’s ultramafic waste rock is potentially acid generating (URS, 2009i).

The TWRMF will receive water from the mill tailings thickener, sewage treatment plant, waste frac sand and the underflow from the frac sand clarifier. Typical tailings water inputs include 503 m^3/h from the process plant and 118 m^3/h from the waste receiving pump box. The waste receiving pump box will contain 100.4 m^3/h of waste frac sand, 12.4 m^3/h of underflow from the frac sand clarifier and 5 m^3/h from the sewage treatment plant (Wardrop, 2009b).

The following design considerations were taken into account for the TWRMF (Wardrop, 2009b):

- Ultramafic waste will be co-deposited with tailings in the TWRMF. This will contain all contaminants into a single area without contaminating other areas. The containment structure (21 m high) will be built initially followed by the construction of a perimeter ramp inside the TWRMF area. This will allow for co-depositing of tailings and ultramafic waste. The tailings will be deposited onto the ultramafic waste to fill in voids within the rock.
- To support the Tailings and Ultramafic Waste Rock Management Facility (TWRMF), a ring main pipe, a floating barge pumping station, and three perimeter ditch pumping stations will be provided. A ring main pipe with spigots will be located along the entire perimeter ramp and placement of tailings will be accomplished by opening and closing of valves along a ring main pipe to eliminate accumulations of solids in a particular area and allow for uniform discharge.

- The tailings deposition will create a decant pond sized for not less than five days of retention time. A minimum water depth of 1.5 m will be maintained in the pond at all times.
- Decant water from the tailings pond will be pumped to the Polishing Pond and flood retention area for subsequent water recycling or discharge to the receiving environment.
- Seepage ditches around the perimeter of the TWRMF will collect the seepage and runoff and transfer the water back into the facility.
- The TWRMF site will be located in permanently-flooded terrain. The construction of the TWRMF dam will be preceded by construction of roads surrounding the site.
- The pond in the TWRMF will be operated under average precipitation conditions, but with the barge pumps capable of pumping a 1-in-20 year, 5 day major rainfall event. The maximum discharge rate will be based on the expected requirements for a major rainfall event over a two week period. The nominal discharge rate will be increased and decant water will be pumped over a two week period during such an event.
- The TWRMF will provide adequate volume for containment of tailings, ultramafic waste and supernatant water. Ice formation over the tailings due to discharge in subfreezing temperatures during winter operations is envisaged, and contingency storage capacity has been provided in the design.

Seepage from the TWRMF will be collected in a perimeter ditch and pumped back into the facility by three 15 hp submersible pumps. Three pumping stations will be located along the low-elevation east-side ditch area. The tailings water from the TWRMF pond will be pumped to the Polishing Pond and flood retention area by three 60 hp vertical turbine pumps, each capable of pumping 530 m^3/h. These pumps will be mounted on a floating barge pump station (Wardrop, 2009b).

2.15.4 Waste Rock and Overburden Disposal Dumps

Ultramafic waste rock will be co-disposed with tailings in the Tailings and Ultramafic Waste Rock Management Facilty (TWRMF). Non-reactive dolomite and country waste rock will be hauled to the designated dump areas. No water quality problems are anticipated from these dump areas since the rock is non-reactive and will not contain contaminants. The majority of the runoff will discharge directly to the environment while a minimal amount of rainfall will runoff to the roadway ditches and eventually to the overburden settling pond. There are no anticipated problems with TSS during a major event due to the nature of coarse waste rock (Wardrop, 2009b).

The overburden dump area will be surrounded by a containment berm. Weirs will allow for discharge of water to a settling pond. Due to restraints on total suspended solids (TSS), the settling pond will be used for settlement prior to discharge to the Oakley Creek watershed. Flocculent addition may be required to meet water quality standards. Placement of material in the overburden dump will be complete within the first two years of construction and re-vegetation of the surface will occur immediately after completion (Wardrop, 2009b).

During a 1-in-20 year, 5-day major rainfall event, the overburden settling ponds will be used for settlement and storage with the presence of an overflow line to discharge benign rainfall. Once the vegetation is established, it is anticipated that the runoff will be benign and will meet TSS water quality standards. The areal boundaries of the Overburden Disposal Facility dump will contain a permeable dyke/road, which will contain a filter cloth and sand bed to filter the water through the roadway. Due to the benign nature of the runoff, water will be discharged to the environment instead of the flood retention area (Wardrop, 2009b).

2.15.5 Water and Wastewater Facilities

The water and wastewater management components at Minago will include:

- dewatering wells to dewater the open pit area;
- a water treatment plant to produce potable water;
- a sewage treatment facility for on-site grey water and sewage;
- mill and Frac Sand Plant tailings and effluents that will be discharged into a Tailings and Ultramafic Waste Rock Management Facility (TWRMF);
- a Tailings and Ultramafic Waste Rock Management Facility (TWRMF) that will store tailings and ultramafic waste rock permanently and effluents from various site operations temporarily;
- waste rock dump seepages that will be discharged into the TWRMF or the receiving environment depending on their water quality;
- overburden dump runoff that will be discharged directly into the receiving environment (if it meets discharge requirements);
- an open pit dewatering system that will ensure safe working conditions in and around the open pit;
- a Polishing Pond and flood retention area to serve as holding pond for water that will either be recycled to site operations or discharged to the receiving environment (if it meets discharge water standards);
- a site drainage system to prevent flooding of site operations;
- site wide water management pumping systems; and
- discharge pipelines to Minago River and Oakley Creek to discharge excess water from the Polishing Pond / flood retention area and the country rock, dolomite rock, and overburden dumps to the receiving environment.

Due to the complexity of the water and wastewater management system, its components, flow volumes, seasonality and elemental concentrations are presented and discussed in a separate subsection (Section 2.14). However, the proposed sewage treatment, potable water treatment, site infrastructure piping and dewatering facilities are also outlined below.

2.15.5.1 Sewage Treatment Plant

The domestic sewage generated on the site will be collected by sanitary sewers and conveyed to an extended aeration mechanical sewage treatment plant. The sewage treatment plant will use an extended aeration system, supplied by Canwest Tanks and Ecological Systems Ltd. or equivalent (Wardrop, 2009b).

The proposed plant meets Manitoba Conservation requirements, and will meet 25/25 mg/L Five-day Biochemical Oxygen Demand (BOD_5) and Total Suspended Solids (TSS) targets. The plant design incorporates nitrification to reduce ammonia concentrations in the effluent to within Manitoba Conservation's winter and summer restrictions. Nitrogen or phosphorous removal is not expected to be required, since the discharge will flow into the catchment area of Lake Winnipeg (Wardrop, 2009b).

The sewage treatment plant will be located east of the maintenance building to allow all sewage to flow by gravity to the plant. A separate sewage pumping station with a fibreglass tank will be located at the modular complex facility to pump the raw sewage from the complex building to the sewage treatment plant.

The treatment plant will accommodate 450 people at 230 L/capita/day plus 10% for the water treatment plant backwash. The average daily flow will be 113,800 L/day (Wardrop, 2009b). The per capita BOD_5 contribution will be 0.091kg/capita/day. The daily BOD_5 loading will be 40.9 kg BOD_5/day (Wardrop, 2009b).

The tanks, which will be buried, will be constructed with fibreglass materials that meet CSA BL66 standards. The effluent will be disinfected using ultraviolet (UV) radiation (Wardrop, 2009b).

The treated effluent will be discharged to the waste receiving pump box, and then discharged to the TWRMF. The treatment plant will include an on-line lockable refrigerated composite sampler that will available to Manitoba Conservation for independent effluent sampling. Treated effluent sampling and analyses will be perforemd on a monthly basis to detect BOD_5, ammonia, TSS, and fecal coliforms (Wardrop, 2009b).

A grease trap will be installed at the discharge from the camp kitchen prior to the connection with the sewer system (Wardrop, 2009b).

The domestic wastewater sludge storage tank will be periodically de-sludged using three submersible pumps installed in the sludge storage tank. The sludge will be pumped into a tanker truck and hauled to the lagoon at Grand Rapids for disposal. The estimated sludge production is 0.15 to 0.23 m^3/h (Wardrop, 2009b).

An insulated pre-fabricated building will house the blowers, control panel, the lockable, refrigerated composite sampler and similar equipment. The building will be skid-mounted and installed on a crushed stone base. There will be no special electrical code requirements since none of the electrical equipment will be exposed to sewage or sludge (Wardrop, 2009b).

2.15.5.2 Potable Water Treatment Plant

The potable water supply will be drawn from the fresh/fire water storage tank and the ground water wells. Potable water will be used for the safety shower/eye wash stations, personal consumption, washrooms, canteen and dry. Potable water will not be used for fire water, process water or general plant distribution. Potable water will be pumped to the modular complex and the maintenance building, primary crusher building, crushed ore delivery tunnel, security building, and frac sand plant (Wardrop, 2009b).

Since raw water will be supplied from a confined aquifer, it is not considered Groundwater Under Direct Influence of Surface Water (GUDI). Accordingly, no special preventitive precautions will be needed for giardia, cryptosporidium or similar parasites (Wardrop, 2009b).

Potable water treatment will consist of a bank of manganese greensand pressure filters to remove iron and manganese to less than 0.05 mg/L and 0.3 mg/L, respectively. These aesthetic limits are recommended by Manitoba Conservation as well as Health Canada's Canadian Guidelines for Drinking Water Quality. The filtration rate will be 6.1 m^3/hr per m^2 (Wardrop, 2009b).

Post-chlorination treatment will be performed by sodium hypochlorate (bleach) with an inline residual chlorine analyser. If the chlorine residual exceeds the range of the high and low level set points, an alarm will alert the operator to review the problem and adjust the chlorine levels appropriately (Wardrop, 2009b).

The treatment plant will include enough treated water storage to accommodate an average day's consumption, expected to be 4.3 m^3/hr; peak demand flows are expected to reach up to 17.3 m^3/hr. The treatment plant will be located west of the modular complex building since the complex building requires the greatest amount of potable water (Wardrop, 2009b).

2.15.5.3 Site Infrastructure Piping

Water supply pipes and sewers will be High Density Polyethylene (HDPE) and will be buried on a benched part of the roadways to prevent freezing. In high density peat areas, concrete pipe weights may be required to secure the pipes and prevent flotation (Wardrop, 2009b).

The domestic sewers will be 50 to 200 mm diameter low pressure force mains in some areas. Gravity sewers will be utilized in areas with suitable ground conditions (Wardrop, 2009b).

2.15.5.4 Dewatering Facilties

Open pit dewatering will be accomplished by perimeter groundwater pumps as well as open-pit centrifugal and submersible pumps to properly dewater the pit (Wardrop, 2009b).

The groundwater pumps will consist of twelve 75 hp Grundfos groundwater pumps which will discharge approximately 35,000 m^3/d directly into the retention area while approximately 5,000 m^3/d will be diverted to the fresh water tank (Wardrop, 2009b).

The open-pit will be de-watered by the use of 11 centrifugal pumps and 6 submersible pumps. The dewatering pumps were sized to accommodate a 1-in-20 year, 5-day major rainfall event, and to eliminate down times within the pit due to flooding and will allow for the pit to be dewatered more rapidly. Pumping stations will be located on designated levels throughout the pit to optimize head loss and pipe lengths. The open pit dewatering will be performed by three separate pumping loops in series and will discharge to the Polishing Pond and flood retention area (Wardrop, 2009b).

2.15.6 Fuelling Storage and Dispensing Facility

A fuel storage facility will be centrally located within the industrial area. The diesel fuel storage capacity for the mining operation will be 380,000 L, which includes the fuel requirements for explosives (Wardrop, 2009b). The fueling system will consist of four 95,000 L above-ground double-walled diesel fuel tanks, a diesel fuel pumphouse, and a receiving station.

The fuel storage facility will be self-contained to ensure that inadvertent spills do not end up into the receiving environment. The facility will be equipped with a spill kit and will be inspected on a regular basis. Fuel will be supplied by a third party.

Bulk quantities of petroleum hydrocarbons will be stored and handled in accordance with Manitoba Regulation 188/2001 and any subsequent amendments.

Standard vehicles will be serviced using a dual-fuel dispensing unit and one 4,500 L double-walled diesel fuel tank and one 4,500 L double-walled gasoline tank (Wardrop, 2009b).

2.15.7 Miscellaneous Service Buildings

Miscellaneous service buildings will include an emergency services building, a process and fresh water pump house, a cold storage building, equipment repair and maintenance facilities, a fire protection water pump house, and a security guardhouse and scale house.

The emergency vehicle garage will be a pre-engineered building with an area of approximately 240 m^2. The garage will house an ambulance and a fire truck and will have one small office and storage space for emergency equipment (Wardrop, 2009b).

The process and freshwater pumphouse will be two pre-engineered buildings side by side with a combined area of approximately 170 m^2. The pumphouse complex will be located east of the ore processing facility. A monorail will be installed above each pump system to facilitate maintenance.

A 950 m^2 cold storage warehouse will be located south of the general maintenance building.

The general maintenance building will include (Wardrop, 2009b):

- seven heavy vehicle repair bays including four drive-through bays;
- a light vehicle repair bay, a tire bay, a welding bay, and a wash bay;
- a lube storage facility;
- a machine/hydraulic shop, a fabrication/welding shop, an electrical shop, and an instrumentation shop;
- a 1,290 m^2 storage warehouse;
- five offices, a lunch room and washroom facilities; and
- an upper level mezzanine with mechanical and compressor rooms.

A fire protection water pump house will be located directly beside the fresh water tank. In the event of a fire, the fire water pumps will discharge water from the fresh water tank to the appropriate area. Fire protection will be required at the modular complex building, frac sand plant, mill, maintenance building, fueling area, and primary crusher building.

The security guardhouse and scale house will be located at the entrance to the site, near Highway 6. The guardhouse and scale house will be a single-storey 3.6 m x 6 m modular trailer complete with a washroom facility (Wardrop, 2009b).

2.15.8 Explosive Storage

All explosives will be handled, transported and disposed of in compliance with the Explosive Act.

2.15.9 Power Supply

The primary source of electrical power will be the Manitoba Hydro 230 kV line along the east side of Highway 6. From the connection at Highway 6, a 6-km, 230 kV power transmission line will feed the main substation located to the west of the process plant in the northwest corner of the site. The connection from the Manitoba Hydro 230 kV line will be provided with gas-filled isolation switches (Wardrop, 2009b).

The main substation will consist of two main transformers rated at 50 MVA each capable of supplying the full load. The transformers will transform the power down from 230 kV to 13.8 kV to the main 13.8 kV switch room via metal clad switchgear (Wardrop, 2009b).

The electrical system will be sized and configured for full redundancy, allowing the transformers to operate in parallel or individually while maintaining full production. Each transformer will be able to accommodate the full operational loads in the event of a failure of the other. The main sub-station will be protected by a secure chain link fence surrounding a crushed stone bed for easy maintenance and to ensure effective drainage (Wardrop, 2009b).

Power from the main switchgear room will be distributed at 13.8 kV via overhead line to the various distribution centres around the site. Outdoor oil filled transformers will transform the primary 13.8 kV to 6,600 V, 4160 V and 600 V as required (Wardrop, 2009b).

2.15.9.1 Emergency Power

Two diesel generator sets rated at 1.5 MW, 13.8 kV with associated switchgear will be housed in a dedicated building located near the main electrical substation (Wardrop, 2009b). The system will be designed to provide power during the construction phase and emergency power during the operations phase for life sustaining and critical process equipment. The emergency power system will feed the entire plant grid with operators isolating non emergency switchgears to direct the standby power to the critical services. Most importantly, the emergency power would provide essential power to feed the dewatering pumps during a utility power failure. Diesel generators will provide redundancy as the 230 kV primary power feed from the main 230 kV Manitoba Hydro AC Line (Wardrop, 2009b).

2.15.9.2 Estimated Load

The peak connected load is estimated to be 42.4 MW (50 MVA), based upon the power requirements of operations and auxiliary equipment on the site and an average power factor of 0.85. The estimated operating load for the five cost centres including future growth is 30.0 MVA (Wardrop, 2009b).

2.15.10 Modular Building Complex including Accommodation

The following buildings will be part of the modular building complex (Wardrop, 2009b):

- mine site staff dormitories;
- mine staff kitchen/cafeteria;
- mine dry including male and female facilities and shift change rooms;
- mine office complex, and
- recreational facilities.

All modular facilities will have wheelchair access and will be connected with an enclosed walkway. The buildings will be designed for use in a heavy-duty industrial environment, with an expected life of approximately 20 years (Wardrop, 2009b).

The mine site staff dormitories will be sized to accommodate 300 personnel, including the construction crew. The dormitory complex will consist of 120 double sleeper units, 60 single sleeper units and 6 executive suites (Wardrop, 2009b).

The project will employ 422 staff members; however, workers will rotate on a 12-hour shift schedule, and each shift worker will vacate the site once for every 2-week shift period. In addition, some daytime workers that commute from Grand Rapids will not require accomodations. Accordingly, it is not necessary for the dormatories to accommodate all 422 workers.

The kitchen/cafeteria will be sized for 200 personnel and will house food storage and food preparation areas, the kitchen and cafeteria and a kitchen staff office. The kitchen/cafeteria area will be approximately 883 m^2 (9500 ft^2 (50’ W x 190’ L)).

The mine dry will accommodate 306 lockers (102 per mudroom area) with two male and one female facility.

The office complex will accommodate up to 60 personnel. The office complex will be approximately 790 m^2 (8,500 ft^2 (50’ W x 170’ L)) and will form part of the modular dormitory building. The office complex will be accessible from the exterior and interior of the building complex (Wardrop, 2009b).

2.15.11 Storm Water Management

The site storm water management at the Minago Project is designed to accommodate a 1-in-20 year storm event over a 5-day period (120 mm) (Wardrop, 2009b).

Site rainfall will be pumped to the Polishing Pond and retention area, contained in designated area settling ponds, or discharged to the local watershed via runoff. Rainfall onto the plant area, Overburden Disposal Facility, dolomite dump and country rock dump will be benign and is not expected to require treatment. Settling ponds will nonetheless be built to control major events in the Overburden Disposal Facility areas. Seepage from the Tailings and Ultramafic Waste Rock Management Facility (TWRMF) will be collected in a perimeter ditch around the exterior of the facility and will be pumped over the dyke back into it. A Polishing Pond and flood retention area will contain the storm water from the TWRMF, mine dewatering and site runoff. This water will be pumped to the Minago River watershed, and a portion will be diverted back to the process water tank (Wardrop, 2009b).

2.15.11.1 Ultramafic Waste Rock Dump

The ultramafic waste rock will be deposited directly into the TWRMF, limiting the potential contamination to a single area. The TWRMF is designed to accommodate a 1-in-20 year, 5-day major rainfall event. The nominal discharge from this area will be increased and pumped over a two week period during such an event (Wardrop, 2009b).

2.15.11.2 Overburden Disposal Facility

The Overburden Disposal Facility dump area will be surrounded by a containment berm. Weirs will allow water to discharge to the settlings pond, which will be used for storage of excess water and precipitation. Due to restraints on TSS, settling ponds will be used for settlement prior to discharging to the Oakley creek watershed. Flocculent addition may be required to meet water quality standards. Placement of material in the Overburden Disposal Facility will be complete within the first two years of construction and vegetation of the surface will commence immediately after completion (Wardrop, 2009b).

During a 1-in-20 year, 5-day major rainfall event, the settling ponds will be used for settlement and storage with the presence of an overflow line to discharge benign rainfall. Once the vegetation is established, it is anticipated that the rainfall will be benign and will meet TSS water quality standards. The area boundaries of the Overburden Disposal Facility dump will contain a permeable dyke/road which will contain a filter cloth and sand bed to filter water through the roadway. Due to the benign nature of the runoff, water will be discharged to the environment instead of the flood retention area (Wardrop, 2009b).

2.15.11.3 Plant Area

The plant area runoff including the frac sand plant and sand storage pile will be clean water and will be discharged directly to the environment. Since the plant area is located in the northwest corner of the site, benign rainfall will runoff through the roadway ditches to the Overburden Disposal Facility settling pond as well as runoff to the Oakley Creek watershed (Wardrop, 2009b).

2.15.11.4 Dolomite and Country Rock Dumps

The non-reactive dolomite and country rock will be hauled to designated dump areas (Figure 2.15.1). The majority of the runoff will discharge directly to the environment, while a minimal amount will runoff to the roadway ditches and eventually to the Overburden Disposal Facility settling pond. There are no anticipated problems with TSS during a major event due to the nature of coarse waste rock (Wardrop, 2009b).

2.15.11.5 Polishing Pond and Flood Retention Area

During a 1-in-20 year, 5-day major rainfall event (120 mm), the Polishing Pond and flood retention area will acquire approximately 550,000 m^3 of water over a surface area of 750,000 m^2, which will produce an average depth of approximately 0.75 m throughout the settling area. The roads surrounding the Polishing Pond and flood retention area will have a minimum height of 1 m (0.75 m depth and 0.25 m freeboard). This height will allow for sufficient water storage capacity for the effects of rainfall on the open pit and TWRMF during a major event (Wardrop, 2009b).

The majority of the site water accumulated in the Polishing Pond and flood retention area will be pumped to the Minago River watershed while 12,000 m^3/d will be diverted to the process water tank as reclaim water. Due to the high head and flow capacities, three 600 hp vertical turbine pumps will be used to generate the flow through an 800 mm (32”) HDPE pipe to the Minago River watershed. In the summer months, the water will be discharged to the Minago River watershed by a distribution manifold, while in winter months the pipe outlet will discharge directly to an open ditch after the distribution manifold (Wardrop, 2009b).

2.15.12 Life Safety and Security Systems

The fire alarm and detection systems will be analog addressable systems from a single manufacturer with proven and reliable technology. The system will integrate all detection and annunciation devices with main annunciation panel located at the security station. The security system will employ proven and reliable technology to integrate door alarms and motion sensors for key areas into a central system monitored at the security station. The system will also provide monitored card access for offices, IT rooms and other secure areas (Wardrop, 2009b).

2.15.13 Data and Communication Systems

The telecommunications design will incorporate proven, reliable and state-of-the-art systems to ensure that personnel at the mine will have adequate data, voice and other communications channels available. The telecommunications system will be procured as a design-build package with the base system installed during the construction period then expanded to encompass the operating plant (Wardrop, 2009b).

The requirements for communications, particularly satellite bandwidth, are a function of the voice and data requirements of the active participants in the project. The expectation is that the need for satellite bandwidth will build to a peak during the plant construction phase, and then taper off slightly as the initial construction crew yields to plant operations (Wardrop, 2009b).

Closed Circuit TV (CCTV) cameras will be installed at various locations throughout the plant, including the primary crushing facility, the stockpile conveyor discharge point, the stockpile reclaim tunnel, the SAG and pebble crushing area, and the concentrate handling building. The cameras will be monitored from the plant control rooms (Wardrop, 2009b).

2.15.13.1 Site Wide Radio Communications System

The site radio communication system will operate in simplex and duplex modes. In simplex mode, only one user may communicate at a time. The system will also be capable of transmitting and receiving both voice and data. Site wide communications system will be comprised of the following (Wardrop, 2009b):

- fixed radios/repeaters,
- portable radios, and
- frequency assignment/approvals.

2.15.13.2 Site Wide Fibre Communications System

The site fibre communication system will be capable of operating in single and multimode depending on the length of fibre run. The fibre trunk will act as the main route of communication for the process LAN, business LAN, VoIP communication, and possibly security communication. The fibre trunk will connect all areas to the process mill and office complex (Wardrop, 2009b).

2.16 Transportation

2.16.1 Existing Access and Roads

The Minago Property is located directly adjacent to Manitoba Provincial Highway 6, a major north-south highway transportation route. The major transportation hubs closest to the Minago site are Winnipeg and Churchill, Manitoba (Figure 2.16-1). To date, the site has only been accessed via a winter road in the winter and by Argo or helicopter in the summer.

The Property may be served by the Hudson Bay Railway Company (HBR), with rail lines accessible from Ponton, MB, approximately 65 km north of the mine site.

Due to the Property's proximity to Provincial Highway 6, Wardrop assumed that all inbound freight for equipment and construction services will arrive by highway transportation. Operational inbound freight was also assumed to arrive via road transport.

2.16.2 Proposed Mine Access Road

The road network to be constructed on the Minago Property will be located in the VNI Mineral Lease Parcel.

In the proposed site layout, illustrated in Figure 2.1-2, roads will be located throughout the site to provide access to all operational areas of the mine. Roads will vary in width and general cross section depending on the location, staging and ultimate use of the roadway. Initial road widths of 6 m, 8 m, 12 m, 20 m and 30 m will be used throughout the mine site and will be constructed so that the finished roadway surface is a minimum 0.8 m – 1.0 m above the surrounding ground elevation (Wardrop, 2009b). Haul roads will facilitate movement of the 218 tonne trucks with the required clearances. The roads carrying highway truck traffic for incoming supplies and materials and outgoing ore concentrate will be designed to accommodate Super B-Train loading (GVW 62,500kg) and roads carrying mining ore will accommodate GVW 324,000kg haul trucks (Wardrop, 2009b).

All roads inpit and around the waste rock dumps and tailings storage facility will be 30 metres in width. The 30 metre roads will allow the trafficking of the 218 tonne trucks. In-pit ramps are designed with an overall width of 30 m. The designed width includes an outside berm at 3.0 m wide and 1.8 m high; ditches at 2.5 m for two-lane traffic to accommodate a 218 tonne haul trucks. All of these 30 m roads will be decommissioned at the end of the mining operations.

The 8 m wide service road network will be for light equipment and not for ore trucks. These service roads include a 10 km road along the discharge pipeline to the Minago River and roads in and around the overburden storage area. All of the service roads will be decommissioned, except for the main access road into the center of the site.

All other 6-20 m wide service roads will be decommissioned, once these roads are not needed anymore.

Source: Wardrop, 2009b

Figure 2.16-1 Minago Shipping Routes

2.16.3 Concentrate Haulage Route

The saleable products of the Minago Mine will include nickel concentrate, two types of fracturing sand, and a flux sand product. It is anticipated that approximately 49,500 t/a of 22.3% nickel concentrate on an average year before transportation losses and 900,000 t/a of Frac Sand Plant products will be marketed annually.

Nickel concentrate may either be hauled by truck to Thompson, MB for smelting or the proposed Railway Siding along the Omnitrax Canada railway line near Ponton, MB or be trucked to Winnipeg for further transport to a suitable smelter for processing (Figure 2.16-2). Wardrop determined that shipping the concentrate by typical highway-type tractor trailer for 223 km (one way) to the smelter in Thompson, MB, is likely the most viable option (Wardrop, 2009b).

A separate study, entitled “Transportation Analysis for the Minago Sand Project” (Wardrop, 2008b), was completed for frac sand products to examine potential methods of bulk transportation from the Minago operation, such as railroad and highway-type haul trucks. It was assumed that the sand products produced at the Minago operation will either be trucked from the mine site directly to buyers, or trucked to a rail siding located at Ponton, MB, where it will be loaded into rail cars for onward shipment (Figure 2.16-2). This siding would be serviced by HBR, which has a working relationship with CN Rail. Alternatively, the sand may be trucked into Winnipeg where both CN Rail and CP Rail lines can be accessed. The Company will not own the facility at Ponton. OmniTrax will own the loadout facility.

2.16.4 Decommissioning Plans

Once the traffic around the site areas is reduced to a point where vehicle access is no longer required, most roads will be decommissioned. However, main access roads to the TWRMF and waste rock storage areas will only be partially decommissioned to permit vehicle access in case of emergency. Partial decommissioning will consist of narrowing the road width to 8 m, but leaving existing culverts in place. Regular decommissioning will consist of removing culverts and replacing them with cross-ditches and swales, ripping and scarifying road surfaces and revegetating them with the Minago custom revegetation mixture.

Access will remain for ATVs or similar transport for monitoring and inspections and with minimal effort vehicle access could be re-established.

Once the railway sidings will no longer be required, it will be decommissioned unless someone wants to take the facility over for further use. The two railside buildings will be removed with the exception of concrete foundations. Concrete foundations will be broken up to ground level and removed from the site. The dismantled materials will be sold to vendors as prevailing market conditions permit and remaining debris will be disposed of in an appropriate manner. Any diesel power gensets will be decommissioned and sold to vendors. Power distribution lines will be

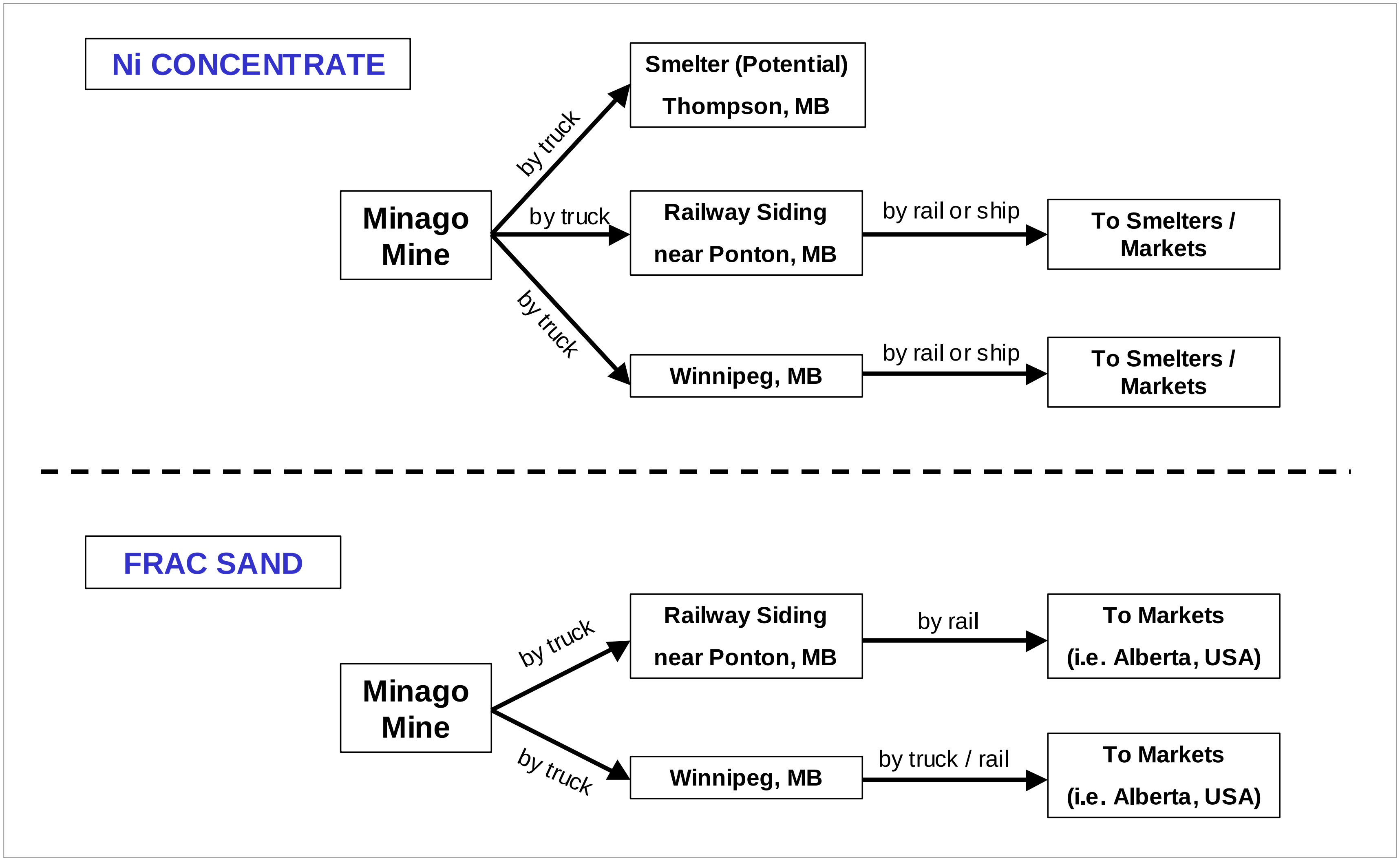


Figure 2.16-2 Concentrate and Frac Sand Haulage Routes

removed from the site and salvaged if possible. The disturbed areas will then be reclaimed using the Minago's revegetation shrubs.

2.16.5 Workforce Logistics

The Minago operation will be staffed by workers on a rotating 14-day basis. The majority of the operational workforce will be comprised of residents from surrounding local communities. Victory Nickel may provide bus service to and from the mine site through a contracted local bus company.

2.16.6 Environmental Impact

There will be no significant increase in environmental impact from these transportation decisions because current and well-established transportation routes and practices already exist on the Provincial Highway 6 corridor.