October 24, 2013

NOTICE TO READER

The attached report entitled “Technical Report on the Dumont Ni Project, Launay and Trécesson Townships, Quebec, Canada” (the “**Technical Report**”) prepared for Royal Nickel Corporation and dated July 25, 2013 has been re-filed to correct the following typographical/proofing errors:

1. In Table 22-4, certain typographical/proofing errors in the numbers have been corrected, making these numbers consistent with the numbers used elsewhere in the Technical Report.

2. In Table 6-3, the mineral resource categories were inadvertently referred to as “Upper” and “Middle” and have been corrected to refer to “Indicated” and “Inferred”.

3. In Section 6.1.5.4, the “preliminary assessment” was inadvertently referred to in the first line of the second paragraph and has been corrected to refer to the “pre-feasibility study”.

4. In the note to Figure 7.20, “Iron Serpentine” was in advertently left out after “4%” and has been added.

5. In the note to Figure 7.21, there was an error in footnote numbering and has been corrected so that “(5)” has been replaced with “(4)”.

6. In the last paragraph of Section 7.7, “Figure 7.22” was inadvertently referred to and has been corrected to refer to “Figure 7.23”.

7. In Section 13.5.6, the number of samples completed was corrected from “75” to “102”.

8. The title of Table 13-33 has been corrected to reflect the data in the table.

Other than as noted above, the Technical Report remains the same in all respects.

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Minerals & Metals

Report No: 2280
Revision Number 0

Royal Nickel Corporation

Technical Report on the Dumont Ni Project, Launay and Trécesson Townships, Quebec, Canada

25 July 2013

Date & Signature Page

Project Name:	Dumont Nickel Project
Title of Report:	Technical Report on the Dumont Ni Project, Launay and Trécesson Townships, Quebec, Canada
Location:	Launay and Trécesson Townships, Quebec, Canada
Effective Date of Report:	25 July 2013
Completion Date of Report:	25 July 2013

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"J.M. Bowen," MAusIMM (CP) (Ausenco Solutions Canada Inc.)	25 July 2013
"S.B. Bernier," P.Geo. (SRK Consulting (Canada) Inc.)	25 July 2013
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1 SUMMARY

1.1 Introduction

Royal Nickel Corporation (RNC) is a mineral resource company headquartered in Toronto, Canada, primarily focused on the exploration, evaluation, development and acquisition of base metal and platinum group metal properties. RNC's principal asset is the Dumont Nickel project (Dumont project) located in the established Abitibi mining camp, 25 km northwest of Amos, Quebec. RNC acquired a 100% interest in the Dumont property in 2007. The mineral claims covering the Dumont deposit are currently held 98% by RNC and 2% by Ressources Québec.

Ausenco Solutions Canada Inc. (Ausenco) was commissioned by RNC in May 2012 to complete the feasibility study (FS) and the NI 43-101 compliant technical report on the project. This technical report was prepared to provide RNC with sufficient information to determine the economic feasibility of developing the Dumont deposit, and to decide whether and on what basis to proceed with construction.

In addition, SRK Consulting (Canada) Inc. (SRK) was engaged to prepare the geology, resource estimate, tailings management, hydrogeology, hydrology, geotechnical and closure planning inputs. Snowden Mining Industry Consultants Inc. (Snowden) was retained for mine design, mine operating costs, mine capital costing, and reserve estimation. GENIVAR Inc. (GENIVAR) was engaged to provide inputs to the environmental and permitting aspects of the project. Golder Associates Ltd. (Golder) contributed to the environmental geochemistry investigations. Norascon contributed to civil engineering design and costing.

The Dumont project is located in the province of Quebec in the municipalities of Launay and Trécesson approximately 25 km by road northwest of the city of Amos, 60 km northeast of the industrial and mining city of Rouyn-Noranda and 70 km northwest of the city of Val d'Or. Amos has a population of 12,584 (2006 Census) and is the seat of the Abitibi County Regional Municipality (Figure 1.1).

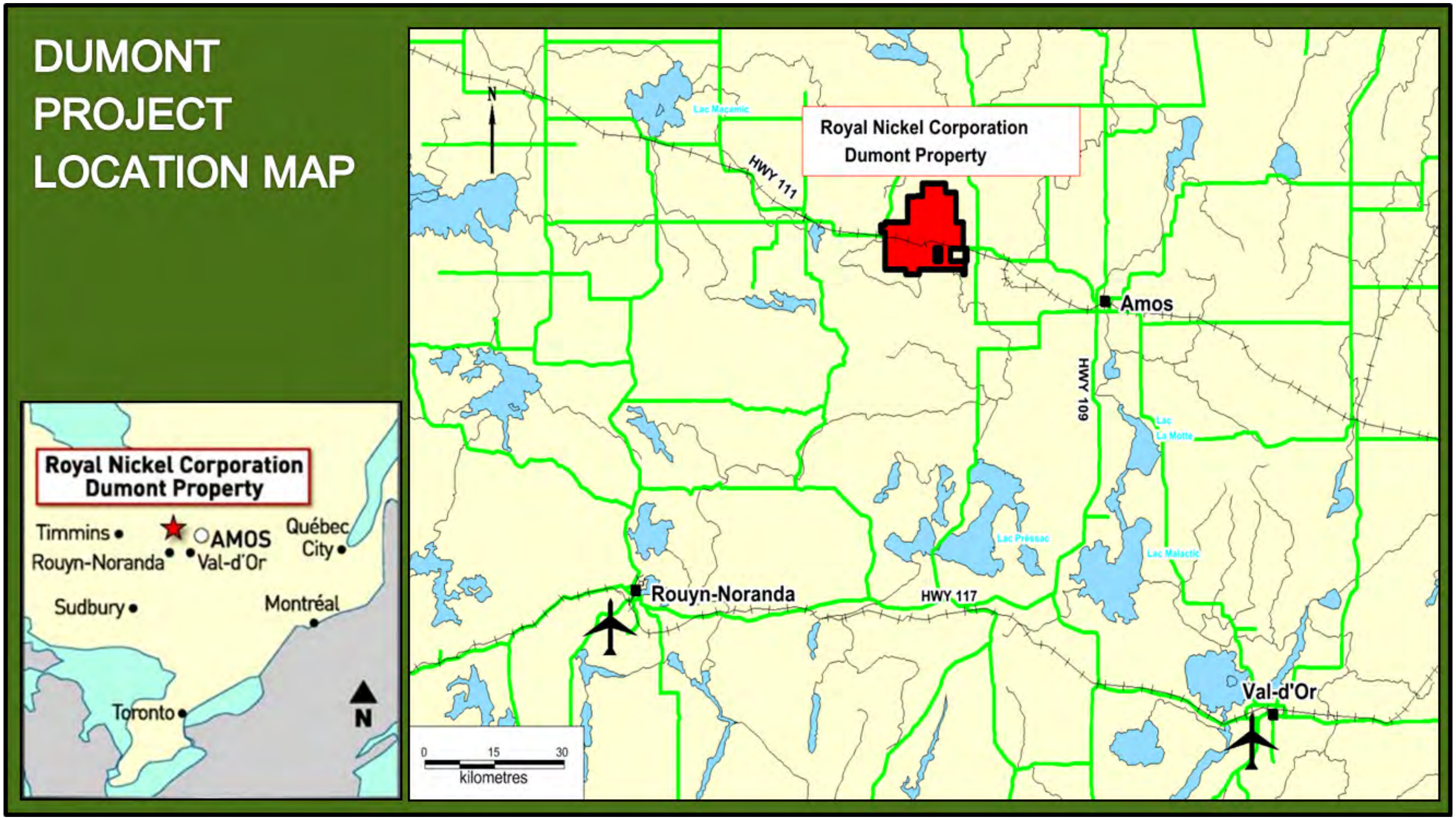
No historical mining or production has been conducted on the Dumont property. However, for the past 100 years, the Val d'Or - Rouyn-Noranda region surrounding the Dumont property has been and continues to be a prolific mining area.

All amounts expressed in this report are in Canadian dollars unless otherwise indicated.

1.2 Geology & Mineralization

The Dumont sill lies within the Abitibi subprovince of the Superior geologic province of the Archean age Canadian Shield. The sill is one of several mafic to ultramafic intrusive bodies that form an irregular, roughly east-west alignment, between Val d'Or, Quebec and Timmins, Ontario. It comprises a lower ultramafic zone which averages 450 m in true thickness and an upper mafic zone about 250 m thick. The ultramafic zone is subdivided into the lower peridotite, dunite and upper peridotite subzones. Cumulus nickel (Ni) sulphide and alloy minerals occur in parts of the dunite subzone and locally in the lower peridotite to form the Dumont deposit.

Figure 1.1: Project Location



Source: RNC.

Disseminated nickel mineralization is characterized by disseminated blebs of pentlandite ($(Ni,Fe)_9S_8$), heazlewoodite (Ni_3S_2), and the ferronickel alloy, awaruite ($Ni_{2.5}Fe$), occurring in various proportions throughout the sill. These minerals can occur together as coarse agglomerates, predominantly associated with magnetite, up to 10,000 µm (10 mm), or as individual disseminated grains ranging from 2 to 1,000 µm (0.002 to 1 mm). Nickel can also occur in the crystal structure of several silicate minerals including olivine and serpentine.

The observed mineralogy of the Dumont deposit is a result of the serpentinization of a dunite protolith, which locally hosted a primary, disseminated (intercumulus) magmatic sulphide assemblage. The serpentinization process whereby olivine reacts with water to produce serpentine, magnetite and brucite creates a strongly reducing environment where the nickel released from the decomposition of olivine is partitioned into low-sulphur sulphides and newly formed awaruite. The final mineral assemblage and texture of the disseminated nickel mineralization in the Dumont deposit and the variability has been controlled primarily by the variable degree of serpentinization that the host dunite has undergone.

Upon acquiring the Dumont property, RNC conducted an initial exploration drilling program in 2007 to confirm the historic drilling results. Results from this drilling campaign confirmed the historical drilling results and encouraged RNC to embark on an extensive drilling campaign to fully evaluate the Dumont deposit. RNC has since conducted core diamond drilling on the Dumont property for the purposes of exploration, resource definition, metallurgical sampling and bedrock geotechnical investigation. Exploration for nickel mineralization on the Dumont property has focussed primarily on diamond drilling due to the lack of outcrop over the ultramafic portions of the Dumont intrusive which host the nickel mineralization. This drilling was initially targeted using data from historical drilling and airborne electromagnetic and magnetic surveys. RNC has also conducted core drilling and cone penetration testing for the purpose of overburden geotechnical characterization. RNC has undertaken an extensive mineralogical sampling program to map mineralogical variability within the Dumont deposit.

1.3 Resources & Reserves

The mineral resource estimate for the Dumont project is presented in Table 1-1; Dumont mineral reserves are summarized in Table 1-2.

The mineral resource estimate was prepared by Mr. Sébastien Bernier, P.Geo, Principal Consultant (Resource Geology) at SRK. The effective date of the current resource estimate is 30 April 2013. The mineral resource estimate considers drilling information available to 31 December 2012 and was evaluated using a geostatistical block modelling approach constrained by seven sulphide mineralization wireframes. The mineral resources have been estimated in conformity with the CIM “Mineral Resource and Mineral Reserves Estimation Best Practices” guidelines and were classified according to the CIM Standard Definition for Mineral Resources and Mineral Reserves (November 2010) guidelines. The mineral resources are reported in accordance with Canadian Securities Administrators’ National Instrument 43-101. SRK is unaware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, and political or other relevant issues that may materially affect the mineral resources.

In addition to nickel, SRK modelled the abundance distribution of seven other main elements: calcium, cobalt, chromium, iron, palladium, platinum, and sulphur as well as specific gravity.

Table 1-1: Mineral Resource Statement*, Dumont Nickel Project, Quebec (SRK Consulting Canada, 30 April, 2013[1])

Resource Category	Quantity (kt)	Grade Ni (%)	Grade Co (ppm)	Contained Nickel (kt)	Contained Nickel (Mlbs)	Contained Cobalt (kt)	Contained Cobalt (Mlbs)
Measured	372,100	0.28	112	1050	2,310	40	92
Indicated	1,293,500	0.26	106	3,380	7,441	140	302
Measured + Indicated	**1,665,600**	**0.27**	**107**	**4,430**	**9,750**	**180**	**394**
Inferred	499,800	0.26	101	1,300	2,862	50	112
Resource Category	**Quantity (kt)**	**Grade Pd (g/t)**	**Grade Pt (g/t)**	**Contained Palladium (koz)**		**Contained Platinum (koz)**	
Measured	372,100	0.024	0.011	288		126	
Indicated	1,293,500	0.017	0.008	720		335	
Measured + Indicated	**1,665,600**	**0.020**	**0.009**	**1,008**		**461**	
Inferred	499,800	0.014	0.006	220		92	
Resource Category	**Quantity (kt)**	**Grade Magnetite (%)**		**Contained Magnetite (kt)**	**Contained Magnetite (Mlbs)**		
Measured	-	-		-	-		
Indicated	1,114,300	4.27		47,580	104,905		
Measured + Indicated	**1,114,300**	**4.27**		**47,580**	**104,905**		
Inferred	832,000	4.02		33,430	73,702		

Notes: 1. *Reported at a cut-off grade of 0.15% nickel inside conceptual pit shells optimized using nickel price of US$9.00 per pound, average metallurgical and process recovery of 40%, processing and G&A costs of US$6.30 per tonne milled, exchange rate of C$1.00 equal US$0.90, overall pit slope of 42° to 50° depending on the sector, and a production rate of 105 kt/d. Values of cobalt, palladium, platinum and magnetite are not considered in the cut-off grade calculation as they are byproducts of recovered nickel. All figures are rounded to reflect the relative accuracy of the estimates. Mineral resources are not mineral reserves and do not have demonstrated economic viability. The Measured and Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce Mineral Reserves.

Table 1-2: Mineral Reserves Statement* (Snowden, 17 June 2013)[1]

Category	(kt)	Grades Ni (%)	Grades Co (ppm)	Grades Pt (g/t)	Grades Pd (g/t)	Contained Metal Ni (Mlb)	Contained Metal Co (Mlb)	Contained Metal Pt (koz)	Contained Metal Pd (koz)
Proven	179,600	0.32	114	0.013	0.029	1,274	45	77	166
Probable	999,000	0.26	106	0.008	0.017	5,667	233	250	550
Total	**1,178,600**	**0.27**	**107**	**0.009**	**0.019**	**6,942**	**278**	**328**	**716**

Notes: 1. *Reported at a cut-off grade of 0.15% nickel inside an engineered pit design based on a Lerchs-Grossmann (LG) optimized pit shell using a nickel price of US$5.58 per pound (62% of the long-term forecast of US$9.00 per pound), average metallurgical recovery of 43%, marginal processing and G&A costs of US$6.30 per tonne milled, long-term exchange rate of C$1.00 equal US$0.90, overall pit slope of 42° to 50° depending on the sector, and a production rate of 105 kt/d. Mineral Reserves include mining losses of 0.28% and dilution of 0.49% that will be incurred at the bedrock overburden interface (which corresponds to mining losses of 1 metre and 2 metres of dilution along this contact). The Proven Reserves are based on Measured Resources included within run-of-mine (ROM) mill feed. Probable Reserves are based on Measured Resources included within stockpile mill feed plus Indicated Resources included in both ROM and stockpile mill feed. All figures are rounded to reflect the relative accuracy of the estimates.

To facilitate RNC's evaluation of nickel recovery, SRK also constructed estimation models of mineral abundances. Specifically, SRK modelled the abundance distribution of awaruite, brucite, coalingite, high iron serpentine, heazlewoodite, serpentine, low-iron serpentine, magnetite, olivine and pentlandite. The mineral model was constructed to support ongoing metallurgical studies.

Reserves were prepared under the direction of David A. Warren, Eng., Principal Consultant - Mining with Snowden Mining Industry Consultants, based on the mineral resource block model described above. Reserves are estimated within an engineered pit design which is based upon a Lerchs-Grossmann (LG) optimized pit shell generated using only nickel values and a nickel price of US$5.58/lb, which is 62% of the long-term forecast of US$9.00/lb and include mining losses of 0.28% and dilution of 0.49%.

1.4 Mining

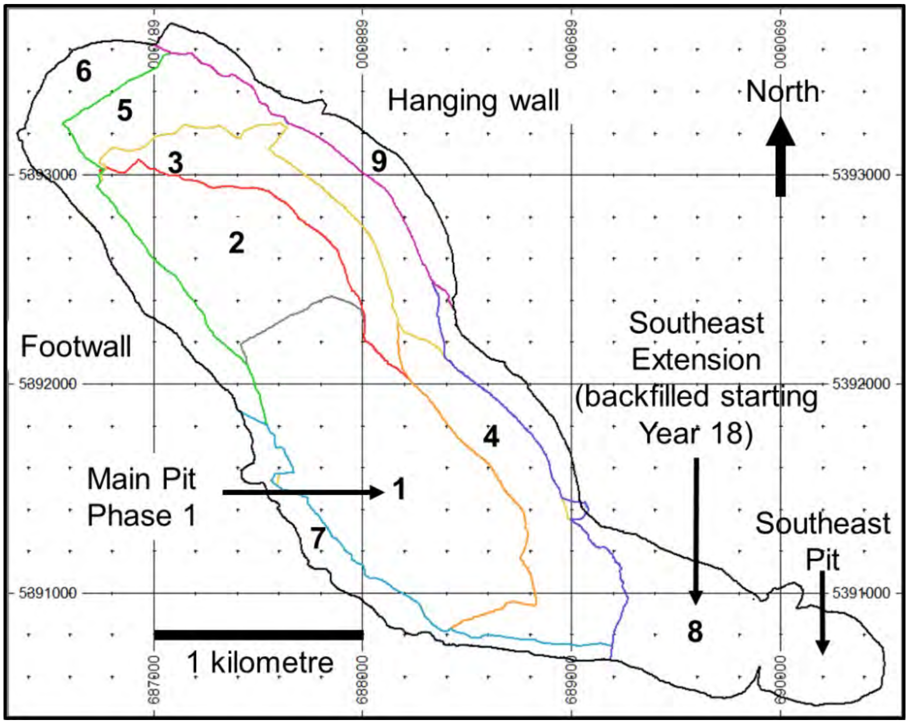
The open pit mine has been designed to provide ore to the plant in a manner that optimises net present value. The initial plant throughput is 52.5 kt/d, with expansion in Year 5 to 105 kt/d. The sequence of mining phases is given in Figure 1.2, with a high level summary of the overall mining sequence being as follows:

- Mining initiates in the southeast pit at the extreme southeast of the deposit. The southeast pit is mined out prior to mill start-up to provide a water reservoir for process operations, a source of waste rock for construction and a stockpile of ore to use in commissioning the mill.
- Upon completion of the southeast pit, the focus of operations will move to the main pit (Phase 1). By Year 2 of mill operations, ex-pit mine production will average 200 kt/d.
- The pit will be pushed back progressively along strike (NW – SE), as well as into the hanging wall to access deeper mineralization. Following expansion of the mill to 105 kt/d in Year 5, the mining rate will progressively increase to 375 kt/d.
- Upon completion of the Main Zone Southeast Extension (SEE) in Year 18, the void will be backfilled with waste rock from the final phases of mining to the northwest.

A key component of the mine plan is the accelerated release of ore from the pit, with higher value ore being fed directly to the mill and lower value material being temporarily stockpiled. During the life of pit, a total of 606 Mt will be loaded to the low-grade stockpiles. Of this, 103 Mt of the highest value stockpile material will be reclaimed during the initial 20 years that the pit is still active. The remaining 503 Mt will be reclaimed after pit closure, extending the life of project for a total of 33 years.

Stockpiling lower-value material maximises the value of material treated during the initial years. Annual output averages 68 Mlbs payable Ni during the first 4.5 years of production (2016 to 2020), when the concentrator throughput is 52.5 kt/d. Annual output increases to 104 Mlbs for 2021 to 2036 when the pit is active. After the pit is depleted and processing of stockpile material only commences, annual output drops to an average of 65 Mlbs.

Figure 1.2: Mining Phase Sequence



Source: RNC.

The strategy of accelerated mining has the additional advantage of creating a void, which would accommodate approximately 43% of the tailings produced, thus reducing the surface footprint of operations.

Approximately 85% of the total 2,514 Mt that will be excavated from the Dumont pit will be mined using large rotary electric blast hole drills (311 mm holes and 15 m benches), rope shovels (43 m^3 dipper) and 230 t payload trucks. A further 10% of the ex-pit total will be mined using rotary diesel blast hole drills (250 mm holes and 10 m benches), large hydraulic excavators (34 m^3 dipper) and 230 t payload trucks. The remaining 5% of material, comprising clay and rock at the sub-outcrop, will be mined with percussion drills (102 mm holes and 5 m benches), small hydraulic excavators (7 m^3 dipper) and 55 t payload trucks. Production equipment will be supported by various units of support equipment, including tracked dozers, wheel dozers, front end loaders, graders, water tankers and utility excavators.

The bulk of the mining fleet will be purchased and operated by the Owner. A local mining contractor with experience operating in similar environments has been pre-selected to assist in the mining operation, particularly during the initial years of operation.

The 2,514 Mt excavated from the pit is comprised of 1,179 Mt ore, 1,159 Mt waste rock, 126 Mt overburden that is mainly sand and gravel, and 50 Mt clay. Approximately 20% of waste rock excavated from the pit will be used to construct the tailings storage facility (TSF) and haul roads. The remainder will be impounded in dumps located on the hanging wall side of the pit. Approximately 56% of waste rock is either gabbro or basalt, and has excellent properties for construction. These rock types will be used to produce roadstone for surfacing roads, in order to reduce dust emissions and improve hauling performance.

In addition to the 50 Mt of clay excavated from the pit, a further 13 Mt will be excavated from the key trench beneath the TSF dam wall for a total of 63 Mt. This clay occurs as either brown clay, which will be used for construction of the TSF or reclamation activities, or grey clay, which has no productive use. Brown clay is estimated to comprise 9 Mt of the total, including 5 Mt from the pit and 4 Mt from the key trench. The 54 Mt of grey clay, including 45 Mt from the pit and 9 Mt from the key trench, will be impounded within cells constructed using sand and gravel or waste rock and located on the hanging wall side of the pit. Sand and gravel will be used for some construction activities, as well as reclamation of waste dumps (as various sub-lithologies within the sand and gravel horizon support organics). The remaining sand and gravel will be impounded in waste dumps located on the hanging wall side of the pit.

Low-grade ore will be located in three distinct dumps depending on NSR value. The highest value stockpile will be located closest to the primary crusher and will be reclaimed first, while the lowest value stockpile will be adjacent to the main waste rock dump.

Infrastructure to support the mining operation will include:

- a roadstone crusher;
- a workshop and associated warehouse (equipment will be maintained under a maintenance contract initially, with a phased handover to in-house personnel as experience is gained);
- a fuel farm and associated fuelling bays; and
- an explosives manufacture facility and magazine. As is the norm in Canada, this will be operated by the explosives supplier.

The Owner’s labour complement averages 331 persons over the life of project, including 463 while the pit is active and 116 during reclaim of the low grade stockpile. These totals do not include personnel allocated to TSF construction, who would average a further 34 during the 20 years that construction of the TSF is active. The Contractor’s complement will average 95 persons during the 8 years that the Contractor is active.

1.5 Metallurgy

The objective of the metallurgical studies was to quantify the metallurgical response of the Dumont ultramafic nickel mineralization. The program was designed to develop the parameters for process design criteria for crushing, grinding, nickel flotation, magnetic recovery and dewatering in the processing plant.

One hundred and two grindability samples were submitted to SGS Mineral Services (Lakefield) to complete a suite of grinding characterization tests including Bond ball work index (BWi), Bond rod work index (RWi), SMC test, and abrasion index (AI). Included in the 102 samples, 10 samples were from the PQ sized core metallurgical variability samples to complete crusher work index (CWi) and JK drop weight tests (JK DWT).

Overall, the ore demonstrated an increase in hardness with finer size, which is typical for many ores. The majority of the test results (percentile 10th to 90th), for the tests performed at coarse size (JK drop-weight test and the SMC test) ranged from moderately soft to medium with an average Axb of 54. In the Bond rod mill grindability test (medium size range) the majority of the samples fell in the medium to moderately hard range with an average RWi of 15 kWh/t. At fine size (Bond ball mill work index and modified Bond tests), the bulk of the test results fall within the hard to very hard range with an average BWi of 21 kWh/t. The Bond low-energy impact test is the exception; the test uses the coarsest rocks, but the sample tested were categorized as moderately hard to hard with an average CWi of 14 kWh/t. Overall the hardness seen in the 102 samples shows a very small range of variability compared with other deposits.

A standard test procedure (STP) to quantify nickel recovery was developed and applied to 105 metallurgical variability samples. The metallurgical variability samples were selected to represent the compositional range of mineralization and to be spatially representative within the pit shell.

The 105 STP tests formed the basis for the rougher nickel recovery equations. The 105 STP samples were divided into four metallurgical domains based on their mineralogy. Metallurgical test results show a clear correlation between mineralogical variations related to degree of serpentinization and metallurgical recovery of nickel. Four metallurgical domains have therefore been established that correspond to these serpentinization domains. They are defined mineralogically on the basis of heazlewoodite to pentlandite ratio (Hz/Pn) and iron-rich serpentine abundance. These are Heazlewoodite Dominant, Mixed Sulphide, Pentlandite Dominant, and High Iron Serpentine.

In all cases the recovery was largely driven by the amount of sulphur in the feed, even for the very low sulphur samples where the main recoverable mineral is awaruite. This may correlate with the amount of nickel present as unrecoverable nickel in silicate minerals, which is variable within known limits throughout the deposit, and is generally higher in the lower sulphide samples.

Seventeen locked cycle tests were completed on different samples to assess the cleaner performance across a variety of feed characteristics. The locked cycle tests showed a wide variation in cleaner recovery. The cleaner recovery was found to be strongly correlated to the sulphur in the ore.

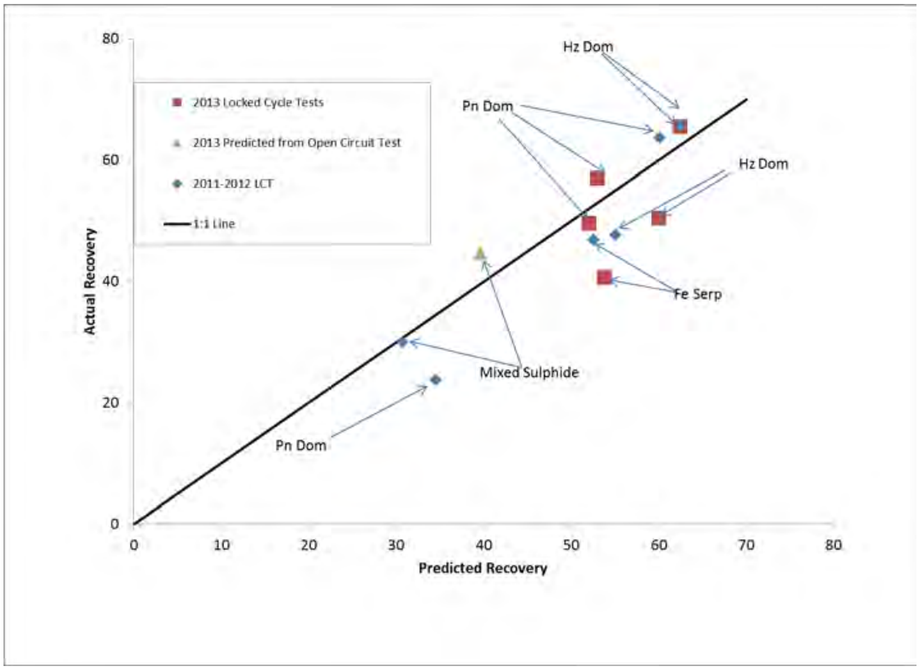
Overall, once the rougher and cleaner recovery equations were applied, the average nickel recovery over the life of the project is 43%.

An additional five locked cycle tests were performed to provide confirmation of the feasibility design and the recovery equations. Although there is some variability around the model, the overall recovery from the locked cycle tests is shown in Figure 1.3 compared to the recovery model used in the feasibility study. Overall the FS recovery model is predicting the Ni recovery demonstrated in the locked cycle tests. The red squares are the 2013 confirmation tests, the blue diamonds are from previous locked cycle tests performed under similar conditions.

Byproduct credits for cobalt (Co), platinum (Pt) and palladium (Pd) were included in the financial analysis. The cobalt recovery is 42% over the life of the project. The calculated Pt + Pd grade in concentrate over the life of the project is 4.3 g/t, based on an average PGE recovery of 61%.

Based on the concentrate assays from the locked cycle test results and the nickel tenor of the recoverable minerals within each metallurgical domain, the concentrate grade has been estimated to be 29% Ni over the life of the project, with a range of 22 to 33%. Other impurities, such as arsenic (As), lead (Pb), chlorine (Cl) and phosphorus (P), were all near or below detection limits in the measured samples. The main impurities in the concentrate are MgO and SiO_2. The measured MgO levels range from 3 to 13% and the average concentrate is expected to be between 7% and 10%, which is in line with the MgO content in concentrates produced by other ultramafic operations.

Figure 1.3: Locked Cycle Test Recovery Performance vs. Model



Source: RNC.

1.6 Mineral Processing

The process plant and associated service facilities will process ore delivered to primary crushers to produce nickel concentrate and tailings. The proposed process encompasses crushing and grinding of the run-of-mine (ROM) ore, desliming via hydrocyclone circuit, slimes rougher flotation, slimes cleaner flotation, nickel sulphide rougher flotation, nickel sulphide cleaning flotation, magnetic recovery of sulphide rougher and cleaner tailings, regrinding of magnetic concentrate and an awaruite recovery circuit (consisting of rougher and cleaner flotation stages).

Concentrate will be thickened, filtered and stockpiled on site prior to being loaded onto railcars or trucks for transport to third-party smelters. The slimes flotation tailings, magnetic separation tailings and awaruite rougher tailings will be combined and thickened before TSF placement.

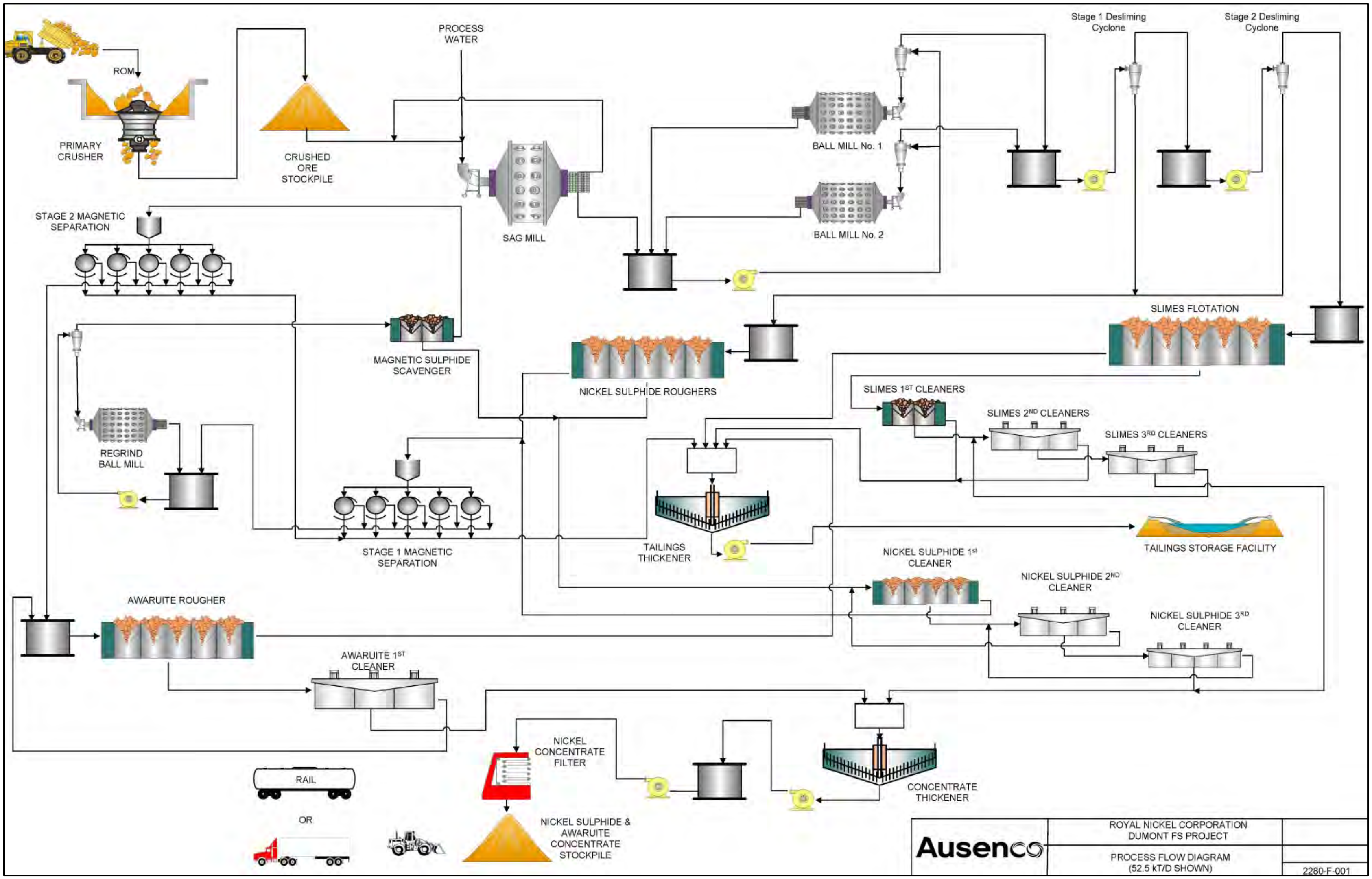
The process plant will be built in two phases. Initially, the plant will be designed to process 52.5 kt/d with allowances for a duplicate process expansion to increase plant capacity to 105 kt/d. Common facilities will include concentrate thickening and handling and sulphuric acid off-loading and containment.

The key criteria selected for the base and expansion plant designs are:

- nominal base plant treatment rate of 52.5 kt/d and a nominal expansion plant treatment rate 52.5 kt/d for a combined 105 kt/d treatment rate;
- design availability of 92% (after ramp-up), which equates to 8,059 operating hours per year, with standby equipment in critical areas; and
- sufficient plant design flexibility for treatment of all ore types at design throughput.

A schematic of the process plant is provided in Figure 1.4.

Figure 1.4: Dumont Process Plant Schematic



1.7 Infrastructure

The project site is well serviced with respect to other infrastructure, including:

- Road – Provincial Highway 111 runs along the southern boundary of the property.
- Rail – The Canadian National Railway (CNR) runs through the property, slightly to the north of Highway 111 but south of the engineered pit.
- Power – The provincial utility, Hydro-Quebec, has indicated that it would be feasible to provide electrical power to the mine site via a 10.5 km long 120 kV overhead powerline to be constructed, which would be connected as a tee-off to an existing line. The line will enter the property from the south near the security entrance gate, and runs up to the process plant main 120 kV substation.
- Water – Water for start-up will be provided by surface water storage at the Southeast Reservoir and, possibly, local groundwater wells. During operations, water demand will largely be met by recycling water from the TSF. Make-up water and freshwater requirements will be provided by the Southeast Reservoir. A water treatment plant will be constructed in approximately the year 2022 to treat excess water from the TSF prior to its discharge to the Villemontel River.
- Natural Gas – Although the use of natural gas is not considered in this study, an existing pipeline extends to within approximately 25 km to the south of the property.

Both the initial and expansion phases of the Dumont project will require three 120:13.8 kV 60/80 MVA main transformers. The new 120 kV substation and six main transformers will be installed near the SAG Mill Feed Conveyor. The 13.8 kV medium voltage network will be used for the primary electrical distribution and for feeding large loads such as the SAG mill and ball mills.

A rail spur that services the process plant is proposed for the project. The total length of the rail spur is approximately 5 km. The rail spur initially consists of a fuel delivery track near the mining truckshop and a freight delivery track north of the process plant. The process plant area consists of the crushing facility, covered stockpile and process plant building. The overall process plant enclosed structure is approximately 350 m long, and consists of four connected buildings: grinding, flotation, cleaning, and filtration.

The TSF will be situated approximately 400 m west of the process plant and consists of two cells. Cell 1 will be constructed initially, followed by Cell 2 during Year 6 of operations.

The TSF is designed to store approximately 680 Mt of tailings produced over a period of approximately 20 years. Once mining has ceased at the open pit, stockpiled ore will be processed for approximately 13 years and those tailings, approximately 498 Mt, will report to the open pit.

1.8 Environmental

The assessment of environmental risks and potential impacts conducted to date originates principally from the Environmental and Social Impact Assessment (ESIA) performed as part of the Dumont project permitting process and integrates a number of studies performed by RNC and its consultants over the past five years. Biophysical data come mainly from three distinct fieldwork programs performed from 2007 to 2009, with some complementary information extracted from the ongoing baseline studies designed to support the ESIA in 2011 and 2012.

Table 1-3 summarizes the sources of information for the various biophysical and social components described in this report.

Table 1-3: Sources of Biophysical & Social Components included in the Feasibility Study

Type of Study	2007[1]	2008[2]	2009[3]	2011[4]	2012
Water and sediment quality	√	√	√	√	
Groundwater quality					√[6]
Vegetation and wetlands		√		√	
Wildlife	√	√	√		
Small mammals				√	
Fish	√	√	√	√	√[6]
Benthic invertebrates	√	√	√		
Birds		√		√	
Reptiles and amphibians				√	
Archaeology		√			
Stakeholders consultation				√[5]	√[7]

Notes: 1. Ménard et Coppola (2008). 2. GENIVAR (2009). 3. GENIVAR (2010). 4. Unpublished data. 5. Transfert Environnement (2011). 6. ESIA (2012) 7. Transfert Environnement (2013) **Source:** RNC.

These environmental baseline studies have not identified any specific inordinate environmental risk to project development. Environmental sensitivities are primarily related to potential impacts associated with the scale and footprint of the proposed operation, and the composition of materials being handled and impounded on the site. Principal impacts anticipated at this stage relate to air quality, wetlands, fish habitat, water resources (surface and groundwater), and the social environment.

To limit environmental impact to one drainage basin, RNC has elected to limit project infrastructure to within the St. Lawrence drainage basin. RNC has also observed a one-kilometre buffer zone between surrounding esker aquifers and project infrastructure.

Although three “at risk” plant species were found within the study area defined for the Dumont ESIA, the current project development plans would not affect the locations where these species were observed. The environmental characterization underlined the presence of rock vole, a small mammal species likely to be listed on Quebec’s threatened or vulnerable species list. Mitigation measures aiming at promoting rock vole habitat were introduced in the ESIA. The presence of three “at risk” bird species was noted during the ESIA: olive-sided flycatcher, rusty blackbird, and common nighthawk. A mitigation measure intended to protect nests during the nesting period was implemented in the ESIA to reduce direct impact on these species.

Results of the ESIA demonstrates that most of the impacts anticipated from the Dumont project are qualified as low or very low once general and specific mitigation measures are applied. Only one impact is qualified as very important or important, namely the risk of nitrogen dioxide formation due to blasting at concentrations likely to affect health as this phenomenon has not yet been modelled and precise impacts could not be evaluated. Atmospheric dispersion modelling studies of airborne nitrogen dioxide concentrations during blasting will allow a more precise assessment of the health risks and whether specific preventive measures are required within the framework of the emergency response plan. These types of emissions are not unique to the Dumont project but are common to all open pit operations.

Environmental geochemistry characterization of tailings, waste and ore indicate that these materials will be non-acid-generating due to their low sulphur content and high neutralization potential. Static tests indicate that waste rock and ore are leachable under the conditions of the tests, but kinetic tests that are more representative of anticipated site conditions showed that leachability is very low, meets Quebec effluent criteria and meets Quebec groundwater quality criteria in the long-term. The waste rock and tailings also demonstrate significant potential for permanent carbon sequestration through spontaneous mineral carbonation.

1.9 Community

Mindful of the interest shown by host communities following the announcement of the Dumont project, RNC voluntarily initiated a public information and consultation process during the exploration phase. The process aims to ensure effective communication and dissemination of information about the project, and to document the concerns, comments and suggestions of the host communities to refine the technical and economical studies where possible, and has helped define the content of the environmental impact study.

To ensure a rigorous approach and to facilitate dialogue with the company, RNC retained the services of a social harmonization firm, Transfert Environnement. Acting as a third party during the consultation activities, its role was to support RNC in the coordination of the consultation activities and to produce the minutes and reports documenting the discussions and how RNC integrated them into the design of the Dumont project.

All information and consultation activities were documented and concerns expressed by the stakeholders were compiled. Results of consultations were submitted to the relevant authorities, and filed as a public document on RNC's website.

The following types of communication were used during the consultation process:

- information sessions;
- open house events and site visits;
- feedback activities;
- establishment of advisory committees:
  - expanded advisory committee;
  - municipalities/company round-table; and
- information and consultation processes for the Abitibiwinni First Nation in Pikogan.

1.10 Capital Cost Estimate

All amounts expressed are in Canadian dollars unless otherwise indicated.

Table 1-4 provides a summary of the capital costs estimate, including initial capital, expansion capital, and sustaining capital. Table 1-5 shows the total capital costs by area, excluding sustaining capital. The costs are expressed in real, Q2 2013 Canadian dollars and include all mining, site preparation, process plant, dams, sumps, first fills, buildings, and roadworks. Items that would be denominated in foreign currency take account of the forecast exchange rate at the time of purchase.

The estimates are considered to have an overall accuracy ±15% and assume the project will be developed on an EPCM basis.

Major cost categories (permanent equipment, material purchase, installation, subcontracts, indirect costs and Owner's costs) were identified and analyzed. To each of these categories, a percentage of contingency was allocated based on the accuracy of the data, and an overall contingency amount was derived in this fashion.

Table 1-4: Summary of Capital Costs ($ M)

Description	Initial Capital ($ M)	Expansion Capital ($ M)	Sustaining Capital ($ M)	LOM Total Capital ($ M)
Mine	320	216	419	955
Process Plant	550	523	254	1,327
Tailings	34	61	172	267
Infrastructure	87	27	-	114
Indirect Costs[1]	172	89	(22)	239
Contingency	105	81	0	186
Total	**1,268**	**997**	**823**	**3,088**

Notes: 1. Negative value represents release of first fills at end of project life.

Table 1-5: Capital Costs by Area ($ M) – Not Including Sustaining Capital

Area	Direct Costs	Initial Capital	Expansion Capital	Total Cost
01	Mining	320	216	536
02	Crushing	55	55	110
03	Process	372	369	741
04	Concentrate Loadout	0.3	0.0	0.3
05	Tailings	34	61	95
06	Utilities	123	99	222
07	Onsite Infrastructure	80	22	102
08	Off-site Infrastructure	7	5	12
Total Direct Costs		**991**	**827**	**1,818**
09	Indirect Costs	125	80	205
10	Owner's Costs	47	9	56
Total Indirect Costs		**172**	**89**	**261**
Total Direct & Indirect Costs		**1,163**	**916**	**2,079**
11	Escalation	Not Included		
11	Contingency	105	81	186
Total Project Costs (as of Q2 2013)		**1,268**	**997**	**2,265**

1.11 Operating Cost Estimate

All amounts expressed are in Canadian dollars unless otherwise indicated.

A summary of life-of-mine (LOM) operating costs is provided in Table 1-6.

Table 1-6: LOM Operating Cost Summary

	Units	52.5 kt/d 2016-2020	105 kt/d 2021-2036	Stockpile 2036-2049	LOM Average
Mine	$/t ore milled	$6.61	$6.15	$0.77	$3.89
Mine[1]	$/t ex-pit material mined	$1.63	$1.69	$0.00	$1.68
Process	$/t ore	$5.04	$4.76	$4.76	$4.78
G&A	$/t ore	$0.94	$0.56	$0.41	$0.52
Site Costs	$/t ore	$12.60	$11.46	$5.94	$9.18
Site Costs	$/lb	$3.45	$4.15	$3.59	$3.90
TC/RC	$/lb	$1.45	$1.40	$1.43	$1.42
Gross C1 Cash Cost	$/lb	$4.90	$5.55	$5.02	$5.32
Byproduct Credits	$/lb	($0.46)	($0.51)	($0.61)	($0.53)
Net C1 Cash Cost	$/lb	$4.44	$5.04	$4.41	$4.79
	US$/lb	US$4.01	US$4.54	US$3.97	US$4.31

Notes: 1. To give a true reflection of expit mining costs, excludes $61 M for rehandle of 103 Mt stockpile ore during ex-pit mine life.

1.12 Economic Analysis

The Dumont Nickel project is expected to produce 2.8 billion pounds payable Ni over 33 years of operation. Table 1-7 summarizes key metrics for the current FS design. The costs and returns for the FS assume a long-term nickel price of US$9.00/lb Ni and a Canadian dollar exchange rate of US$0.90. A full list of price assumptions and further details can be found in Section 22.

Table 1-7: Summary Economic Metrics

	Unit	C$	US$
Ore Mined	Mt	1,179	1,179
Payable Ni	Mlbs	2,774	2,774
Payable NiEq[1]	Mlbs	2,922	2,922
Gross Revenue	$/t ore	24.88	22.40
TC/RC	$/t ore	3.33	3.00
Net Smelter Return	$/t ore	21.54	19.40
Site Operating Costs	$/t ore	9.18	8.27
Gross C1 Costs	$/lb Ni	5.32	4.79
Net C1 Costs	$/lb Ni	4.79	4.31
Initial Capital	$M	1,268	1,205
Expansion Capital	$M	997	898
Sustaining Capital	$M	823	741
Total Capital	$M	3,088	2,844
Pre-Tax $NPV_{8\%}$	$M	2,293	2,003
Pre-Tax IRR		19.5%	18.7%
Post-Tax $NPV_{8\%}$	**$M**	**1,330**	**1,137**
Post-Tax IRR		**15.9%**	**15.2%**

Notes: 1. Based on the production profile given in Table 16-1 and the price profiles given in Table 22-2.

The NPV reported in Table 1-7 is in real, Q2 2013, terms with the start date for discounting being the commencement of project construction in September 2014. No material expenditures are expected prior to this date.

The post-tax NPV includes planned changes to the fiscal tax regimes that include:

- the phasing out of categories that allow for accelerated depreciation of pre-production expenditures;
- the phasing out of the investment tax credit; and
- a fundamental change to the Quebec mining tax regime, with the new regime including both a minimum tax that would be payable from the outset of commercial operations and a variable tax that would increase as a function of profitability.

The NPV is most sensitive to factors impacting on revenue, with the impact of a ±10% variation in Ni price or Ni recovery having a 37% impact on NPV. The project is also sensitive to exchange rate, with a 10% change in exchange rate impacting NPV by over 30%. The project is less sensitive to costs, with a 10% change in total site operating costs having a 17% impact on NPV, while a 10% change in total capital has an 11% impact.

The peak funding requirement is $1,320 M, which is reached during the first quarter of mill operations. While the project economics are based on equity only, based on a potential debt financing of approximately 60% of the total invested capital, payback of debt finance would be achieved in the third year of operation, allowing for re-investment in the expansion of concentrator capacity to 105 kt/d. The expansion would be commissioned during Year 5 of operation. Following the expansion, annual free cash flow increases to over $310 M for the period that the pit is operational and payback of all invested capital (including the expansion) is achieved approximately 6.1 years after the initial start-up. After the pit is depleted and low-grade stockpiles are treated, free cash flow averages $210 M annually, due to the low cost structure of the operation.

1.13 Project Implementation

Overall schedule duration from commencement of basic engineering (to order long-lead equipment) to the end of ore commissioning is 36 months. Key milestone dates are described in Table 1-8.

Table 1-8: Dumont Nickel Project Schedule – Key Milestone Dates

Criteria	Date
Commence Detailed Engineering for Long Lead Equipment	Q3 2013
Order Long Lead Equipment	Q4 2013
Commence Full EPCM	Q1 2014
C of A Approval	Q3 2014
Construction Permit Approval	Q3 2014
Substantial Completion of Engineering	Q1 2015
Hydro Contract Power	Q3 2015
Start of Commissioning	Q3 2015
Mechanical Completion	Q1 2016
Reception of First Ore	Q2 2016
Plant Operational	Q3 2016

1.14 Conclusions & Recommendations

The investigation and analysis carried out are considered appropriate to feasibility level mine design. Further investigations are recommended as the project advances to detailed design.

Recommendations for future work are listed below:

- Continue environmental permitting process;
- Continue environmental baseline studies;
- Complete detailed design that considers the following points:
  - Evaluate opportunities for pit optimization, including:
    - Alternative mining sequences that may allow access to higher value ore to be accelerated and/or deferral of waste stripping; and
    - Evaluate alternative ramp locations in the pit stages taking advantage of changes in wall slopes.
  - Continue to monitor the opportunity of implementing trolley-assisted truck haulage. Factors that should be considered before a decision is ultimately made include the relative prices of diesel and electricity, the regional availability of electricity, changing legislation regarding emissions from fossil fuels and the development of trolley-assist technology;
  - Begin detailed engineering in Q3 2013 and procure long lead equipment in order to maintain the Q3 2016 plant operational date;
  - Undertake detailed geotechnical evaluations of the early rock exposures, throughout the open pit areas, to assess the reliability of structural and geotechnical models. Optimize design based on field performance of pit slopes in the various geotechnical domains;
  - Continue to collect and evaluate pore pressure data from existing piezometer installations within the pit slope areas to verify the assumption that these will have a limited impact on slope stability;
  - Conduct further geotechnical investigations to define the extent, thickness and, in some cases, the location-specific strength of the weak, soft soils beneath all surface infrastructure, including the plant site area and related facilities, rail lines, TSF Cell 1, the low-grade ore stockpile within the pit limits, and water management features that have a significant earthworks component to them and are required within the first few years of operation;
  - Implement a metallurgy testwork program that will include:
    - Trade-off study to evaluate removal of slimes circuit;
    - Reagent optimization testwork;
    - Concentrate thickening and filtration testwork;
    - Slimes cyclone pilot scale testing for detailed engineering design;
    - Awaruite recovery circuit optimization;
    - Recovery opportunities from scavenger non-magnetic stream; and
    - Complete testwork to quantify grindability characteristics of regrind mill feed.
- Specific high voltage power studies as recommended for confirmation of high voltage supply by Hydro Quebec;
- Continue mining lease process;

- Initiate surface lease process;
- Continue to investigate the natural cementation of tailings and waste fines and its impact on reducing the potential for these project components to act as dust sources;
- Continue stakeholder consultation during detailed engineering as well as during mine operations to minimize and/or mitigate the impact of the project and foster acceptance. Define the structure of stakeholder committees that will be created during mine construction and operations; and
- Continue to assess the carbon sequestration potential of spontaneous mineral carbonation of tailings and waste rock on an operational basis and its impact on the carbon footprint of the project.

2 INTRODUCTION

2.1 Background

RNC is a Toronto, Canada headquartered mineral resource company focused primarily on the exploration, evaluation, development, and acquisition of base metal and platinum group metal properties. RNC's principal asset is the Dumont Nickel project (Dumont project) located in the established Abitibi mining camp, 25 km northwest of Amos, Quebec. RNC acquired a 100% interest in the Dumont property in 2007. The mineral claims covering the Dumont deposit are currently held 98% by RNC and 2% by Ressources Québec.

This technical report, prepared for RNC and dated July 25, 2013, as well as the resource estimate, has been prepared in compliance with the disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101 (NI 43-101), Companion Policy 43-101CP, and Form 43-101F1.

2.2 Project Scope & Terms of Reference

This technical report was prepared for RNC by Ausenco to provide RNC with sufficient information to determine the economic feasibility of developing the Dumont deposit.

Ausenco was commissioned by RNC in May 2012 to prepare the feasibility study and NI 43-101 compliant technical report on the project. SRK was engaged to prepare the resource estimate, hydrogeology, hydrology and geotechnical inputs and to supervise geology inputs. David Penswick, a private mining consultant, was engaged for mine design, mine operating costs, mine capital costing and economic modelling. Snowden reviewed and qualified the mine design, and Ausenco reviewed and qualified the economic modelling. GENIVAR has been engaged since 2007 to conduct environmental studies on behalf of RNC for the Dumont project and prepare the Environmental and Social Impact Assessment (ESIA). Golder Associates Ltd. (Golder) prepared the environmental geochemistry investigations. In September 2012 Norascon was selected for the overburden pre-stripping phase allowing further early operational de-risking and optimization to be included in the feasibility study through integration of Norascon's local experience in overburden stripping and tailings storage facility construction into project design.

The feasibility study has, at its focus, the Dumont low-grade ultramafic nickel deposit. However, RNC has explored extensively throughout the Dumont property and this report presents some new information in relation to exploration, data, and detailed geology outside of this deposit in Section 10.6.

The Dumont project consists of an open pit mine and an associated processing facility along with on-site and off-site infrastructure to support the operation. The mine, process plant and associated infrastructure are designed to initially process 52.5 kt/d of ore, with expansion to 105 kt/d in Year 5.

2.3 Qualified Persons

The responsibilities of each author are provided in Table 2-1.

Table 2-1: Participants in the Dumont Prefeasibility Study

Activity	Lead	Report Section	Responsible Qualified Person	Organization
Study Coordination	P. Staples	1, 2, 3, 25, 26, 27	P. Staples	Ausenco
Property Description & History	A. St-Jean	4, 5, 6	P. Staples	Ausenco
Geology, Exploration & Database	A. St-Jean	7, 8, 9, 10, 11	S. Bernier	SRK
Data Verification	S. Bernier	12	S. Bernier	SRK
Mineral Processing & Metallurgical Testing	J. Muinonen	13	J. Bowen	Ausenco
Mineral Resource Estimate	S. Bernier	14	S. Bernier	SRK
Mineral Reserve Estimates	D. Penswick	15	D. Warren	Snowden
Hydrology & Hydrogeology	J. Duncan	16.1	J. Duncan	SRK
Geotechnical Design Criteria – Rock	B. Murphy	16.2.1, 16.2.2, 16.2.3	B. Murphy	SRK
Geotechnical Design Criteria – Soil	C. Scott	16.2.4	C. Scott	SRK
Open Pit Mine Plan	D. Penswick	16.3	D. Warren	Snowden
Mining Process	D. Penswick	16.4	D. Warren	Snowden
Recovery Methods	J. Bowen	17	P. Staples	Ausenco
Project Infrastructure	D. Markovic	18 (except 18.6, 18.7, 18.16)	P. Staples	Ausenco
Overburden, Waste & Low-grade Stockpiles	C. Scott	18.6	C. Scott	SRK
Tailings Storage Facility	C. Scott	18.7	C. Scott	SRK
Surface Water Management	J. Duncan	18.16	J. Duncan	SRK
Market Studies & Contracts	M. Selby	19	P. Staples	Ausenco
Environmental Studies, Permitting & Social/Community Impact	A. St-Jean	20 (except 20.7.1, 20.7.2, 20.7.5)	S. Latulippe	GENIVAR
Environmental Geochemistry	V. Bertrand	20.7.1, 20.7.2, 20.7.5	V. Bertrand	Golder
Capital & Operating Costs	S. Booth	21 (except 21.3.1, 21.3.3, 21.5.3)	P. Staples	Ausenco
Capital & Operating Costs - Mining	D. Penswick	21.3.1, 21.5.3	D. Warren	Snowden
Capital & Operating Costs - TSF	C. Scott	21.3.3	C. Scott	SRK
Economic Analysis	D. Penswick	22	K. Scott	Ausenco
Adjacent Properties	A. St-Jean	23	P. Staples	Ausenco
Site Wide Geotechnical Overview	C. Scott	24.1.1	C. Scott	SRK
Plant Site	C. Scott	24.1.2	C. Scott	SRK
Project Implementation	D. Markovic	24.1.3	P. Staples	Ausenco
Closure	C. Scott	24.1.4	C. Scott	SRK
Cost Benchmarks	D. Penswick	24.1.5	P. Staples	Ausenco
Opportunities – Trolley Assist	D. Penswick	24.2.1	D. Warren	Snowden
Magnetite	J. Muinonen	24.2.2	J. Bowen	Ausenco
Ferro Nickel	J. Muinonen	24.2.3	J. Bowen	Ausenco

The Qualified Persons listed below have contributed to the Technical Report as specified.

- Paul Staples of Ausenco for mineral processing, plant and infrastructure capital and operating costs and study coordination. Paul visited the property on May 19, 2011 and August 8, 2012.
- Jeff Bowen of Ausenco for metallurgy. Jeff visited the property on August 8, 2012.
- Sebastien Bernier of SRK completed the mineral resource estimation and data verification, and supervised the geology and exploration contribution. Sebastien visited the property during April 27 to May 2, 2011, and on May 17, 2013.
- David Warren for reserve estimation, mining, mine capital and operating costs. David visited the property on August 8, 2012.
- Cam Scott of SRK for mine soil geotechnical, tailings storage facility design, waste rock and overburden dump design, and low-grade ore stockpile design. Cam visited the property on February 2, May 19 and June 21 in 2011 and on July 13 and August 8, 2012.
- John Duncan of SRK for hydrology and hydrogeology. John visited the property on April 10 and 11, 2012 and August 8, 2012.
- Bruce Murphy of SRK for mine rock geotechnical and pit slopes, Bruce visited the property during June 17 and 18, 2011.
- Valerie Bertrand from Golder for environmental geochemistry. Valerie visited the property on August 8, 2012.
- Kevin Scott of Ausenco for Economic Analysis. Kevin did not visit the site.
- Simon Latulippe of Genivar for Environmental Studies, Permitting and Social/Community Impact. Simon visited the property on July 13, 2013.

2.4 Frequently Used Acronyms, Abbreviations, Definitions, Units of Measure

All currency amounts are stated in Canadian dollars (C$, CAD), unless otherwise specified, with commodity prices typically expressed in US dollars (US$, USD). Quantities are generally stated using the Système International d'Unités (SI) or metric units, the standard Canadian and international practice, including metric tonnes (t), kilograms (kg) or grams (g) for weight, kilometres (km) or metres (m) for distance and hectares (ha) for area. Wherever applicable, imperial units have been converted to SI units for reporting consistency.

Frequently used acronyms and abbreviations are listed below.

Above mean sea level	amsl
Annum (year)	a
Centimetre	cm
Cubic centimetre	cm^3
Cubic metre	m^3
Cubic metres per day	m^3/d
Concentration by weight	Cw
Day	d
Days per year (annum)	d/a
Degree	°
Degrees Celsius	°C
Dry metric ton	dmt

Engineering, procurement and construction	EPC
Engineering, procurement and construction management	EPCM
Foot	ft
Gram	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Horsepower	hp
Hour	h
Hours per day	h/d
Hydro Quebec	HQ
Inch	"
Inverse distance	ID
Kilogram	kg
Kilometre	km
Kilovolts	kV
Kilowatt hour	kWh
Kilowatt	kW
Less than	<
Litre	L
Life of mine	LOM
Litres per second	L/sec
Measure of the acidity or basicity of a solution	pH
Metre	m
Metres above sea level	masl
Metres per annum	m/a
Metres per hour	m/h
Metres per minute	m/min
Metres per second	m/sec
Micrometre (micron)	µm
Millimetre	mm
Million pounds	Mlbs
Million pounds per annum	Mlbs/a
Million tonnes	Mt
Million tonnes per annum	Mt/a
Million	M
Million years	Ma
Minute (plane angle)	'
Minute	min
Net present value	NPV
Net smelter return per tonne	NSR/tonne
Ounce	oz
Parts per billion	ppb
Parts per million	ppm
Percent	%

Pound(s)	lb(s)
Run of mine	ROM
Second (plane angle)	"
Second (time)	sec
Square kilometre	km^2
Square metre	m^2
Tonne (1,000 kg)	t
Thousand tonne	kt
Thousand tonne per day	kt/d
Tonne Force	tonf
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Troy ounces	troy oz
Year (annum)	a

3 RELIANCE ON OTHER EXPERTS

In preparing this report, Ausenco has relied on input from RNC and a number of well-qualified, independent consulting groups.

Ausenco is not an expert in legal, land tenure, or environmental matters. Ausenco has relied on data and information provided by RNC and on previously completed technical reports (refer to Section 27 for details). Although Ausenco has reviewed the available data and visited the site, these activities serve to validate only a portion of the entire data set. Therefore, Ausenco has made judgments about the general reliability of the underlying data; where deemed either inadequate or unreliable, the data were either not used or procedures were modified to account for the lack of confidence in that specific information.

While exercising all reasonable diligence in checking, confirming and testing it, Ausenco has relied upon RNC's presentation of its project data and that of previous operators of the Dumont property, in formulating its opinion.

The various agreements under which RNC holds title to the mineral claims for this project have not been reviewed by Ausenco, and Ausenco offers no legal opinion as to the validity of the mineral title claimed. A description of the property, and ownership thereof, is provided for general information purposes only. Comments on the state of environmental conditions, liability, and estimated costs of closure and remediation have been made where required by NI 43-101. In this regard Ausenco has relied on the work of GENIVAR and other experts it understands to be appropriately qualified, and Ausenco offers no opinion on the state of the environment on the property. The statements are provided for information purposes only.

The descriptions of geology, mineralization and exploration used in this report are taken from reports prepared by various companies or their contracted consultants. The conclusions of this report rely on data available in published and unpublished reports supplied by the various companies which have conducted exploration on the property, and information supplied by RNC. The information provided to RNC was supplied by reputable companies or government agencies and Ausenco has no reason to doubt its validity.

Ausenco has relied upon RNC's legal counsel for legal input for Sections 4.3 and 4.4.

4 PROPERTY DESCRIPTION & LOCATION

4.1 Location

The Dumont property is located in the province of Quebec, approximately 25 km by road, northwest of the city of Amos. Amos has a population of 12,584 (2006 Census) and is the seat of the Abitibi County Regional Municipality (Figure 4.1 overleaf).

RNC advises that the Dumont property consists of 233 contiguous mineral claims totalling 9,306 hectares (ha). The longitude and latitude for the Dumont property are 48°38’53” N, 78°26’30”W (UTM coordinates are 5,391,500N, 688,400E within UTM zone 17 using the NAD83 Datum). As shown in Figure 4.1, the property is located approximately 25 km west of the city of Amos, 60 km northeast of the industrial and mining city of Rouyn-Noranda, 70 km northwest of the city of Val d’Or. The mineral resource is located mainly in Ranges V, VI and VII on Lots 46 to 62 of Launay Township, and in Range V on Lots 1 to 3 of Trécesson Township.

4.2 Mineral Tenure

4.2.1 Mineral Claims

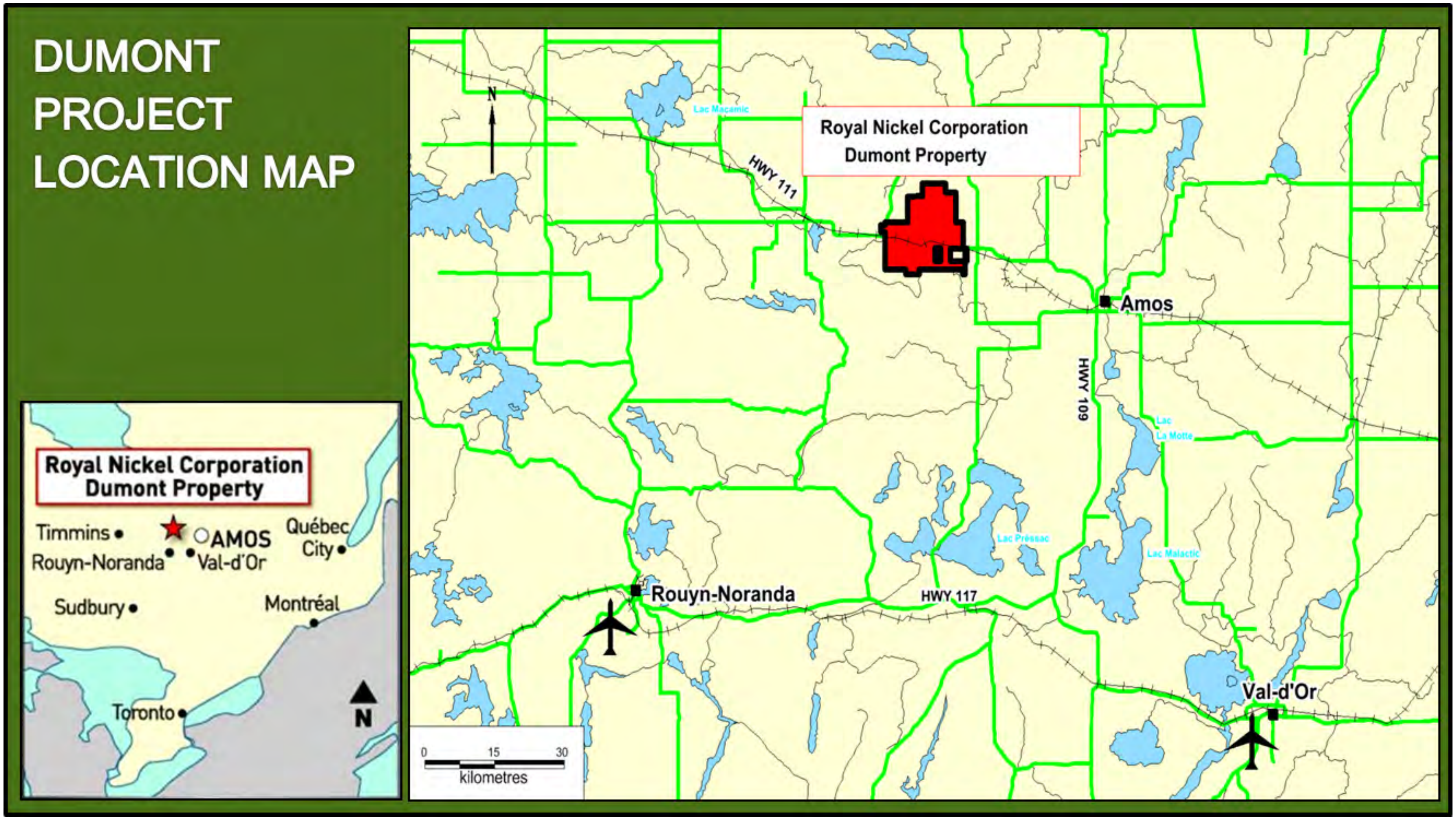
RNC advises the mineral properties comprising the Dumont property are all mineral claims. RNC holds 100% beneficial interest in five claims; the beneficial interest in the remaining 228 claims is held 98% by RNC and 2% by Ressources Québec to secure the Ressources Québec royalty. Identifying numbers, as well as ownership details for each claim, are given in Table 4-1 and claim locations with respect to the Dumont deposit are shown in Figure 4.2.

4.2.2 Mineral Claims Conversion

On February 18, 2013, as part of the ongoing program of claim standardization being carried out by the Quebec Ministry of Natural Resources, the ground-staked (CL) claims that were part of the Dumont property were converted to map-staked (CDC) claims that conform to the 30-second by 30-second map-staking fabric.

The area corresponding to these CL has been converted to new claims as shown in Figure 4.2 above. Consequently, the royalty boundaries shown in Figure 4.2 no longer necessarily correspond to current claim boundaries.

Figure 4.1: Project Location



Source: RNC.

Table 4-1: Dumont Property Mineral Claims

Claim Number	Township	Type	Date Renewal Due	Area (ha)	Renewal Cost ($)	Interest
2025446	LAUNAY	CDC	19-Sep-14	43.16	1854.25	98% RNC, 2% Ressources Québec
2025447	LAUNAY	CDC	19-Sep-14	43.12	1854.25	98% RNC, 2% Ressources Québec
2025448	LAUNAY	CDC	19-Sep-14	43.08	1854.25	98% RNC, 2% Ressources Québec
2025449	LAUNAY	CDC	19-Sep-14	43.05	1854.25	98% RNC, 2% Ressources Québec
2025450	LAUNAY	CDC	19-Sep-14	43	1854.25	98% RNC, 2% Ressources Québec
2025451	LAUNAY	CDC	19-Sep-14	42.97	1854.25	98% RNC, 2% Ressources Québec
2025452	LAUNAY	CDC	19-Sep-14	42.91	1854.25	98% RNC, 2% Ressources Québec
2031504	LAUNAY	CDC	6-Nov-14	47.94	1854.25	98% RNC, 2% Ressources Québec
2031505	LAUNAY	CDC	6-Nov-14	39.78	1854.25	98% RNC, 2% Ressources Québec
2031506	LAUNAY	CDC	6-Nov-14	39.74	1854.25	98% RNC, 2% Ressources Québec
2054112	LAUNAY	CDC	8-Feb-15	42.63	1854.25	98% RNC, 2% Ressources Québec
2054113	LAUNAY	CDC	8-Feb-15	42.64	1854.25	98% RNC, 2% Ressources Québec
2054114	LAUNAY	CDC	8-Feb-15	42.63	1854.25	98% RNC, 2% Ressources Québec
2054115	LAUNAY	CDC	8-Feb-15	42.64	1854.25	98% RNC, 2% Ressources Québec
2054116	LAUNAY	CDC	8-Feb-15	42.63	1854.25	98% RNC, 2% Ressources Québec
2054117	LAUNAY	CDC	8-Feb-15	42.64	1854.25	98% RNC, 2% Ressources Québec
2054118	LAUNAY	CDC	8-Feb-15	42.65	1854.25	98% RNC, 2% Ressources Québec
2054119	LAUNAY	CDC	8-Feb-15	42.65	1854.25	98% RNC, 2% Ressources Québec
2054120	LAUNAY	CDC	8-Feb-15	42.66	1854.25	98% RNC, 2% Ressources Québec
2054121	LAUNAY	CDC	8-Feb-15	42.66	1854.25	98% RNC, 2% Ressources Québec
2054122	LAUNAY	CDC	8-Feb-15	42.67	1854.25	98% RNC, 2% Ressources Québec
2054124	LAUNAY	CDC	8-Feb-15	41.8	1854.25	98% RNC, 2% Ressources Québec
2054125	LAUNAY	CDC	8-Feb-15	41.74	1854.25	98% RNC, 2% Ressources Québec
2054126	LAUNAY	CDC	8-Feb-15	41.69	1854.25	98% RNC, 2% Ressources Québec
2054127	LAUNAY	CDC	8-Feb-15	41.65	1854.25	98% RNC, 2% Ressources Québec
2054128	LAUNAY	CDC	8-Feb-15	41.59	1854.25	98% RNC, 2% Ressources Québec
2054129	LAUNAY	CDC	8-Feb-15	41.54	1854.25	98% RNC, 2% Ressources Québec
2054130	LAUNAY	CDC	8-Feb-15	42.39	1854.25	98% RNC, 2% Ressources Québec
2054131	LAUNAY	CDC	8-Feb-15	42.8	1854.25	98% RNC, 2% Ressources Québec
2054132	LAUNAY	CDC	8-Feb-15	39.7	1854.25	98% RNC, 2% Ressources Québec
2054133	LAUNAY	CDC	8-Feb-15	39.65	1854.25	98% RNC, 2% Ressources Québec
2054894	LAUNAY	CDC	13-Feb-15	42.41	1854.25	98% RNC, 2% Ressources Québec
2054895	LAUNAY	CDC	13-Feb-15	42.4	1854.25	98% RNC, 2% Ressources Québec
2054896	LAUNAY	CDC	13-Feb-15	39.73	1854.25	98% RNC, 2% Ressources Québec
2054897	LAUNAY	CDC	13-Feb-15	42.68	1854.25	98% RNC, 2% Ressources Québec
2054898	LAUNAY	CDC	13-Feb-15	42.73	1854.25	98% RNC, 2% Ressources Québec
2054899	LAUNAY	CDC	13-Feb-15	43.2	1854.25	98% RNC, 2% Ressources Québec
2054900	LAUNAY	CDC	13-Feb-15	47.82	1854.25	98% RNC, 2% Ressources Québec
2054901	LAUNAY	CDC	13-Feb-15	38.02	1854.25	98% RNC, 2% Ressources Québec
2054902	LAUNAY	CDC	13-Feb-15	38.74	1854.25	98% RNC, 2% Ressources Québec
2137941	LAUNAY	CDC	4-Feb-15	42.63	1254.25	98% RNC, 2% Ressources Québec
2137943	LAUNAY	CDC	21-Apr-15	41.84	1254.25	98% RNC, 2% Ressources Québec
2152798	LAUNAY	CDC	19-May-14	41.89	1254.25	98% RNC, 2% Ressources Québec
2152799	LAUNAY	CDC	19-May-14	41.95	1254.25	98% RNC, 2% Ressources Québec
2180765	LAUNAY	CDC	12-Mar-15	18.67	527.75	98% RNC, 2% Ressources Québec
2180766	LAUNAY	CDC	12-Mar-15	42.49	1254.25	98% RNC, 2% Ressources Québec
2180767	LAUNAY	CDC	12-Mar-15	42.5	1254.25	98% RNC, 2% Ressources Québec
2180768	LAUNAY	CDC	12-Mar-15	42.48	1254.25	98% RNC, 2% Ressources Québec

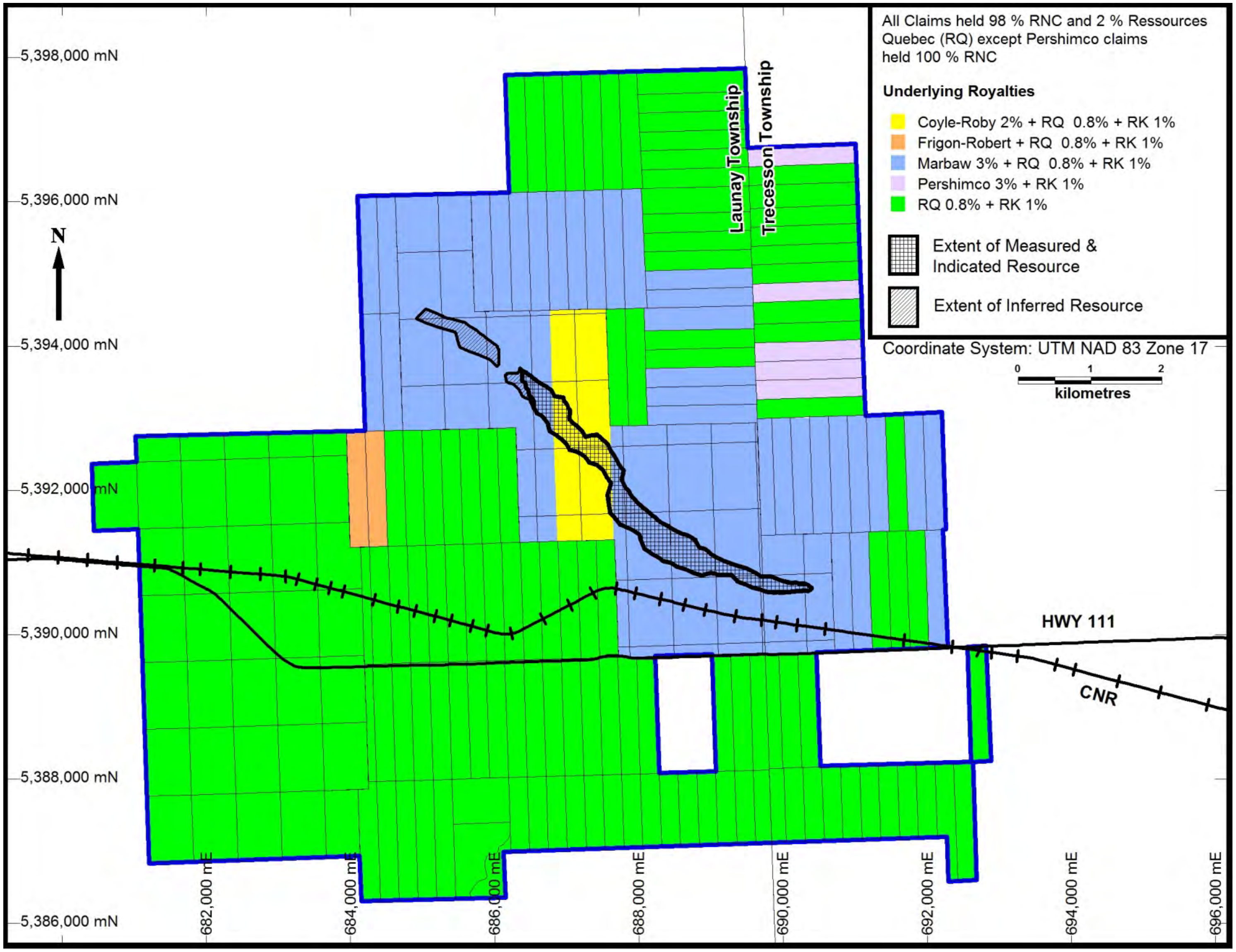
Claim Number	Township	Type	Date Renewal Due	Area (ha)	Renewal Cost ($)	Interest
2180769	LAUNAY	CDC	12-Mar-15	42.5	1254.25	98% RNC, 2% Ressources Québec
2180770	LAUNAY	CDC	12-Mar-15	42.48	1254.25	98% RNC, 2% Ressources Québec
2180771	LAUNAY	CDC	12-Mar-15	42.49	1254.25	98% RNC, 2% Ressources Québec
2180772	LAUNAY	CDC	12-Mar-15	42.49	1254.25	98% RNC, 2% Ressources Québec
2180773	LAUNAY	CDC	12-Mar-15	42.49	1254.25	98% RNC, 2% Ressources Québec
2180774	LAUNAY	CDC	12-Mar-15	42.48	1254.25	98% RNC, 2% Ressources Québec
2180775	LAUNAY	CDC	12-Mar-15	42.47	1254.25	98% RNC, 2% Ressources Québec
2180776	LAUNAY	CDC	12-Mar-15	42.49	1254.25	98% RNC, 2% Ressources Québec
2180777	LAUNAY	CDC	12-Mar-15	42.48	1254.25	98% RNC, 2% Ressources Québec
2180778	LAUNAY	CDC	12-Mar-15	42.48	1254.25	98% RNC, 2% Ressources Québec
2180779	LAUNAY	CDC	12-Mar-15	42.47	1254.25	98% RNC, 2% Ressources Québec
2180780	LAUNAY	CDC	12-Mar-15	42.48	1254.25	98% RNC, 2% Ressources Québec
2180781	LAUNAY	CDC	12-Mar-15	42.48	1254.25	98% RNC, 2% Ressources Québec
2180782	LAUNAY	CDC	12-Mar-15	42.46	1254.25	98% RNC, 2% Ressources Québec
2180783	LAUNAY	CDC	12-Mar-15	35.6	1254.25	98% RNC, 2% Ressources Québec
2180784	LAUNAY	CDC	12-Mar-15	19.53	527.75	98% RNC, 2% Ressources Québec
2180785	LAUNAY	CDC	12-Mar-15	42.61	1254.25	98% RNC, 2% Ressources Québec
2204676	LAUNAY	CDC	7-Feb-14	38.82	1254.25	98% RNC, 2% Ressources Québec
2204677	LAUNAY	CDC	7-Feb-14	38.9	1254.25	98% RNC, 2% Ressources Québec
2204678	LAUNAY	CDC	7-Feb-14	38.91	1254.25	98% RNC, 2% Ressources Québec
2204679	LAUNAY	CDC	7-Feb-14	53.04	1254.25	98% RNC, 2% Ressources Québec
2224811	LAUNAY	CDC	29-Apr-14	42.67	1254.25	98% RNC, 2% Ressources Québec
2224812	LAUNAY	CDC	29-Apr-14	42.68	1254.25	98% RNC, 2% Ressources Québec
2224813	LAUNAY	CDC	29-Apr-14	42.67	1254.25	98% RNC, 2% Ressources Québec
2224814	LAUNAY	CDC	29-Apr-14	42.68	1254.25	98% RNC, 2% Ressources Québec
2224815	LAUNAY	CDC	29-Apr-14	42.9	1254.25	98% RNC, 2% Ressources Québec
2229202	LAUNAY	CDC	4-May-14	38.86	1254.25	98% RNC, 2% Ressources Québec
2229203	LAUNAY	CDC	4-May-14	38.81	1254.25	98% RNC, 2% Ressources Québec
2247681	LAUNAY	CDC	26-Aug-14	42.68	1254.25	98% RNC, 2% Ressources Québec
2247682	LAUNAY	CDC	26-Aug-14	42.68	1254.25	98% RNC, 2% Ressources Québec
2251083	LAUNAY	CDC	23-Sep-14	41.78	1254.25	98% RNC, 2% Ressources Québec
2255618	LAUNAY	CDC	24-Oct-14	43.3	1254.25	98% RNC, 2% Ressources Québec
2255619	LAUNAY	CDC	24-Oct-14	43.33	1254.25	98% RNC, 2% Ressources Québec
2255620	LAUNAY	CDC	24-Oct-14	43.3	1254.25	98% RNC, 2% Ressources Québec
2255621	LAUNAY	CDC	24-Oct-14	43.31	1254.25	98% RNC, 2% Ressources Québec
2255622	LAUNAY	CDC	24-Oct-14	52.65	1254.25	98% RNC, 2% Ressources Québec
2255623	LAUNAY	CDC	24-Oct-14	48.65	1254.25	98% RNC, 2% Ressources Québec
2255628	LAUNAY	CDC	24-Oct-14	38.91	1254.25	98% RNC, 2% Ressources Québec
2255629	LAUNAY	CDC	24-Oct-14	39.05	1254.25	98% RNC, 2% Ressources Québec
2255630	LAUNAY	CDC	24-Oct-14	39.16	1254.25	98% RNC, 2% Ressources Québec
2255631	LAUNAY	CDC	24-Oct-14	48.2	1254.25	98% RNC, 2% Ressources Québec
2255641	LAUNAY	CDC	24-Oct-14	22.12	527.75	98% RNC, 2% Ressources Québec
2255642	LAUNAY	CDC	24-Oct-14	26.69	1254.25	98% RNC, 2% Ressources Québec
2255643	LAUNAY	CDC	24-Oct-14	26.66	1254.25	98% RNC, 2% Ressources Québec
2255644	LAUNAY	CDC	24-Oct-14	26.63	1254.25	98% RNC, 2% Ressources Québec
2255645	LAUNAY	CDC	24-Oct-14	26.6	1254.25	98% RNC, 2% Ressources Québec
2255646	LAUNAY	CDC	24-Oct-14	26.56	1254.25	98% RNC, 2% Ressources Québec
2255647	LAUNAY	CDC	24-Oct-14	26.54	1254.25	98% RNC, 2% Ressources Québec
2255648	LAUNAY	CDC	24-Oct-14	26.5	1254.25	98% RNC, 2% Ressources Québec
2255649	LAUNAY	CDC	24-Oct-14	26.48	1254.25	98% RNC, 2% Ressources Québec

Claim Number	Township	Type	Date Renewal Due	Area (ha)	Renewal Cost ($)	Interest
2255650	LAUNAY	CDC	24-Oct-14	26.43	1254.25	98% RNC, 2% Ressources Québec
2255651	LAUNAY	CDC	24-Oct-14	26.41	1254.25	98% RNC, 2% Ressources Québec
2255652	LAUNAY	CDC	24-Oct-14	26.37	1254.25	98% RNC, 2% Ressources Québec
2255653	LAUNAY	CDC	24-Oct-14	26.34	1254.25	98% RNC, 2% Ressources Québec
2255654	LAUNAY	CDC	24-Oct-14	26.3	1254.25	98% RNC, 2% Ressources Québec
2255655	LAUNAY	CDC	24-Oct-14	22.36	527.75	98% RNC, 2% Ressources Québec
2025453	TRECESSON	CDC	19-Sep-14	42.83	1854.25	98% RNC, 2% Ressources Québec
2025454	TRECESSON	CDC	19-Sep-14	42.8	1854.25	98% RNC, 2% Ressources Québec
2025455	TRECESSON	CDC	19-Sep-14	42.59	1854.25	98% RNC, 2% Ressources Québec
2025456	TRECESSON	CDC	19-Sep-14	42.58	1854.25	98% RNC, 2% Ressources Québec
2025457	TRECESSON	CDC	19-Sep-14	32.69	1854.25	98% RNC, 2% Ressources Québec
2031507	TRECESSON	CDC	6-Nov-14	42.6	1854.25	98% RNC, 2% Ressources Québec
2031508	TRECESSON	CDC	6-Nov-14	42.6	1854.25	98% RNC, 2% Ressources Québec
2031509	TRECESSON	CDC	6-Nov-14	42.58	1854.25	98% RNC, 2% Ressources Québec
2031510	TRECESSON	CDC	6-Nov-14	42.57	1854.25	98% RNC, 2% Ressources Québec
2031511	TRECESSON	CDC	6-Nov-14	42.56	1854.25	98% RNC, 2% Ressources Québec
2054109	TRECESSON	CDC	8-Feb-15	42.78	1854.25	98% RNC, 2% Ressources Québec
2054110	TRECESSON	CDC	8-Feb-15	42.75	1854.25	98% RNC, 2% Ressources Québec
2054111	TRECESSON	CDC	8-Feb-15	42.73	1854.25	98% RNC, 2% Ressources Québec
2054123	TRECESSON	CDC	8-Feb-15	42.58	1854.25	98% RNC, 2% Ressources Québec
2054892	TRECESSON	CDC	13-Feb-15	42.71	1854.25	98% RNC, 2% Ressources Québec
2054893	TRECESSON	CDC	13-Feb-15	42.41	1854.25	98% RNC, 2% Ressources Québec
2180762	TRECESSON	CDC	12-Mar-15	29.76	1254.25	98% RNC, 2% Ressources Québec
2180763	TRECESSON	CDC	12-Mar-15	41.68	1254.25	98% RNC, 2% Ressources Québec
2180764	TRECESSON	CDC	12-Mar-15	41.71	1254.25	98% RNC, 2% Ressources Québec
2194108	TRECESSON	CDC	9-Nov-13	39.25	1254.25	100% RNC
2194109	TRECESSON	CDC	9-Nov-13	39.27	1254.25	100% RNC
2194110	TRECESSON	CDC	9-Nov-13	39.26	1254.25	100% RNC
2194115	TRECESSON	CDC	9-Nov-13	38.73	1254.25	100% RNC
2204674	TRECESSON	CDC	7-Feb-14	39.12	1254.25	98% RNC, 2% Ressources Québec
2204675	TRECESSON	CDC	7-Feb-14	39.13	1254.25	98% RNC, 2% Ressources Québec
2220724	TRECESSON	CDC	25-Apr-14	39.12	1254.25	100% RNC
2229201	TRECESSON	CDC	4-May-14	39.22	1254.25	98% RNC, 2% Ressources Québec
2249118	TRECESSON	CDC	8-Sep-14	39.25	1254.25	98% RNC, 2% Ressources Québec
2255617	TRECESSON	CDC	24-Oct-14	42.91	1254.25	98% RNC, 2% Ressources Québec
2255624	TRECESSON	CDC	24-Oct-14	41.92	1254.25	98% RNC, 2% Ressources Québec
2255625	TRECESSON	CDC	24-Oct-14	39.09	1254.25	98% RNC, 2% Ressources Québec
2255626	TRECESSON	CDC	24-Oct-14	47.12	1254.25	98% RNC, 2% Ressources Québec
2255627	TRECESSON	CDC	24-Oct-14	39.19	1254.25	98% RNC, 2% Ressources Québec
2255656	TRECESSON	CDC	24-Oct-14	20.8	527.75	98% RNC, 2% Ressources Québec
2255657	TRECESSON	CDC	24-Oct-14	26.82	1254.25	98% RNC, 2% Ressources Québec
2255658	TRECESSON	CDC	24-Oct-14	26.81	1254.25	98% RNC, 2% Ressources Québec
2255659	TRECESSON	CDC	24-Oct-14	26.79	1254.25	98% RNC, 2% Ressources Québec
2255660	TRECESSON	CDC	24-Oct-14	26.79	1254.25	98% RNC, 2% Ressources Québec
2255661	TRECESSON	CDC	24-Oct-14	26.78	1254.25	98% RNC, 2% Ressources Québec
2255662	TRECESSON	CDC	24-Oct-14	26.76	1254.25	98% RNC, 2% Ressources Québec
2255663	TRECESSON	CDC	24-Oct-14	26.76	1254.25	98% RNC, 2% Ressources Québec
2255664	TRECESSON	CDC	24-Oct-14	26.69	1254.25	98% RNC, 2% Ressources Québec
2255665	TRECESSON	CDC	24-Oct-14	35.26	1254.25	98% RNC, 2% Ressources Québec
2276187	TRECESSON	CDC	8-Mar-15	39.29	1254.25	98% RNC, 2% Ressources Québec

Claim Number	Township	Type	Date Renewal Due	Area (ha)	Renewal Cost ($)	Interest
2276188	TRECESSON	CDC	8-Mar-15	45.83	1254.25	98% RNC, 2% Ressources Québec
2180786	LAUNAY	CDC	12-Mar-15	56.93	1254.25	98% RNC, 2% Ressources Québec
2180787	LAUNAY	CDC	12-Mar-15	56.93	1254.25	98% RNC, 2% Ressources Québec
2180788	LAUNAY	CDC	12-Mar-15	56.93	1254.25	98% RNC, 2% Ressources Québec
2180789	LAUNAY	CDC	12-Mar-15	56.93	1254.25	98% RNC, 2% Ressources Québec
2180790	LAUNAY	CDC	12-Mar-15	56.93	1254.25	98% RNC, 2% Ressources Québec
2180791	LAUNAY	CDC	12-Mar-15	56.92	1254.25	98% RNC, 2% Ressources Québec
2180792	LAUNAY	CDC	12-Mar-15	56.92	1254.25	98% RNC, 2% Ressources Québec
2180793	LAUNAY	CDC	12-Mar-15	56.92	1254.25	98% RNC, 2% Ressources Québec
2180794	LAUNAY	CDC	12-Mar-15	56.92	1254.25	98% RNC, 2% Ressources Québec
2180795	LAUNAY	CDC	12-Mar-15	56.92	1254.25	98% RNC, 2% Ressources Québec
2180796	LAUNAY	CDC	12-Mar-15	56.91	1254.25	98% RNC, 2% Ressources Québec
2180797	LAUNAY	CDC	12-Mar-15	56.91	1254.25	98% RNC, 2% Ressources Québec
2180798	LAUNAY	CDC	12-Mar-15	56.91	1254.25	98% RNC, 2% Ressources Québec
2180799	LAUNAY	CDC	12-Mar-15	56.91	1254.25	98% RNC, 2% Ressources Québec
2180800	LAUNAY	CDC	12-Mar-15	51.74	1254.25	98% RNC, 2% Ressources Québec
2180801	LAUNAY	CDC	12-Mar-15	56.9	1254.25	98% RNC, 2% Ressources Québec
2180802	LAUNAY	CDC	12-Mar-15	56.9	1254.25	98% RNC, 2% Ressources Québec
2180803	LAUNAY	CDC	12-Mar-15	56.9	1254.25	98% RNC, 2% Ressources Québec
2180804	LAUNAY	CDC	12-Mar-15	56.9	1254.25	98% RNC, 2% Ressources Québec
2180805	LAUNAY	CDC	12-Mar-15	43.32	1254.25	98% RNC, 2% Ressources Québec
2180806	LAUNAY	CDC	12-Mar-15	21.78	527.75	98% RNC, 2% Ressources Québec
2180807	LAUNAY	CDC	12-Mar-15	21.5	527.75	98% RNC, 2% Ressources Québec
2180808	LAUNAY	CDC	12-Mar-15	21.1	527.75	98% RNC, 2% Ressources Québec
2180809	LAUNAY	CDC	12-Mar-15	20.68	527.75	98% RNC, 2% Ressources Québec
2180810	LAUNAY	CDC	12-Mar-15	15.48	527.75	98% RNC, 2% Ressources Québec
2235659	LAUNAY	CDC	12-Mar-15	56.94	1254.25	98% RNC, 2% Ressources Québec
2255632	LAUNAY	CDC	24-Oct-14	56.94	1254.25	98% RNC, 2% Ressources Québec
2255633	LAUNAY	CDC	24-Oct-14	56.94	1254.25	98% RNC, 2% Ressources Québec
2255634	LAUNAY	CDC	24-Oct-14	56.94	1254.25	98% RNC, 2% Ressources Québec
2255635	LAUNAY	CDC	24-Oct-14	56.94	1254.25	98% RNC, 2% Ressources Québec
2255636	LAUNAY	CDC	24-Oct-14	56.93	1254.25	98% RNC, 2% Ressources Québec
2255637	LAUNAY	CDC	24-Oct-14	56.93	1254.25	98% RNC, 2% Ressources Québec
2255638	LAUNAY	CDC	24-Oct-14	56.93	1254.25	98% RNC, 2% Ressources Québec
2255639	LAUNAY	CDC	24-Oct-14	56.94	1254.25	98% RNC, 2% Ressources Québec
2255640	LAUNAY	CDC	24-Oct-14	43.32	1254.25	98% RNC, 2% Ressources Québec
2267113	LAUNAY	CDC	11-Jan-15	56.9	1254.25	98% RNC, 2% Ressources Québec
2377418	LAUNAY	CDC	13-Jan-14	56.92	3346.39	98% RNC, 2% Ressources Québec
2377419	LAUNAY	CDC	13-Jan-14	56.92	3346.39	98% RNC, 2% Ressources Québec
2377420	LAUNAY	CDC	13-Jan-14	56.92	3346.39	98% RNC, 2% Ressources Québec
2377421	TRECESSON	CDC	13-Jan-14	56.92	3346.39	98% RNC, 2% Ressources Québec
2377422	LAUNAY	CDC	13-Jan-14	56.91	3345.82	98% RNC, 2% Ressources Québec
2377423	LAUNAY	CDC	13-Jan-14	56.91	3345.82	98% RNC, 2% Ressources Québec
2377424	LAUNAY	CDC	13-Jan-14	56.91	3345.82	98% RNC, 2% Ressources Québec
2377425	LAUNAY	CDC	13-Jan-14	56.9	3345.24	98% RNC, 2% Ressources Québec
2377426	LAUNAY	CDC	13-Jan-14	56.9	3345.24	98% RNC, 2% Ressources Québec
2377427	LAUNAY	CDC	13-Jan-14	56.9	3345.24	98% RNC, 2% Ressources Québec
2377428	LAUNAY	CDC	13-Jan-14	56.9	3345.24	98% RNC, 2% Ressources Québec
2377429	LAUNAY	CDC	13-Jan-14	56.9	3345.24	98% RNC, 2% Ressources Québec
2377430	LAUNAY	CDC	13-Jan-14	56.89	3344.66	98% RNC, 2% Ressources Québec

Claim Number	Township	Type	Date Renewal Due	Area (ha)	Renewal Cost ($)	Interest
2377431	LAUNAY	CDC	13-Jan-14	56.88	3344.08	98% RNC, 2% Ressources Québec
2377432	LAUNAY	CDC	13-Jan-14	56.88	3344.08	98% RNC, 2% Ressources Québec
2377433	LAUNAY	CDC	13-Jan-14	56.88	3344.08	98% RNC, 2% Ressources Québec
2377434	LAUNAY	CDC	13-Jan-14	36.08	2141.05	98% RNC, 2% Ressources Québec
2377435	LAUNAY	CDC	13-Jan-14	54.69	3217.42	98% RNC, 2% Ressources Québec
2377436	LAUNAY	CDC	13-Jan-14	54.41	3201.22	98% RNC, 2% Ressources Québec
2377437	LAUNAY	CDC	13-Jan-14	46.65	2752.4	98% RNC, 2% Ressources Québec
2377438	LAUNAY	CDC	13-Jan-14	37.9	2246.32	98% RNC, 2% Ressources Québec
2377439	LAUNAY	CDC	13-Jan-14	43.69	2581.2	98% RNC, 2% Ressources Québec
2377440	LAUNAY	CDC	13-Jan-14	36.43	2161.29	98% RNC, 2% Ressources Québec
2377441	LAUNAY	CDC	13-Jan-14	9.06	551.77	98% RNC, 2% Ressources Québec
2377442	LAUNAY	CDC	13-Jan-14	23.21	1370.17	98% RNC, 2% Ressources Québec
2377443	LAUNAY	CDC	13-Jan-14	45.83	2704.96	98% RNC, 2% Ressources Québec
2377444	LAUNAY	CDC	13-Jan-14	4.39	281.65	98% RNC, 2% Ressources Québec
2377445	LAUNAY	CDC	13-Jan-14	22.27	1315.8	98% RNC, 2% Ressources Québec
2377446	LAUNAY	CDC	13-Jan-14	3.95	256.21	98% RNC, 2% Ressources Québec
2377447	LAUNAY	CDC	13-Jan-14	2.28	159.62	98% RNC, 2% Ressources Québec
2377448	LAUNAY	CDC	13-Jan-14	14.85	886.64	98% RNC, 2% Ressources Québec
2377449	LAUNAY	CDC	13-Jan-14	31.37	1868.62	98% RNC, 2% Ressources Québec
2377450	LAUNAY	CDC	13-Jan-14	45.79	2702.65	98% RNC, 2% Ressources Québec
2377451	LAUNAY	CDC	13-Jan-14	40.94	2422.13	98% RNC, 2% Ressources Québec
2377452	LAUNAY	CDC	13-Jan-14	2.57	176.39	98% RNC, 2% Ressources Québec
2377453	LAUNAY	CDC	13-Jan-14	8.83	538.45	98% RNC, 2% Ressources Québec
2377454	LAUNAY	CDC	13-Jan-14	17.22	1023.72	98% RNC, 2% Ressources Québec
2377455	LAUNAY	CDC	13-Jan-14	9.02	549.44	98% RNC, 2% Ressources Québec
2377456	LAUNAY	CDC	13-Jan-14	16.77	997.69	98% RNC, 2% Ressources Québec
2377457	LAUNAY	CDC	13-Jan-14	9.21	560.43	98% RNC, 2% Ressources Québec
2377458	LAUNAY	CDC	13-Jan-14	16.32	971.66	98% RNC, 2% Ressources Québec
2377459	TRECESSON	CDC	13-Jan-14	10.18	616.54	98% RNC, 2% Ressources Québec
2377460	TRECESSON	CDC	13-Jan-14	35.03	2080.31	98% RNC, 2% Ressources Québec
2377461	LAUNAY	CDC	13-Jan-14	2.88	194.32	98% RNC, 2% Ressources Québec
2377462	LAUNAY	CDC	13-Jan-14	0.81	74.59	98% RNC, 2% Ressources Québec
2377463	TRECESSON	CDC	13-Jan-14	6.39	397.33	98% RNC, 2% Ressources Québec
2377464	TRECESSON	CDC	13-Jan-14	35.71	2119.64	98% RNC, 2% Ressources Québec
2377465	TRECESSON	CDC	13-Jan-14	21.18	1252.75	98% RNC, 2% Ressources Québec

Figure 4.2: Dumont Property Mineral Claims



Source: RNC.

4.2.3 Underlying Agreements

The Dumont mineral claims are subject to various royalty agreements arising from terms of the property acquisitions by RNC or through the sale of royalties. The details of the underlying mineral claim agreements are described below and the extent and location of the property subject to the agreements are shown in Figure 4.2.

4.2.3.1 Marbaw Royalty

The Marbaw International Nickel Corporation (Marbaw) property comprises an area totalling 2,639.0 ha as shown in Figure 4.2. This area originally consisted of 65 claims. Thirty-four of these claims were ground-staked claims that were converted to map-staked claims by the MRN in 2013.

This property was originally held by Marbaw, but a 100% interest in the claims was sold and transferred to RNC for future consideration under an agreement dated 8 March 2007.

Future consideration consisted of the following: (1) issuance of 7 million shares in RNC to Marbaw upon the property being placed into commercial production or upon transfer of the property to a third party; (2) payment of $1.25 M to Marbaw on 8 March 2008. This amount has been paid by RNC.

RNC also committed to incurring a minimum expenditure of $8,000,000 on the property prior to ceasing operations. This commitment was met in 2008. The Marbaw property is subject to a 3% NSR royalty payable to Marbaw. RNC has the right to buy back half of the 3% NSR for $10 M at any time.

This property is subject to the Ressources Québec royalty and Red Kite royalty.

4.2.3.2 Coyle-Roby Royalty

The Sheridan-Ferderber property comprises an area of 256.47 ha corresponding to six historical contiguous ground-staked claims (Figure 4.2). The claims corresponding to the Sheridan-Ferderber property were converted to map staked claims in 2013.

The property was originally held 50% by Terrence Coyle and 50% by Michel Roby, but they were optioned to Patrick Sheridan and Peter Ferderber under an agreement dated 26 October 2006. The option agreement was subsequently assigned to RNC through an agreement dated 4 May 2007.

RNC's option to acquire 100% interest in this property was exercised by the completion of $75,000 in work on the property before 26 October 2008 and by paying $10,000 to Coyle-Roby by 26 October 2007 and $30,000 to Coyle-Roby by 26 October 2008. The claims were transferred 100% to RNC on 25 August 2008.

The property is subject to a 2% NSR royalty payable to Terrence Coyle (1%) and Michel Roby (1%). RNC has the right to buy back half of this 2% NSR for $1 M at any time. An advance royalty of $5,000 per year is also payable to Coyle-Roby beginning in 2011. Scheduled royalty payments were made in October 2011 and October 2012.These claims are subject to the Ressources Québec royalty and Red Kite royalty.

4.2.3.3 Frigon-Robert Royalty

The Frigon-Robert property comprises two contiguous claims totalling 83.84 ha. The claims were originally held 50% by Jacques Frigon and 50% by Gérard Robert. They were transferred to RNC through a purchase agreement dated 1 November 2010.

The property is subject to a 2% NSR royalty payable to Jacques Frigon (1%) and Gérard Robert (1%). RNC has the right to buy back half of this 2% NSR for $1 M at any time.

These claims are subject to the Ressources Québec royalty and Red Kite royalty.

4.2.3.4 Pershimco Claims (Pershimco Royalty)

The Pershimco mineral claim block comprises five claims totalling 195.64 ha. The claims were originally held 100% by Pershimco Resources. They were transferred to RNC through a purchase agreement dated 18 March 2013 for $30,000. These claims are subject to a 3% NSR royalty payable to Pershimco Resources. RNC has the option to buy back the NSR in stages at any time by paying $1,000,000 for the first percent, $3,000,000 for the second percent and $6,000,000 for the third percent. As these claims were acquired after the Ressources Québec agreement, they are not subject to the Ressources Québec royalty.

These claims are subject to the Red Kite royalty.

4.2.3.5 Ressources Québec Royalty

On 1 August 2012, RNC entered into an investment agreement with Ressources Québec. Pursuant to the agreement, RNC received $12 million and Ressources Québec became entitled to receive 0.8% of the net smelter return from the sale of minerals produced from Dumont and acquired a 2% undivided co-ownership interest in the property. RNC has the right to repurchase, at any time after the fifth anniversary, all or any portion of Ressources Québec's interest for $10 million for each 0.2% of the net smelter return, to a maximum consideration of $40 million for the entire interest (including the 2% interest in the property). The Ressources Québec royalty applies to all Dumont claims except the five Pershimco claims that were acquired after the Ressources Québec agreement.

4.2.3.6 Red Kite Royalty

On 9 May 2013, RNC entered into an investment agreement with RK Mine Finance (Master) Fund II LP ("Red Kite"). Under the terms of the agreement, Red Kite acquired a 1% net smelter return royalty in the Dumont project for a purchase price of US$15 million. The Red Kite royalty applies to all Dumont claims listed in Table 4-1.

4.3 Exploration Permits & Authorizations

Exploration work on public land (Crown land) is conducted under a forestry operational permit granted by the Quebec Ministry of Natural Resources and Wildlife (MNRF) and renewed periodically. Exploration work on agricultural zoned lands is conducted under a permit granted by the Quebec Agricultural Land Commission (CPTAQ). Exploration work on private surface rights not owned by RNC is conducted under the terms of access agreements between RNC and individual landowners. Stream crossings have been constructed under permits issued variously or jointly by the MNRF, CPTAQ, and the Quebec Ministry of Sustainable Development, Environment and Parks (MDDEP). RNC advises there are no known formal native land claims on the territory of the Dumont property within the St. Lawrence drainage basin. Algonquin First Nations; however, assert aboriginal rights over parts of western Quebec and eastern Ontario. Consultation with First Nations is a responsibility of the federal and provincial governments. Nonetheless, RNC has initiated discussions with the local Algonquin Conseil de la Première nation Abitibiwinni and on 5 April 2013 entered into a memorandum of understanding for cooperation regarding the development of the Dumont Nickel project.

4.4 Mineral Rights in Quebec

RNC advises that under Quebec Mining Law, the holder of a claim has the exclusive right to explore for mineral substances (other than petroleum, natural gas and brine, sand, gravel and other surfaces substances) on the parcel of land subject to the claim. A claim has a term of two years. It may be renewed for additional periods of two years by completing minimum exploration work requirements and paying renewal fees. The holder of one or more claims may obtain a mining lease for the parcels of land subject to such claims, provided the holder can prove the existence of a workable deposit on the property.

The mineral claims confer subsurface mineral rights only. Surface rights tenure is shown in Figure 4.3 on the following page. Approximately 40% of the surface rights for the property are held privately by a number of owners, resident both in the area and outside the region. RNC has purchased or acquired options to purchase approximately 680 ha of private surface rights overlying the Dumont resource as shown in Figure 4.3. The remainder of the surface rights are public land (Crown land).

Figure 4.3 (overleaf) also shows the extent of the lands that are classified as an agricultural zone, where agricultural land and agricultural activities are to be respected and preserved. Mining activity on these lands would require authorization for non-agricultural use or exclusion of these lands from the agricultural zone by the Quebec Agricultural Land Commission (CPTAQ). This must be requested by the local municipality. The application for exclusion must demonstrate that there are no suitable non-agricultural lands available for the stated purpose in the municipality. RNC does not expect that exclusion of these lands to develop the Dumont project would be unreasonably withheld. The application exclusion for has been filed and was received by the CPTAQ on 20 February 2013.

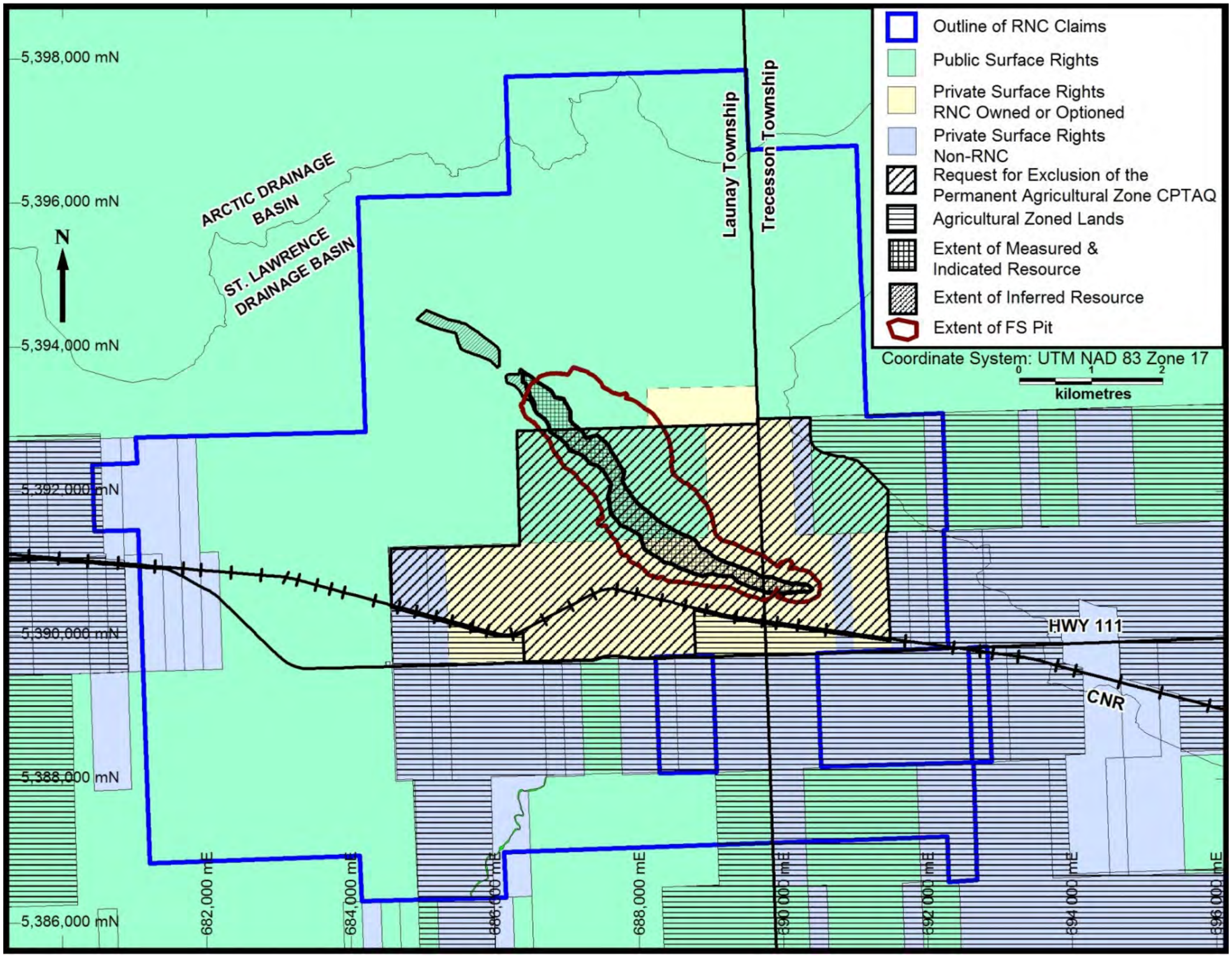
Use of surface rights for mining and associated activities under the terms of a mining lease is subject to environmental permitting. Access to surface rights for private lands would be obtained by negotiating purchase from private surface rights holders. Access to surface rights for public lands would be obtained through the mining lease and surface lease processes. Prior to commencing any mining, the operator of a mine or mill on the land subject to a lease must submit a rehabilitation and restoration plan for the site and deposit a financial guarantee. No compensation may be claimed by the holder of a mining claim from the holder of a mining lease for the depositing of tailings on the parcel of land that is subject to the claim.

4.5 Environmental Liabilities

Neither Ausenco nor RNC is aware of any outstanding environmental liabilities attached to the Dumont property and is unable to comment on any remediation that may have been undertaken by previous companies.

Additional detail on environmental matters is provided in Section 20.

Figure 4.3: Dumont Property Surface Considerations



Source: RNC.

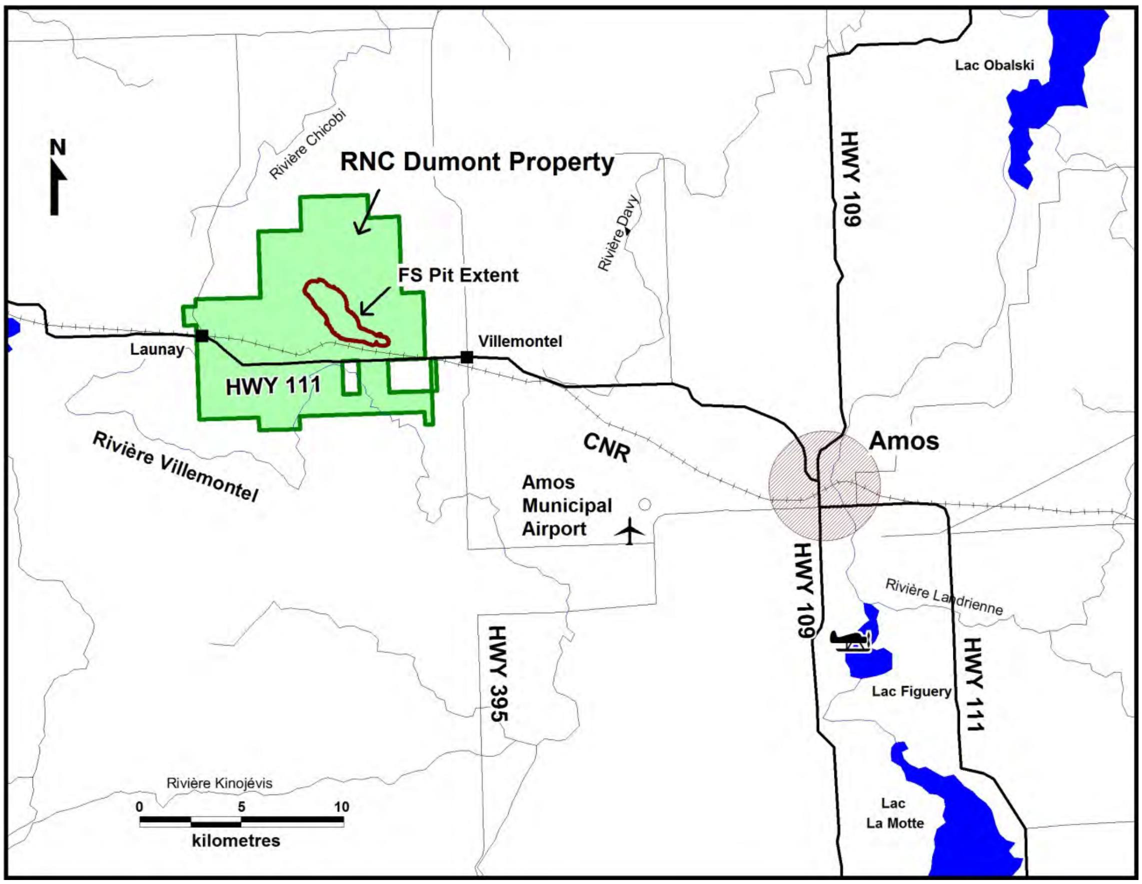
Ausenco

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

5.1 Accessibility

The Dumont property is located in the province of Quebec; approximately 25 km northwest of the city of Amos (see Figure 5.1).

Figure 5.1: Location & Infrastructure



Source: RNC.

5.2 Local Resources & Infrastructure

The principal economic activities in the region are agriculture and forestry. The sustainable nature of these industries has contributed to a stable population. As a result, Amos is well serviced by a large number of businesses and industrial suppliers. The Dumont Nickel project would require construction of additional accommodation in town, but the municipal economy is sufficiently evolved and diversified that responsibility for the investment in, and construction of, additional accommodation would likely be provided by third parties. The existing infrastructure in town is likely adequate to support the expanded population.

Amos has a municipal airport but is not serviced by regularly scheduled commercial flights. The nearest cities with airports serviced by regularly scheduled flights are Rouyn-Noranda (2011 Census population 41,012), which is 120 km by road to the southwest, and Val d'Or (2011 Census population 31,862), which is 90 km by road to the southeast. Both Rouyn-Noranda and Val d'Or have traditionally been centres for the mining industry, and there is a large base of skilled mining personnel resident within the region.

The project site is well serviced with respect to other infrastructure, including:

- Road – Provincial Highway 111 runs along the southern boundary of the property.
- Rail – The Canadian National Railway (CNR) runs through the property, slightly to the north of Highway 111 but south of the engineered pit.
- Power – The provincial utility, Hydro-Quebec, has indicated that it would be feasible to extend the powerline to site from the high voltage line that runs 5 km south of Highway 111 and that power from the grid would be made available to the project.
- Water – The project concept includes a closed system for water, with water that would be reclaimed from tailings being reused in the process plant. Make-up water would be taken from the southeast reservoir and, if required under exceptional circumstances, from the Villemontel River, at a point located approximately 5 km from the planned site for the mill.
- Natural Gas – Although the use of natural gas is not considered in this study, an existing pipeline extends to within approximately 25 km to the south of the property.

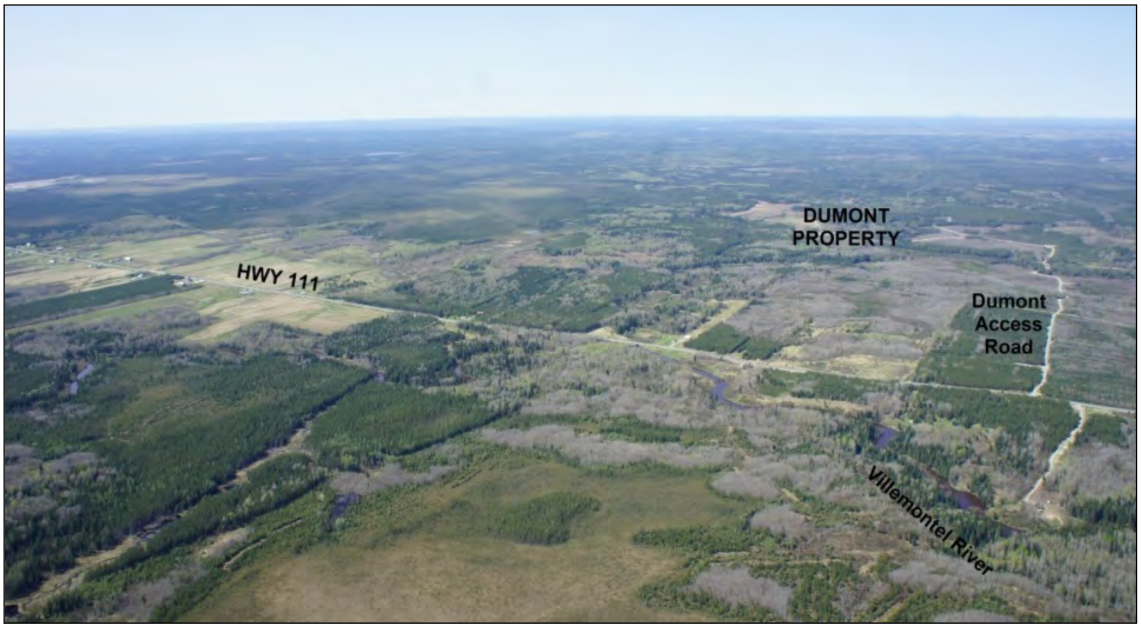
5.3 Climate

The climate at the Dumont property is continental with mean temperatures ranging from -17.3°C in January to +17.2°C in July, with an annual mean temperature of 1.2°C. Total average annual precipitation is 918 mm. While field exploration work can be conducted year-round, drill access in low-lying boggy areas is best during the frozen winter months. Also, periodic heavy rainfall or snowfall can hamper exploration at times during the summer or winter months. The climate at Dumont would be suitable to year-round open-pit mining operations. The climate setting is analogous to that of the former Dome Mine open-pit near Timmins, Ontario or Osisko's Canadian Malartic open-pit mine 60 km to the south of Dumont.

5.4 Physiography

The property exhibits low to moderate relief up to a maximum of 40 m and lies between 310 and 350 m above sea level (Figure 5.2). The Arctic-Atlantic continental drainage divide runs along the northern boundary of the property as shown in Figure 5.3. Water for the diamond drilling programs is obtained from several creeks which run through the property and is generally pumped to the drill sites. However, fresh water can also be supplied by the nearby Villemontel River. Wildlife on the property consists of moose, black bear, beaver, rabbit and deer. Some logging has been conducted on the property with the wood being used primarily for pulp.

Figure 5.2: View of Dumont Property from the South



Source: RNC.

Figure 5.3: Dumont Property showing Typical Flat Topography, Drill Rig & Localized Clear-cutting

Source: RNC.

5.5 Surface Rights

Surface rights tenure is shown in Section 4 in Figure 4.3. Approximately 40% of the surface rights for the property are held privately by a number of owners, resident both in the area and outside the region. Of these privately held surface rights approximately 1,409 hectares are required for the development of the Dumont project. To date, RNC has purchased or acquired options to purchase approximately 1,200 ha or 85% of the private surface rights required for the development of the Dumont project as shown in Figure 4.3, and negotiations are in progress to purchase the remaining private surface rights. The remainder of the surface rights are public land (Crown land). `

Figure 4.3 shows the extent of the lands that are classified as an agricultural zone within the meaning of the Act respecting the preservation of agricultural land and agricultural activities. Mining activity on these lands would require authorization for non-agricultural use or exclusion of these lands from the agricultural zone by the Quebec Agricultural Land Commission (CPTAQ). This exclusion must be requested by the local municipality. The application for exclusion must demonstrate that there are no suitable non-agricultural lands available for the stated purpose in the municipality. RNC does not expect that exclusion for the purpose of developing the Dumont project would be unreasonably withheld. The application for exclusion has been filed and was received by the CPTAQ on 20 February 2013.

Use of surface rights for mining and associated activities under the terms of a mining lease is subject to environmental permitting. Access to surface rights for private lands not yet secured by RNC would be obtained by negotiating purchase thereof from private surface rights holders. Access to surface rights for public lands would be obtained through the mining lease and surface lease processes. No compensation may be claimed by the holder of a mining claim from the holder of a mining lease for the depositing of tailings on the parcel of land that is subject to the claim.

6 HISTORY

6.1 Exploration & Development Work

While the presence of ultramafic and mafic rocks has been known on the Dumont property since 1935, the presence of nickel within the rock sequence was only discovered in 1956. It was not until the 1970s that the existence and potential of the large low-grade nickel mineralization was first recognized.

The major exploration phases for the Dumont property are discussed below with the exploration and associated work listed in point form by year.

6.1.1 Phase 1: 1935 to 1969

The exploration programs and geological surveys during this period led to the discovery of the Dumont ultramafic sill and associated nickel mineralization.

In 1935, the Geological Survey of Canada (GSC) conducted a mapping survey over Launay and Trécesson Townships that identified the presence of ultramafic and mafic rocks.

In 1950, Quebec Asbestos Corporation (Quebec Asbestos) conducted a magnetometer survey over the upper contact of the sill and drilled five diamond drill holes totalling 475 m.

In 1951, an aeromagnetic survey conducted by the GSC outlined the ultramafic sill.

In 1956, Barry Exploration Ltd. (Barry Exploration) conducted a magnetometer survey over the group of claims previously explored by Quebec Asbestos and drilled a further six diamond drill holes. These drill holes resulted in the first reporting of the presence of nickel mineralization.

6.1.2 Phase 2: 1969 to 1982

The exploration programs and related geological and engineering studies during this period resulted in the identification of three zones of nickel mineralization.

In 1969, drill holes DT-1 and DT-2, totalling 182 m, were drilled over a group of mineral claims acquired in 1962 by Georges H. Dumont, P. Eng.

In 1970, drill holes DT-3 and DT-4, totalling 364 m, were drilled on an enlarged group of claims with nickel mineralization intersected in each drill hole (DT-3: 0.47% Ni over 2.7 m). Additional mineral claims were acquired to form what was then known as the Dumont property covering the whole of the Dumont ultramafic sill.

In 1970-1971, an enlarged exploration campaign was carried out on the Dumont property that consisted of prospecting, trenching, magnetometer survey and the drilling of an additional 57 diamond drill holes, totalling 21,052 m. The drilling program discovered three zones of nickel mineralization that were nearly adjacent and parallel within the dunite subzone. The central part of the middle zone, having higher nickel content, was identified as the Main Zone or Main deposit. A portion of the Main Zone is also referred to as the No. 1 deposit where it is defined as the middle mineralized band located between sections 35+00W and 49+00W and located between surface and the 1,500 ft (457.18 m) level (Dumont, 1970/1971a,b; Dumont, 1971/1972).

In 1971, Newmont Exploration Ltd. (Newmont) conducted metallurgical testwork (heavy media and magnetic separation only) and a mineralogical study on the mineralization (Hausen, 1971). Also in that year, Canada Department of Energy, Mines and Resources, Ottawa, conducted a "Mineralogical Investigation of the Low-Grade Nickel-Bearing Serpentinite of Dumont Nickel Corporation, Val d'Or, Quebec," a study that involved XRD and electron microprobe analysis of the nickel-bearing phases (Harris, 1972).

In 1971-1972, the Centre de Recherches Minérales (CRM) carried out a laboratory testwork program on drill core composite samples from the Main Zone, including locked-cycle tests to develop the flowsheet for the concentration process. Pilot plant tests were also conducted on a bulk sample, blasted out of an outcrop located to the east of the Main Zone.

In 1971-1972, the engineering firm Caron, Dufour, Séguin & Associates (CDS) completed an ore reserve estimation and feasibility study on the project with the objective of bringing the Main deposit into production, to a depth of 455 m below surface using underground mining methods. The mineral resources of the Main deposit were estimated at 15,517,662 tonnes grading 0.646% nickel after dilution. Based on the results of the feasibility study, CDS recommended that the Main deposit be brought into production (Caron, 1972; Honsberger, 1971a,b).

In 1974-1975, in association with Dumont Nickel Corporation (Dumont Nickel), Timiskaming Nickel Ltd. (Timiskaming) paid for bench and pilot plant tests to be conducted at the University of Minnesota to evaluate the amenability of the low-grade resources to a patented process. Timiskaming and Boliden AB, which evaluated the testwork results, concluded positively that the project had economic potential for a 13,600 t/d open pit mining operation on the estimated 320 Mt of resources at 0.34% nickel, from which the patented segregation process would recover 75% of the nickel.

In 1974, Canex Placer (Canex) had bench tests conducted at Britton Research Centre Ltd. (Britton Research), where a combined flotation-hydrometallurgical process was developed to recover 80% of the nickel contained in the Main Zone. The testwork indicated that this process would also result in the production of magnesia (MgO).

After 1974, with lower nickel prices in the world market, there was reduced interest in developing the property due to the low-grade nature of the deposit.

6.1.3 Phase 3: 1982 to 1992

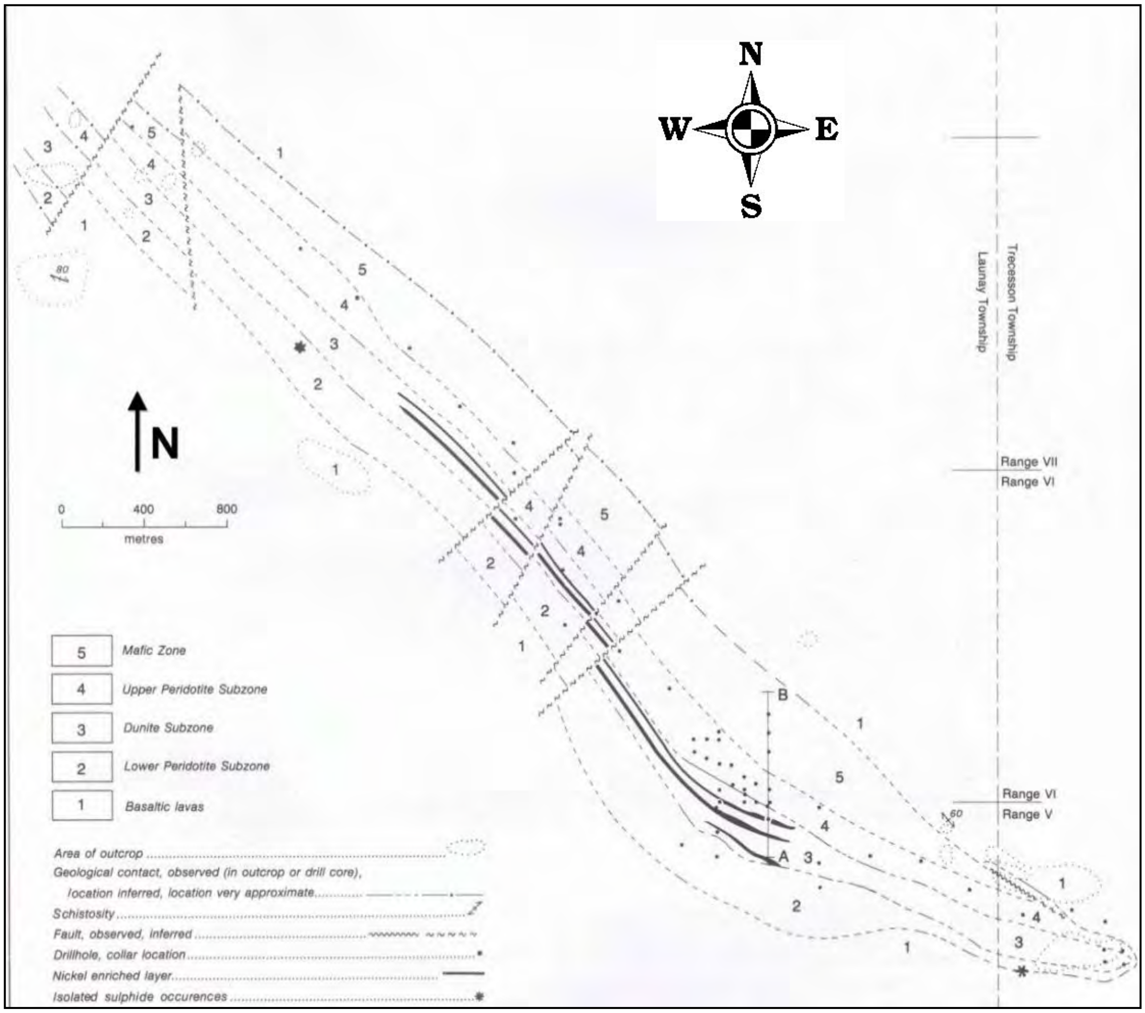
In 1982, exploration resumed on the property and four percussion 15.2 cm (6") diameter holes were drilled and cuttings recovered to prepare a bulk sample.

In 1986, CRM conducted, for the account of Magnitec, a H_2S0_3 leaching test on samples of "rejects from the Dumont mine" to evaluate the possibility of scrubbing the Noranda smelter SO_2-bearing gas with the tailings from an eventual mining operation on the property (Delisle, 1992). The test solubilized 66% of the MgO and 72.4% of the nickel contained in the samples. Magnitec also tested two core samples for their platinum group element (PGE) content but none was detected.

In 1986, La Société Nationale de l'Amiante (SNA) reviewed the results of the CRM H2S03 leach test and indicated that the tailings from an operation on the Dumont property would give a low extraction rate of the SO2 contained in the Noranda smelter emission gas.

In 1986, J. M. Duke, a geologist from the GSC, studied the mineralization and petrogenesis of the Dumont sill. Figure 6.1 is the geology map for the Dumont sill as outlined by Duke.

Figure 6.1: Geology of the Dumont Sill



Source: Supplied by RNC after Duke (1986).

From his understanding of the sill petrogenesis, Duke concluded that it was possible to discover sulphide enrichment zones at the basal contact of the intrusion and recommended that drilling should be conducted to explore this contact. In his 1986 report, Duke estimated the potential resources for the Dumont property at 175 Mt grading 0.47% nickel over the three nickel enriched layers.

In 1986 and 1987, Dumont Nickel carried out a geological mapping survey along the basal contact of the sill and drilled 11 holes in mineral claims located in Trécesson Township. Sulphide mineralization was recognized at the basal contact and a relatively high-grade nickel sulphide accumulation was intersected by four holes that also returned significant PGE values. Three holes drilled in the central part of the Dumont property were stopped short due to poor ground conditions in a faulted area (Daxl, 1988).

In 1988 and 1990, Beep Mat (electromagnetic) and induced polarization surveys were carried out for Dumont Nickel and various anomalies were reported.

In 1992, CRM conducted dry grinding and air aspiration tests to separate the fibrous texture minerals, for the account of Timmins Nickel Inc. (Timmins Nickel).

After 1992 exploration interest in the Dumont property waned and no work was conducted on the property for a number of years.

6.1.4 Phase 4: 1999 to 2006

Since 1999, the following exploration work has been conducted on the Dumont property on behalf of Frank Marzoli.

In 1999, diamond drill hole FM-99-01 was drilled on the southwest of the Main deposit. This 318 m drill hole intersected the basal sill contact but no significant mineralization was encountered.

In 2001, geological and prospecting work was carried out together with the establishment of a network of cut grid lines totalling 96 km.

In 2002, a 150 m long diamond drill hole (DNN-2002-01) was drilled in the northwest portion of the property; however, no core samples were assayed from this hole (Derosier, 2002).

In 2003, a 125 m long diamond drill hole (DNS-03-01) was positioned on section line 36+00 W. This drill hole was successful in intersecting the upper part of the Main deposit and returned a 19.2 m drill core intersection grading 0.56% nickel.

In 2004, diamond drill hole DNN-01-04 was drilled to a length of 125 m in the northwestern portion of the property with no significant results obtained from the eight 2.5 m long core intersections that were assayed (Berthelot and Cloutier, 2004).

In 2004, J.C. Caron, P.Eng, former principal of CDS and then with Les Consultants PROTEC, prepared a valuation report on the property in accordance with CIM valuation standards and guidelines.

There was no exploration activity from 2005 to 2006.

6.1.5 Phase 5: 2007 to present (RNC)

RNC acquired the property in 2007 and initiated field exploration work in March 2007. Exploration work completed by RNC since 2007 is described in Section 9. Metallurgical and process development work completed by RNC since 2007 is described in Section 13. Resource estimations are described in Section 14.

Recent development studies completed by RNC are summarized below. These studies are superseded by the current study presented in Sections 15 to 22 of this report.

6.1.5.1 RNC Conceptual Study

After Dumont was acquired by RNC, a conceptual study was completed by Aker Solutions in October 2007 and updated in August 2008. The initial report was based on historical resource estimates, which pre-dated the requirements of NI 43-101. These estimates were supported by five new twinned holes, which demonstrated that the historical assays (on which the earlier resource estimates were based) were comparable to results obtained from the twin holes. The independent resource consultants (Micon) considered the historical estimates to be relevant for the purposes of the study (Lewis, 2007).

An updated conceptual study was completed based on a revised NI 43-101 compliant resource estimate prepared by Micon in April 2008 (Lewis, 2008), which incorporated 38 holes of new drilling as well as historical drilling (see Table 6-1). The resource model used a block size of 10 m (X) x 25 m (Y) x 10 m (Z) and an inverse distance interpolation. The bulk of material included in the conceptual study mine plan was classified as inferred resources.

Table 6-1: Drilling Used in Resource Model for Conceptual Study

	Holes	Metres	% of Holes	% of Metres
Historical Drilling	79	28,322	65	62
Twin Holes	5	1,682	4	4
New Drilling	38	15,606	31	34
Total	122	45,610	100	100

Source: RNC

The conceptual study considered two scopes of open pit design:

- a smaller pit (50 kt/d concentrator) that would extract 452 Mt grading 0.32% Ni. The ultimate pit would be 350 m deep with a stripping ratio of 1.6:1
- a larger pit (75 kt/d concentrator). With the economies of scale from the higher milling rate, the economic pit shell would contain 571 Mt grading 0.32% Ni. The pit would extend to a depth of 470 m and have a stripping ratio of 1.8:1.

Both concepts used a cut-off grade of approximately 0.25% Ni.

In the absence of comprehensive results from metallurgical testwork, the study assumed that the concentrator would achieve a constant recovery of 65%, while sensitivity analysis tested the impact of recovery ranging from 55% to 70%.

The conceptual study concluded that the 75 kt/d option generated more attractive economics and that the project was potentially robust.

6.1.5.2 RNC Preliminary Assessment

Following the positive results of the conceptual study, a Preliminary Assessment was completed in September 2010 entitled, “A Preliminary Assessment of the Dumont property, Launay and Trécesson Townships, Quebec, Canada” (September 2010) (Lewis et al, 2010). The study was managed by RNC, with key external contributors including Golder (resource model), GENIVAR (geotechnical design), BBA (process design) and PasteTec (tailings management). The mine design and process flowsheet were developed in house by RNC, assisted by external consultants. Key changes in the scope of design compared to the scoping study included:

- The quantity of new drilling used to support the resource model was increased by a factor of more than six to 254 holes (totaling 96,701 m). This allowed material to be updated to measured and indicated resources. No inferred resources were included in the scoping study mine plan; this was considered to be waste in the production schedule.
- Whereas the conceptual study resource model included only Ni grade, the scoping study resource model included an interpolation of the three main economic minerals (pentlandite, heazlewoodite and awaruite) along with non-recoverable Ni silicate minerals. This allowed a more granular estimate of recovery, as discussed in a subsequent bullet. The resource model block size was also increased to 20 m (X) x 20 m (Y) x 15 m (Z) to reflect the smallest mining

unit (SMU) for the scale of load and haul equipment that would be used. Use of a larger SMU resulted in a smoother grade estimate and eliminated some of the high-grade zones that the conceptual study assumed could be mined selectively.

- Recovery of Ni to concentrate was estimated uniquely for each block in the resource model, based on the interpolated mineralogy. These estimates were supported by variability testing of 32 bench scale samples representing the different types of mineralization that would be encountered. Metallurgical tests focused on the rougher flotation circuit and estimates of losses during cleaning were based on benchmarks.
- The mining rate for ore was accelerated relative to the requirements of the process plant, leading to the creation of a low-grade stockpile. This stockpile would be treated at the end of mine life after depletion of the open pit. The depleted pit would be used as an impoundment for tailings, reducing the size of the tailings dam by approximately 30%.
- Unlike the conventional SAG mill – ball mill – pebble crusher (SABC) comminution circuit used in the conceptual study, the scoping study assumed a four-stage crushing comminution circuit, based on the process employed in the chrysotile industry. While this flowsheet would be more energy efficient than the SABC circuit, the individual components are considerably smaller and therefore more numerous, which would possibly lead to operational inefficiencies. Additionally, the crushing circuit would require approximately 30% of feed to be dried – at considerable expense and with a potential negative impact on recovery (drying would promote oxidation of sulphide mineralization). Due to these negative impacts, the pre-feasibility study reverted to a conventional SAG mill – ball mill circuit.

This study found that the project is robust (after-tax IRR >> 10%) and that returns will increase non-linearly as the scale of project increases (the 25% increase in mill throughput from 80 to 100 kt/d would result in a 42% increase in after-tax NPV10%). However, the forecast capital (US$2.0 billion for 80 kt/d, increasing to US$2.3 billion for 100 kt/d) was significant, and reflected the complexity of the scoping study flowsheet, as well as the decision to start the project at the full nameplate production rate. The study noted that the key area of risk was forecast deportment of Ni to recoverable minerals and associated estimates of recovery. These items (capital estimate, concentrator flowsheet and recovery estimates) were a key focus of work during the pre-feasibility study.

6.1.5.3 RNC Pre-feasibility Study

Following the positive results of the Preliminary Assessment, Ausenco Solutions Canada Inc. (Ausenco) was commissioned by RNC to complete the pre-feasibility study and the NI 43-101 compliant Technical Report on the project entitled, “Technical Report on the Dumont project, Launay and Trécesson Townships, Quebec, Canada” (16 December 2011) (Ausenco, 2011). SRK Consulting Inc. (SRK) was engaged to prepare the geology, resource estimate, hydrogeology, hydrology and geotechnical inputs and David Penswick, a private mining consultant, was retained for mine design, mine operating costs, mine capital costing and economic modelling. GENIVAR was engaged to provide inputs to the environmental and permitting aspects of the project. Golder Associates Ltd. (Golder) contributed to the environmental geochemistry investigations.

Key changes in the scope of design compared to the Preliminary Assessment included:

- The quantity of new drilling used to support the resource model was increased by an additional 65 holes (totaling 43,261 m). This allowed material to be updated to measured and indicated resources. In addition to nickel, cobalt was reported in the resource estimate.
- The mineralogical database for the deposit was expanded by adding 505 new EXPLOMIN™ QEMSCAN mineralogical samples that were taken throughout the deposit to bring the

number of mineralogical samples from 189 to 694. This expanded database allowed refinement of the mineralogical model and geometallurgical domaining.

- In contrast to the PEA production plan that processed 100,000 kt/d from the beginning of production, the PFS mine, process plant and associated infrastructure were designed to initially process 50 kt/d of ore, with expansion to 100 kt/d in Year 5.
- Site operating costs were reduced by 24% and initial capital outlay was reduced by more than 50% to US$1.1 billion from the 100 kt/d scenario in the PEA. Expansion to 100 kt/d in Year 5 would require US$0.7 billion of additional capital.
- The processing plant would produce a single high-grade concentrate containing an average of 33% nickel over life of project instead of the separate sulphide and alloy concentrate in the PEA.
- Recovery of Ni to concentrate was estimated uniquely for each block in the resource model, based on geometallurgical domaining. In the 2010 Preliminary Assessment, the rougher recovery equations were defined from 32 samples from five drill holes that were available at the time of the evaluation. The samples were grouped by mineralization type (sulphide, alloy and mixed) and structural domain. For the PFS, an additional 38 samples, for a total of 70, were added to the STP suite to update the recovery equations. A review of the expanded mineralogical database for the deposit showed that there were distinct populations of samples, either Pn-rich or Hz-rich with a very small amount that fell in a mixed category between the two extremes. Accordingly, the 70 samples were split into three subgroups: Hz-rich (Hz/Pn > 5), Pn-rich (Hz/Pn < 1) and the mixed sulphide (1<Hz/Pn>5), and recovery equations were developed based on regressions between STP recovery and concentration of select elements as determined by assays. It was decided that mineral abundances not be used as factors in the recovery equations for the PFS, as they had been in the Preliminary Assessment, due to the higher confidence in the deposit assay model compared with the deposit mineralogical model.
- All metal price assumptions are the same as the figures used for the PEA with the exception of nickel price which was increased to $9.00 per pound.

This study found that the project is robust yielding US$1.1 billion after-tax NPV8%, after-tax IRR of 17% and C1 cash costs of US$4.13 per pound of nickel. Average annual contained nickel production of 96 million pounds (44 kt) during the 19-year mine life and 59 million pounds (27 kt) for the subsequent 12 years from processing of the lower grade stockpile. Additional potential upsides including production of a final ferronickel product, production of iron ore (magnetite) concentrate byproduct, additional recovery optimization and use of in-pit crushing or trolley system were identified for further study in the PFS.

6.1.5.4 RNC Revised Pre-feasibility Study

Following the positive results of the pre-feasibility study, Ausenco was commissioned by RNC to produce a revised pre-feasibility study and NI 43-101 compliant technical report for the Dumont project. SRK was engaged to prepare the geology, resource estimate, hydrogeology, hydrology and geotechnical inputs, and David Penswick, a private mining consultant, was retained for mine design, mine operating costs, mine capital costing and economic modelling. GENIVAR was engaged to provide inputs to the environmental and permitting aspects of the project. Golder contributed to the environmental geochemistry investigations.

Key changes in the scope of design compared to the pre-feasibility study included:

- The quantity of new drilling used to support the resource model was increased by an additional 50,000 m. This allowed material to be updated to measured and indicated resources. In addition to nickel, cobalt, platinum and palladium were reported in the resource estimate.

- The mineralogical database for the deposit was expanded by adding 403 new EXPLOMIN^{TM} QEMSCAN mineralogical samples that were taken throughout the deposit to bring the number of mineralogical samples from 694 to 1,097. This expanded database allowed refinement of the mineralogical model and geometallurgical domaining. This allowed estimation of the magnetite content for a portion of the deposit.
- Project recoveries were improved to 45% in the revised PFS from 41% in the PFS due to the combination of significant additional metallurgical testwork, a 50% increase in mineralogy samples and the revised resource model. Recoveries are 57% in Years 1 to 5 of the mine life; 51% in Years 6 to 19; and 33% in Years 20 to 31. This improvement contributed an additional US$296 M to the project NPV8%. The revised metallurgical ore classification was further refined into five separate domains rather than the four used in the initial PFS. Cobalt recovery is estimated to be an average of 45% over the life of the project, a decrease from 70% in the PFS, as the deportment of cobalt between the recoverable minerals and silicates is similar to nickel. Platinum and palladium payable metals were not included in the revised PFS, as their ability to upgrade above a minimum payable level in concentrate is uncertain due to limited technical resource and recovery work on PGEs.
- The average concentrate grade was reduced to 29% as additional mineralogy work revealed that the nickel content of the pentlandite in certain areas of the orebody contained 27% nickel rather than the 33% nickel found throughout the majority of the orebody.
- A mining scenario including the use of trolley assist to improve overall mining costs for the project by using electricity to replace a portion of the diesel fuel consumed by trucks was evaluated. The implementation of trolley assist during expansion in Year 5 and other improvements reduced mining costs by US$0.14 per tonne mined (US$0.32 per tonne ore) and reduced estimated diesel consumption by 28% to 872 ML over the life of the project.
- All metal price assumptions are the same as the figures used for the pre-feasibility study.

The revised PFS (base case plus trolley assist option) yielded an increase of project after-tax NPV8% of 31% from US$1.1 billion to US$1.4 billion with an after-tax IRR of 19.5% and net C1 cash costs of US$4.07 per pound of nickel. Average annual contained nickel production of 108 Mlbs (49 kt) during the 19-year mine life and 63 Mlbs (29 kt) for the subsequent 12 years from processing of the lower grade stockpile. Additional potential upsides including production of a final ferronickel product, production of iron ore (magnetite) concentrate byproduct, additional recovery optimization and optimization of the trolley system configuration were identified for further study in the feasibility study.

6.2 Historical & Mining Production

No historical mining or production has been conducted on the Dumont property. However, the Val d'Or-Rouyn-Noranda region surrounding the Dumont property has been a prolific mining area for the past 100 years.

6.3 Dumont Property Resource & Reserve Estimates

The discussions related to the resource and reserve estimates contained in this section refer to historical estimates and subsequent RNC resource estimates. The historical estimates may have been prepared according to the accepted standards for the mining industry for the period to which they refer; however, they do not comply with the current CIM standards and definitions for estimating resources and reserves as required by NI 43-101 guidelines. A qualified person has not done sufficient work to classify the historical estimates as a current resource estimate and the issuer is not treating the historical estimates as a current resource estimate. As a result, historical estimates should not be relied upon unless they have been validated and restated to comply with the latest CIM standards and definitions.

6.3.1 1971 to 1986 Resource Estimation

A summation report (Honsberger, 1971) stated the potential resources for the deposit and the reserves for the No. 1 deposit using a 0.50% nickel cut-off grade. This estimate was part of the earlier CDS feasibility study for an underground mine that was planned to produce 4,500 tons per day. The potential of the Dumont property was determined from drilling results obtained between sections 36+00W and 84+00W where higher grade bands were intersected on drill sections 800 ft apart and mineralized intersections grading 0.5% nickel or higher were obtained.

Using these intersections and those for the No. 1 orebody, both Honsberger and Caron reported that the estimated potential of the higher grade bands was 70 Mt of material grading 0.5% nickel and higher, down to a depth of 2,000 ft.

The estimation of the reserves for the 1971/1972 feasibility study was completed using the sectional estimation method where the drill holes were plotted on sectional views; the area of influence of each drill hole intersection was measured on the section; and the necessary corrections were made for the dip and strike of the deposit to measure the area in the plane perpendicular to the strike of the zone. The volume of influence of the drill core intersection was obtained by multiplying its area of influence by half the distance measured along strike between two adjacent sections. A volume factor of 12 ft^2/ton was used to convert the volumes of influence into tonnages. The tonnage of the reserves was estimated by adding the tonnages from all the holes while the grade was determined by using the weighted average of the grades for each tonnage block. In depth, the tonnage was estimated from elevation 250 to 1,500 ft.

To account for dilution, an underground mining scenario was selected for the August 1971 report. It was determined that 6% was appropriate due to the competence of the rock and the continuity of the mineralization. The average nickel content of the mineralization located within the hanging wall and within 5 ft of the zone was estimated at 0.45% nickel. Since most of the dilution was expected to come from the hanging wall, this grade was determined to be the grade of the diluting material.

The tonnage and grade of the reserves above the 900 level were estimated separately using the same method. After dilution, the tonnage was 6,906,609 at an average grade of 0.660% nickel.

There is mention of a second historical resource or reserve estimate that was conducted by Timiskaming in 1974-1975. Timiskaming and Boliden AB concluded positively that the project had economic potential for a 13,600 t/d open pit mining operation on the estimated 320 Mt of resources at 0.34% nickel, from which the patented segregation process would recover 75% of the nickel. The authors of this report were unable to obtain any data regarding this estimate and it has therefore been excluded from the current discussion.

A third historical estimate (Duke, 1986) of the resource potential of the mineral deposit was conducted. Table 6-2 summarizes the resource potential in the 1986 estimate.

Table 6-2: Historical 1986 Potential Resource Estimate for the Three Nickel-Enriched Layers

Layer	Strike Length (m)	Average Thickness (m)	Average Grade (% Nickel)	Tonnage (Mt)
Upper	2,430	24	0.45	80
Middle	2,430	24	0.50	82
Lower	350	26	0.44	13
Total of the Layers			0.47	175
High-grade Middle Layer Resource	730	14	0.65	14

Source: After Duke (1986)

6.3.2 2008 Mineral Resource Estimation (RNC)

The historical 1971 reserve estimate was superseded by RNC's 2008 preliminary mineral resource estimate, the details of which are contained in a Technical Report entitled "NI 43-101 Technical Report, Preliminary Mineral Resource Estimate for the Dumont Property, Launay and Trécesson Townships, Quebec, Canada" (April 2008) (Lewis, 2008).

The April 2008 preliminary resource estimate was based on the results of both the 2007 exploration drilling and the historical drilling. The tonnages and grades for the April 2008 indicated and inferred mineral resource estimates are summarized Table 6-3.

Table 6-3: April 2008 Indicated & Inferred Mineral Resources at a Cut-Off of 0.35% Ni

Mineral Resource Category	Tonnage (kt)	Nickel Grade (%)	Nickel (kt)	Nickel (klbs)
Indicated	50,076	0.353	177	390,012
Inferred	693,013	0.308	2,133	4,704,118

Note: * The inferred mineral resource contained in this table represents the combination of the current and historical models. **Source:** RNC.

The April 2008 preliminary mineral resource estimate was compliant with the current CIM standards and definitions required by NI 43-101 regulations and was reportable as a mineral resource by RNC.

The April 2008 preliminary Mineral Resource estimate was superseded by an updated mineral resource estimate effectively dated 31 October 2008. The details on this mineral resource estimate are contained in a Technical Report entitled, "NI 43-101 Technical Report, Updated Mineral Resource Estimate for the Dumont Property, Launay and Trécesson Townships, Quebec, Canada" (January 2009).

The October 2008 resource estimate was based on the drilling conducted in 2007 and 2008 by RNC; use of the historical information was limited to the peripheral areas of the deposit or at depth where RNC had not conducted any drilling. The tonnages and grades for the October 2008 indicated and inferred mineral resource estimates are summarized Table 6-4.

Table 6-4: Indicated & Inferred Mineral Resource at a Cut-off of 0.25% Ni (31 October 2008)

Area Within Deposit Model	Mineral Resource Category	Tonnage (kt)	Nickel Grade (%)	Nickel (kt)	Nickel (klbs)
Central Portion	Indicated	365,024	0.320	1,168	2,575,025
6000 – 9400 Portion	Inferred*	257,718	0.306	790	1,740,888
NW Portion		146,041	0.268	391	861,450
SE Portion		29,660	0.275	82	180,056
Historical Solid		65,931	0.324	214	471,313
Total Deposit		499,350	0.296	1,476	3,253,707

Note: ***The inferred mineral resource contained in this table represents the combination of the current and historical solids. **Source:** RNC

The mineral resource estimate as of 31 October 2008 was compliant with the current CIM standards and definitions required by NI 43-101 and was reportable as a mineral resource by RNC.

6.3.3 2010 Mineral Resource Estimation (RNC)

The 31 October 2008 Mineral Resource estimate contained in the January 2009 Technical Report was then superseded by an updated mineral resource contained in the 2010 Technical Report entitled “NI 43-101 Technical Report, Mineral Resource Estimate for the Dumont Property, Launay and Trécesson Townships, Quebec, Canada” (April 2010)(Lewis, 2010).

The resource estimate contained in the April 2010 Technical Report was based on the drilling conducted from 2007 to 2009 by RNC and on the geological structural information developed by Itasca Consulting. The introduction of the structural model resulted in the separation of the Dumont deposit into seven separate domains, rather than two. The seven solid models did not overlap each other in space. However, all solid models were contiguous and were constrained using a 0.2% nickel cut-off grade. Constructing the seven solids was a result of the available structural model and the confidence level in the data set.

The overburden surface was constructed using the drill hole data. No lithological solid model was generated and used for the resource estimate since the mineralization is hosted primarily within the dunite unit. No historical drill holes were used for the mineral resource estimate contained in the April 2010 Technical Report.

Along the strike direction, the resource model extends between sections 3600E and 10400E. Due to the differing strike directions of the seven domains, the total length is 7,035 m. The vertical boundaries are defined using the overburden and rock interface as the upper boundary, while the lower boundary is defined by using a variable projected distance of approximately 50 m below the deepest drilling assays above the cut-off grade. The hanging wall and footwall boundaries are projected in the down dip direction (average of -58°) as defined by the actual assays above the cut-off criterion.

The effective date of the mineral resource estimate in the April 2010 Technical Report was 4 December 2009. Table 6-5 summarizes this resource estimate.

Table 6-5: Measured, Indicated & Inferred Mineral Resource in the Seven Domain Solids at a Cut-off of 0.25% Ni (4 December 2009)

Area Within Deposit Model	Mineral Resource Category	Tonnage (kt)	Nickel Grade (%)	Nickel (kt)	Nickel (klbs)
All Domains	Measured (M)	73,935	0.33	246	543,257
All Domains	Indicated (I)	576,745	0.31	1,800	3,966,328
All Domains	Total M + I	650,680	0.31	2,046	4,509,585
All Domains	Inferred	257,804	0.28	709	1,563,865

Source: RNC

The mineral resource estimate as of the effective date of 4 December 2009 was compliant with the current CIM standards and definitions required by NI 43-101 and was reportable as a mineral resource by RNC.

The December 2009 Mineral Resource estimate contained in the April 2010 Technical Report was then superseded by an updated mineral resource contained in the Technical Report entitled “NI 43-101 Technical Report, Mineral Resource Estimate for the Dumont Property, Launay and Trécesson Townships, Quebec, Canada” (August 2010).

The resource estimate contained in the August 2010 Technical Report was based on the drilling conducted from 2007 to 2010 by RNC and on the geological structural information developed by Itasca Consulting. Micon estimated the updated mineral resource based on the geological information and assaying data for the Dumont property available as of 22 April 2010. The effective date of the resource estimate was 16 August 2010.

For the August 2010 Technical Report, it was possible to refine the estimated cut-off grade to 0.20% nickel based on work from the concurrent September 2010 preliminary assessment.

Recognizing that the amount of nickel in recoverable minerals is of paramount importance to mine planning and plant design, RNC retained Golder to prepare a resource block model that would incorporate nickel grade and major mineralogical abundances. The resource block model work was completed by Olivier Tavchandjian, P.Geo, and was reviewed by Greg Greenough, P.Geo, both of Golder (Warren, 2010; Golder Associates, 2010).

The August 2010 resource block model interpolated nickel, copper, cobalt, chromium, platinum, palladium and gold grades, specific gravity, and ten factor scores used to calculate the mineral abundances of pentlandite, heazlewoodite, awaruite, olivine, magnetite, serpentine, brucite and coalingite.

Golder and RNC conducted all of the 3D modelling work. Micon verified and audited the mineralization envelopes.

RNC provided to Micon the 3D modelling work of the mineralization envelopes based on the geometallurgical model provided by Golder and a 0.2% nickel cut-off grade. Micon reviewed the block model extensively and in some cases the model was refined in discussions with RNC.

The overburden surface was constructed using the drill hole data. No lithological solid model was generated, since the mineralization considered in the resource is hosted entirely within the dunite unit.

Based on all of the data currently available, seven separate solid models were generated. The seven solid models do not overlap each other in space, but all are contiguous and have been constrained using a 0.2% nickel cut-off grade. The seven solids were constructed on the basis of the available structural model and the confidence level in the data set.

Along the strike direction, the current resource model extends between sections 3600E and 10000E. Due to the differing strike directions of the seven domains, the total length is 7,035 m. The vertical boundaries are defined using the overburden and rock interface as the upper boundary, while the lower boundary is defined by using a variable projected distance of approximately 50 m below the deepest drilling assays above the cut-off grade. The hanging wall and footwall boundaries are projected in the down dip direction (average of -58°) as defined by the actual assays above the cut-off criterion.

Micon reviewed and audited the updated mineral resource estimate for RNC which is CIM compliant. The tonnages and grades for the August 2010 mineral resource estimate are summarized Table 6-6.

The mineral resource estimate as of the effective date of 16 August 2010 was compliant with the current CIM standards and definitions required by NI 43-101 and is reportable as a mineral resource by RNC.

Table 6-6: Summary of the Measured, Indicated & Inferred Mineral Resource in the Seven Structural Domain Solids at a Cut-off of 0.20% Nickel (16 August 2010)

Area Within Deposit Model	Mineral Resource Category	Tonnage (kt)	Nickel Grade (%)	Nickel (kt)	Nickel (klbs)
All Domains	Measured (M)	155,680	0.29	447	985,365
All Domains	Indicated (I)	1,003,487	0.27	2,707	5,966,826
All Domains	Total M + I	1,159,167	0.27	3,154	6,952,191
All Domains	Inferred	581,405	0.27	1,451	3,198,220

Source: RNC

6.3.4 2011 Mineral Resource Estimation & Mineral Reserve (RNC)

The 2010 Mineral Resource Estimate contained in the August 2010 Technical Report was superseded by an updated mineral resource effective 13 December 2011 (Ausenco, 2011).

The 13 December 2011 Mineral Resource Estimate for the Dumont project presented in Table 6-7 was prepared by Mr. Sébastien Bernier, P.Geo, at SRK. The mineral resource estimate considers drilling information available to 3 October 2011 and was evaluated using a geostatistical block modelling approach constrained by seven sulphide mineralization wireframes. The mineral resources have been estimated in conformity with the CIM "Mineral Resource and Mineral Reserves Estimation Best Practices" guidelines and were classified according to the CIM Standard Definition for Mineral Resources and Mineral Reserves (December 2005) guidelines.

In addition to nickel and cobalt, SRK modelled the abundance distribution of seven other main elements: arsenic, gold, calcium, chromium, copper, iron, lead, palladium, platinum and sulphur.

Table 6-7: Mineral Resource Statement* (SRK, 13 December 2011)

Resource Category	Quantity (kt)	Grade Ni (%)	Grade Co (ppm)	Contained Nickel (kt)	(M lbs)	Contained Cobalt (kt)	(M lbs)
Measured	189,770	0.29	111	550	1,203	20	46
Indicated	1,220,300	0.27	108	3,270	7,216	130	290
Measured + Indicated	**1,410,070**	**0.27**	**109**	**3,820**	**8,419**	**150**	**336**
Inferred	695,200	0.26	100	1,790	3,939	70	154

Note: *Reported at a cut-off grade of 0.2% Ni inside conceptual pit shells optimized using nickel price of US$9.00/lb, average metallurgical and process recovery of 41%, processing and G&A costs of US$5.40/t milled, exchange rate of CAD$1.00 = US$0.90, overall pit slope of 40° to 44° depending on the sector and a production rate of 100 kt/d. All figures rounded to reflect the relative accuracy of the estimates. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

To facilitate RNC's ongoing evaluation of metallurgical recovery, SRK also constructed estimation models of mineral abundances. Specifically, SRK modelled the abundance distribution of pentlandite, heazlewoodite, awaruite, olivine, magnetite, serpentine, brucite, coalingite, and iron-rich serpentine. Although these mineral abundances do not directly impact the mineral resource at the Dumont project, they do affect the metallurgical recovery, which has a direct impact on the feasibility of this project.

Reserves were estimated by David Penswick, P. Eng, an independent consultant, based on the mineral resource block model described above and the results of the pre-feasibility study. Reserves are based on a Lerchs-Grossmann optimized pit shell generated using only nickel values and a nickel price of US$6.70/lb, which is 74% of the long-term forecast of US$9.00/lb and include planned and unplanned dilution of 4.2% and 0.65%, respectively. The 13 December 2011 Dumont mineral reserves are summarized in Table 6-8.

Table 6-8: Mineral Reserves Summary* (David Penswick, 13 December 2011)

Resource Category	Reserves (kt)	Grade Ni (%)	Grade Co (ppm)	Contained Nickel (kt)	(M lbs)	Contained Cobalt (kt)	(M lbs)
Proven	0	0.00	0	0	0	0	0
Probable	1,069,700	0.27	108	2,876	6,340	116	255
Total Proven & Probable	**1,069,700**	**0.27**	**108**	**2,876**	**6,340**	**116**	**255**

Note: Reported at a cut-off grade of 0.2% nickel inside an engineered pit design. This design was based on a Lerchs-Grossmann optimized pit shell using nickel price of $6.70 per pound, average metallurgical and process recovery of 41%, processing and G&A costs of $6.30 per tonne milled, exchange rate of CAD$1.00 = US$0.90, overall pit slope of 40° to 44° depending on the sector and a production rate of 50 kt/d. All figures rounded to reflect the relative accuracy of the estimates. Mineral reserves are based on a smallest mining unit of 6,000 m^3 and include allowances of 0.65% for unplanned dilution and 0.80% for mining losses.

Since the December 13, 2011 mineral resource estimate and reserve was published, RNC has performed additional drilling and mineralogical sampling. Because of this work, RNC was able update its resource estimate. RNC's updated resource model as estimated by SRK is discussed in Section 14 of this Technical Report.

6.3.5 2012 Mineral Resource Estimation & Mineral Reserve (RNC)

The 13 December 2011 Mineral Resource estimate contained in the December 2011 Technical Report was superseded by an updated mineral resource effective 13 April 2012 (Ausenco, 2012).

The 13 April 2012 Mineral Resource Estimate for the Dumont project presented in Table 6-9 was prepared by Mr. Sébastien Bernier, P.Geo, at SRK. The mineral resource estimate considers drilling information available to 1 February 2012 and was evaluated using a geostatistical block modelling approach constrained by seven sulphide mineralization wireframes. The mineral resources have been estimated in conformity with the CIM "Mineral Resource and Mineral Reserves Estimation Best Practices" guidelines and were classified according to the CIM Standard Definition for Mineral Resources and Mineral Reserves (December 2005) guidelines.

The Mineral Resource Statement included the first disclosure of palladium and platinum grade and magnetite concentration.

In addition to nickel, palladium, platinum and cobalt, SRK modelled the abundance distribution of four other main elements: calcium, chromium, iron and sulphur.

To facilitate RNC's ongoing evaluation of metallurgical recovery, SRK constructed estimation models of mineral abundances. Specifically, SRK modelled the abundance distribution of awaruite, coalingite, heazlewoodite, serpentine, low-iron serpentine, iron-rich serpentine, magnetite, olivine, and pentlandite.

Reserves were estimated by David Penswick, P. Eng, an independent consultant, based on the mineral resource block model described above. Reserves are based on a Lerchs-Grossmann optimized pit shell generated using only nickel values and a nickel price of US$6.70/lb, which is 74% of the long-term forecast of US$9.00/lb and include planned and unplanned dilution of 4.2% and 0.65%, respectively.

The 14 May 2012 Dumont mineral reserves are summarized in Table 6-10. Since the 13 April 2012 mineral resource estimate and 14 May 2012 reserve were published, RNC has performed additional drilling and mineralogical sampling. Because of this work, RNC was able to update its resource estimate. RNC's updated resource model, as estimated by SRK, is discussed in Section 14 of this technical report.

Table 6-9: Mineral Resource Statement* (SRK, 13 April 2012)

Resource Category	Quantity (kt)	Grade Ni (%)	Grade Co (ppm)	Contained Nickel (kt)	Contained Nickel (Mlbs)	Contained Cobalt (kt)	Contained Cobalt (Mlbs)
Measured	359,440	0.29	112	1030	2,260	40	89
Indicated	1,261,630	0.26	106	3,330	7,336	130	295
Measured + Indicated	**1,621,070**	**0.27**	**109**	**4,360**	**9,596**	**170**	**384**
Inferred	513,080	0.26	100	1,320	2,904	50	113
Resource Category	**Quantity (kt)**	**Grade Pd (g/t)**	**Grade Pt (g/t)**	**Contained Palladium (oz)**		**Contained Platinum (oz)**	
Measured							
Indicated	182,860	0.036	0.018	211,000		107,000	
Measured + Indicated	**182,860**	**0.036**	**0.018**	**211,000**		**107,000**	
Inferred	256,530	0.030	0.016	243,000		135,000	
Resource Category	**Quantity (kt)**	**Grade Magnetite (%)**		**Contained Magnetite (kt)**	**Contained Magnetite (Mlbs)**		
Measured							
Indicated	579,620	3.87		22,450	49,500		
Measured + Indicated	**579,620**	**3.87**		**22,450**	**49,500**		
Inferred	1,301,540	4.13		53,760	118,515		

Note: * Reported at a cut-off grade of 0.2% nickel inside conceptual pit shells optimized using nickel price of US$9.00 per pound, average metallurgical and process recovery of 41%, processing and G&A costs of US$5.40 per tonne milled, exchange rate of CAD$1.00 equal US$0.90, overall pit slope of 40° to 44° depending on the sector, and a production rate of 100 kt/d. Values of palladium, platinum and magnetite are not considered in the cut-off grade calculation as they are byproducts of recovered nickel. All figures are rounded to reflect the relative accuracy of the estimates. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

Table 6-10: Mineral Reserves Summary* (David Penswick, 14 May 2012)

Reserve Category	Reserve (kt)	Grade Ni (%)	Grade Co ppm	Contained Nickel (kt)	Contained Nickel (Mlbs)	Contained Cobalt (kt)	Contained Cobalt (Mlbs)
Proven	0	0.00	0	0	0	0	0
Probable	1,066,200	0.27	107	2,876	6,340	114	252
Total Proven & Probable	**1,066,200**	**0.27**	**107**	**2,876**	**6,340**	**114**	**252**

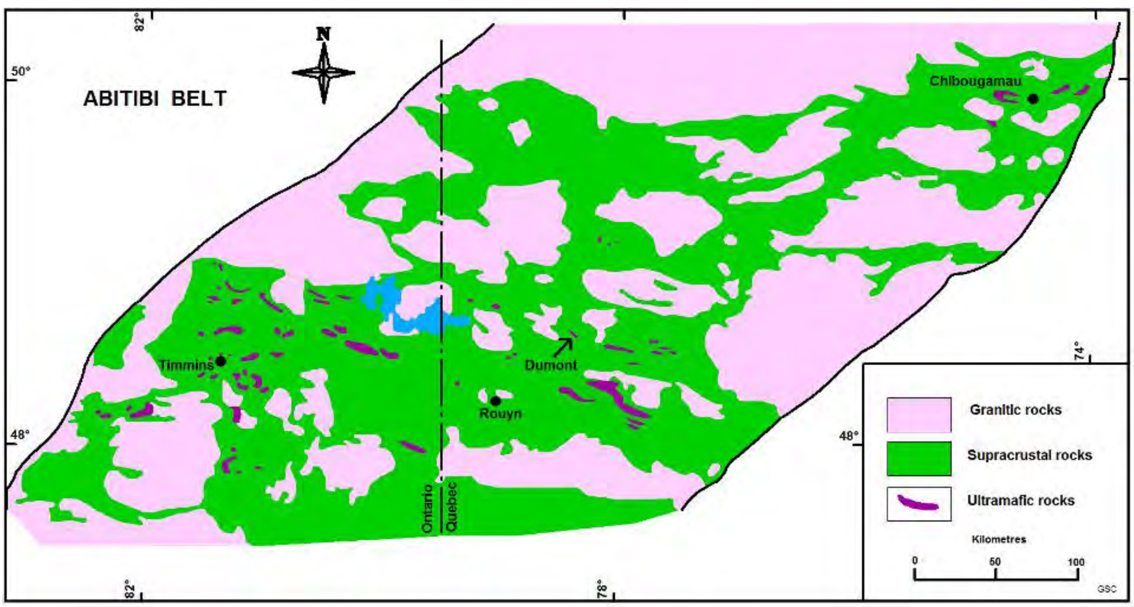
Note: Reported at a cut-off grade of 0.2% nickel inside an engineered pit design. This design was based on a Lerchs-Grossmann optimized pit shell using nickel price of US$6.70 per pound, average metallurgical and process recovery of 41%, processing and G&A costs of US$6.30 per tonne milled, exchange rate of CAD$1.00 = US$0.90, overall pit slope of 40°to 44° depending on the sector and a production rate of 50 kt/d. All figures rounded to reflect the relative accuracy of the estimates. Mineral reserves are based on a smallest mining unit of 6000 m^3 and include allowances of 0.65% for unplanned dilution and 0.80% for mining losses.
Source: David Penswick.

7 GEOLOGICAL SETTING

7.1 Regional Geology

A thick supracrustal succession of Archean volcanic and sedimentary rocks underlies about 65% of the Abitibi belt, and there is evidence to suggest that these supracrustal rocks lie unconformably upon a basement complex of sialic composition. The volcanic rocks are mainly of mafic composition although ultramafic, intermediate and felsic types are also present. The abundance of pillowed and nonvesicular lavas, together with the flyschoid character of much of the sedimentary component, demonstrates the prevalence of deep submarine conditions. However, the occurrence of some fluvial sedimentary rocks and airfall tuffs attest to occasional local non-marine conditions. Numerous small to medium sized synvolcanic intrusions reflect the range of compositions of the lavas themselves. See Figure 7.1 for a map reflecting the location of the Dumont ultramafic sill within the Abitibi Greenstone Belt.

Figure 7.1: Location of the Dumont Ultramafic Sill within the Abitibi Greenstone Belt



Source: Supplied by RNC after Duke (1986).

The supracrustal rocks were deformed and intruded by granitic stocks and batholiths during the Kenoran event about 2,680 to 2,700 million years (Ma) ago. Folding along generally east-trending axes has commonly produced isoclinal structures. Regional metamorphism is predominantly greenschist and prehnite-pumpellyite facies except in the contact aureoles of the Kenoran granites where amphibolite grade is usually attained. The amphibolite facies metamorphism also occurs in the sedimentary rocks of the Pontiac Group. Two main sets of diabase dykes occur in the Abitibi belt; the north-trending Matachewan swarm and northeast-trending Abitibi swarm which have Rb-Sr ages of 2,690 and 2,147 Ma, respectively. The latter are prominent near the Dumont intrusion, although none is known to have cut the body.

The Dumont sill is hosted by lavas and volcaniclastic rocks assigned to the Amos Group. The lavas may be traced eastwards through the town of Amos and are part of the Barraute volcanic complex. Three cycles of mafic to felsic volcanism are recognized and the Dumont sill is one of at least five ultramafic-mafic complexes in the Amos area, which occur at approximately the same stratigraphic level within the mafic lavas of the middle cycle. The host rocks of the sill are for the most part iron-rich tholeiitic basaltic lavas although some intermediate rocks are known to occur at the body at its eastern end of the sill.

Although the volcanic rocks have been folded and now dip steeply, a penetrative deformational fabric is only locally developed. In the vicinity of the Dumont sill, pillows in the lavas are not strongly deformed and primary textures such as “swallow-tail” plagioclase microlites are preserved. However, the chemical compositions of many of the rocks are highly altered with many rocks containing significant levels of CO_2. Three main directions of faulting are recognized in the Amos area with the earliest being the east-trending set of “bedding plane” faults which are believed to have developed during the major period of folding. The second set of faults occurred during the intrusion of the granitic rocks, which was accompanied by the development of steeply dipping faults that strike north to northwest. However, the most prominent faults strike northeast and probably postdate the granitic plutonism with the Dumont sill cut by a number of these northeast, northwest and east-trending faults.

7.2 Project Area Geology

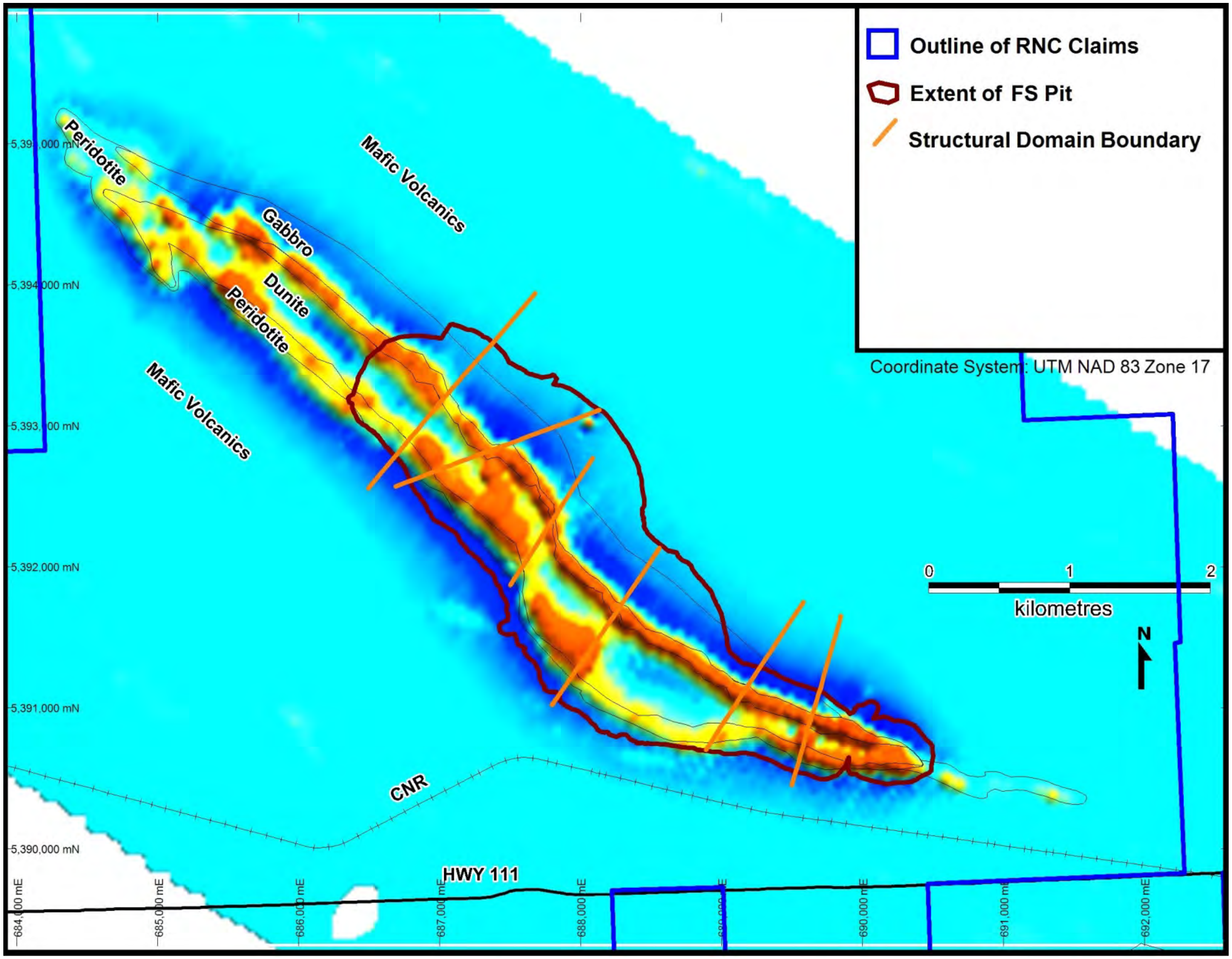
The property is covered by a layer of glacial overburden and muskeg. Mineralization subcrops approximately 30 m below the surface. Contacts between the Dumont sill and its host rocks have not been observed in outcrop but, in overall attitude, the body appears to be conformable to the layering of the volcanic rocks. This is consistent with the interpretation of the Dumont ultramafic body as a sill by Duke (1986), but is also consistent with alternate interpretations for conformable ultramafic bodies that occur in ophiolitic associations. Pillowed basalts exposed at the eastern end of the sill clearly indicate a northeast facing direction.

Offsets in the magnetic contours and internal stratigraphy of the ultramafic zone along with oriented drill hole data have provided evidence for a number of faults at a high angle to the long axis of the sill consistent with the northeast, northwest and east-trending regional faults. Structural logging has also identified several faults parallel to the strike of the intrusion. Based on other offsets in mineralization and alteration, there are undoubtedly other faults which have not yet been recognized (Figure 7.2).

The sill, considered to be a layered mafic-ultramafic intrusion (Duke, 1986) is comprised of a lower ultramafic zone and an upper mafic zone. Although less than 2% of the bedrock surface of the intrusion is exposed in outcrop, the boundaries of the ultramafic zone can be drawn with some confidence based on a magnetometer survey (Figure 7.2) and diamond drilling (Figure 7.3).

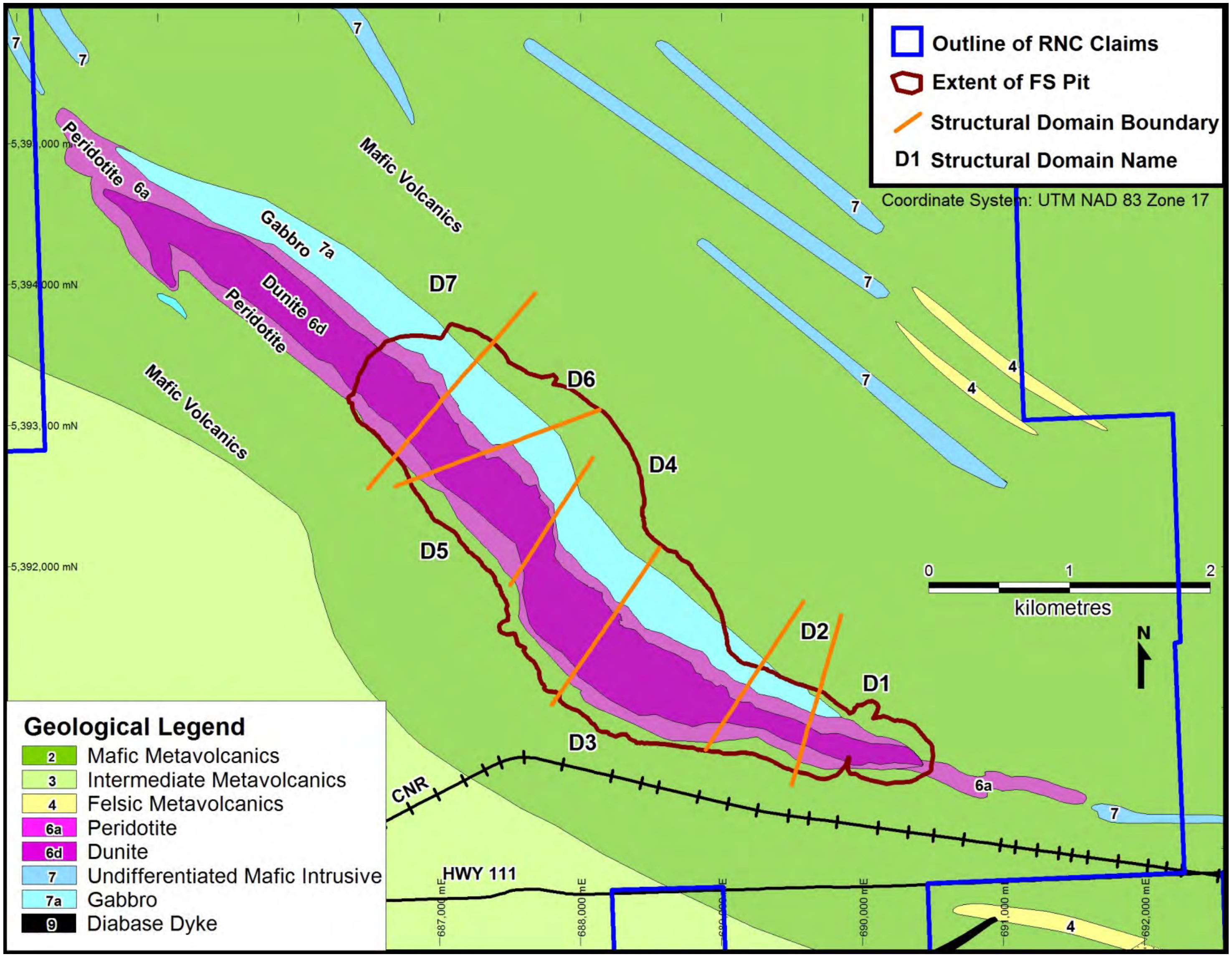
Based on the identified prominent northwest (NW) and northeast (NE) trending faults, the sill can be divided into structural blocks/domains. The true thickness of the upper mafic and lower ultramafic zone varies by location or fault block though the sill. The north-western end of the body has not been outlined precisely; however, the ultramafic zone is a lenticular mass at least 6,600 m in length with an average true thickness of 450 m, with a maximum of 600 m in the central region to a minimum of 150 m in the extreme southeast. The true dip of the ultramafic zone also varies with location in the sill from 60° to 70°. The extent of the mafic zone is much less well defined due to the low density of drill hole data intersecting this zone and its contact with the host rock. An estimated thickness of 200 m is given to this unit based on limited drill hole data and outcrop locations. No feeder to the Dumont sill has been observed to date.

Figure 7.2: Map of Magnetometer Survey of the Dumont Property (1st Vertical Derivative)



Source: RNC.

Figure 7.3: Geological Map of the Dumont Property



Source: RNC.

The ultramafic zone is subdivided into the lower peridotite, dunite and upper peridotite subzones. The lower and upper peridotite subzones are olivine-chromite cumulates with variable amounts of intercumulus clinopyroxene. The dunite subzone is an extreme olivine adcumulate containing very small amounts of intercumulus chromite and clinopyroxene. Cumulus sulphide occurs in certain parts of the dunite subzone and also locally in the lower peridotite. The mafic zone is comprised of three subzones which are from the base upwards, the clinopyroxenite, the gabbro and the quartz gabbro. The clinopyroxenite subzone is an extreme clinopyroxene adcumulate at its base grading into clinopyroxene + plagioclase cumulate rocks in the overlying gabbro subzone. The quartz gabbro subzone includes both plagioclase + clinopyroxene cumulates and noncumulate gabbros that contain modal and normative quartz. Olivine and chromite are restricted to the ultramafic zone, and plagioclase occurs only in the mafic zone.

7.2.1 Primary Sill Features

The magnesium to magnesium plus iron ratios (Mg/Mg+Fe) of the ferromagnesian cumulus phases corresponds to the overall whole rock assay (Duke 1986). Whole rock assays show an increase gradually from the base of the sill upwards across the lower peridotite, and undergo an abrupt increase at or just above the base of the dunite. The magnesium to iron ratio through the dunite, remains essentially constant, however the stratigraphically lower dunite contains more iron than the stratigraphically upper dunite. At the upper dunite limit where it approaches the upper peridotite, there is a decrease in the Mg/Fe ratio, followed by iron enrichment upwards through the overlying part of the intrusion.

Chromium content is lowest in the centre of the dunite sub layer and increases toward both the upper and lower margins of the dunite and into both the upper and lower peridotite. The increase in chromium corresponds to an increase in chromite. The increase in chromite towards the base of the lower dunite corresponds with the increase in iron of the lower dunite subzone.

Magmatic sulphides are restricted to the lower peridotite and dunite subzones, in the latter they are strongly affiliated with the magnesium-rich upper dunite. Sulphides present in the lower peridotite represent a post-cumulus phase. Four olivine-sulphide cumulate layers occur locally within the dunite subzone but do not extend over the entire strike length of the sill.

Two types of mineralization have been identified historically within the Dumont sill, the primary, large low-grade to medium-grade disseminated nickel deposit (Duke, 1986) and the contact type nickel-copper-platinum group elements (PGE) occurrence discovered in 1987 (Oswald, 1987). Drilling by RNC has also identified discontinuous PGE mineralization associated with disseminated sulphides at lithological contacts in the layered intrusion and within the dunite.

7.2.2 Secondary Sill Features

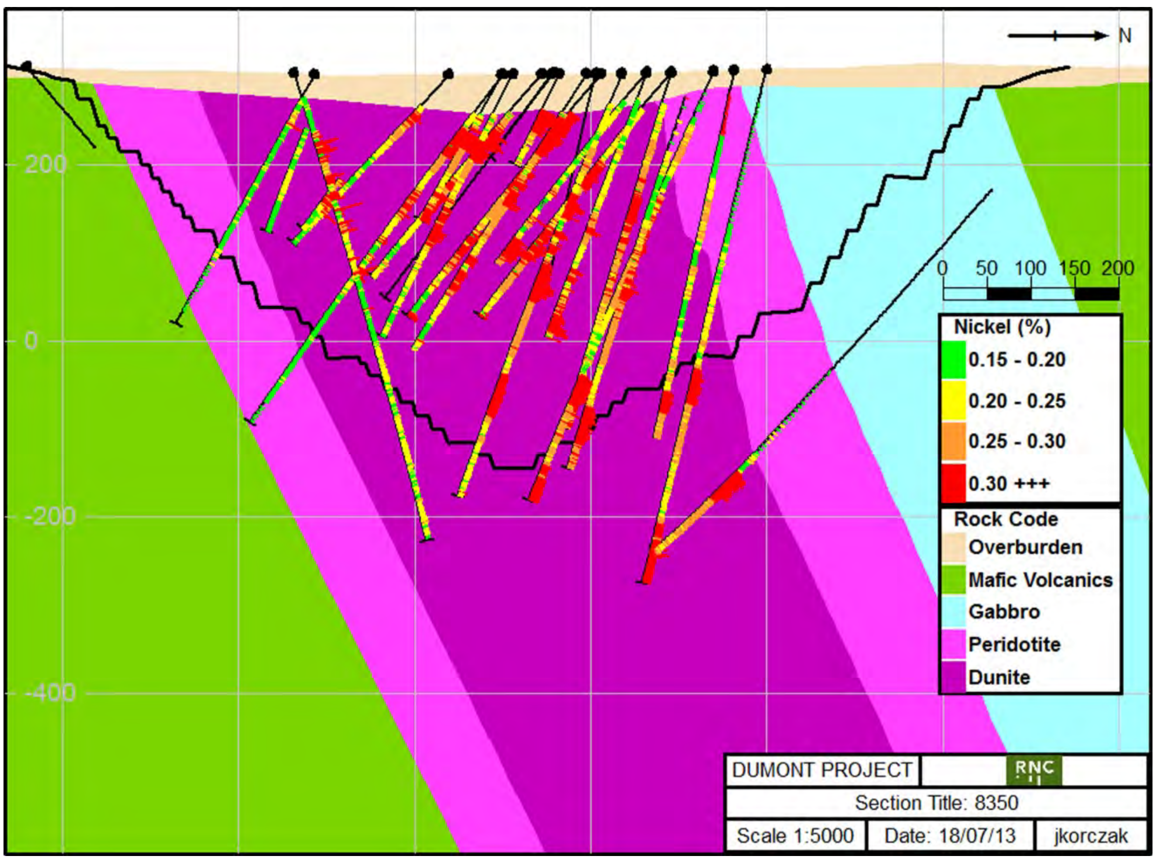
The ultramafic rocks have been serpentinized to varying degrees from partial to complete serpentinization. Along the basal contact of the sill (outside the resource envelope) serpentinization is frequently overprinted by varying degrees of talc-carbonate alteration. The predominant secondary assemblage is lizardite + magnetite + brucite + chlorite + diopside ± chrysotile ± pentlandite ± awaruite ± heazlewoodite. Antigorite is developed locally, particularly in the uppermost ultramafic zone. Native copper occurs in and along major fault systems and alongside intercumulus nickel sulphide and awaruite mineralization, more frequently this has been observed in zones that are partially serpentinized. Trace millerite can occur in the steatitized rocks of the basal contact zone and more rarely in large fault zones. The mafic zone is ubiquitously altered to the assemblage actinolite + epidote + chlorite ± quartz. Primary textures are pseudomorphously preserved throughout most of the intrusion.

Serpentinization proceeded isovolumetrically on the microscopic scale. On the microscopic scale, serpentinization was isochemical. However, on the whole, as the major elements are re-partitioned into new phases during the process, with the addition of hydrogen, oxygen (water) and chlorine to the system, some phases can be dissolved and transported. The extent of this process is not well described in literature; however, within the Dumont sill, RNC has observed some evidence (areas of lower than expected whole rock assays) indicating losses to the system, namely calcium, and sulphur.

The textures and assemblages of the secondary minerals are indicative of, retrograde, low temperature (<350°C) alteration that may well have occurred as a result of an influx of water during the initial cooling of the intrusion. The sill was faulted and tilted into a steeply inclined attitude during the Kenoran event but no penetrative deformational fabric is evident, and the effects of regional metamorphism are minimal.

Figure 7.4 is a typical section through the Dumont sill illustrating the distribution of nickel grades in the dunite in the central portion of the deposit.

Figure 7.4: Typical Cross-Sectional View of the Dumont Deposit from Line 8350E – Looking Northwest showing outline of FS Pit



Source: RNC. Note that the scale is given in metres. Section shown is 100 m wide.

The age of the Dumont sill is not explicitly known. In early 2010, the Geological Survey of Canada (GSC) attempted to date the upper mafic zone, but was unsuccessful due to the lack of dateable minerals. The conformable nature of the body, together with the character of its differentiation, suggests that it was emplaced as a virtually horizontal sill that was folded and faulted during the Kenoran event. It is reasonable to conclude that the Dumont sill is of late Archean age, but is only slightly younger than the enclosing lavas; that are approximately 2,700 Ma (Duke, 1986).

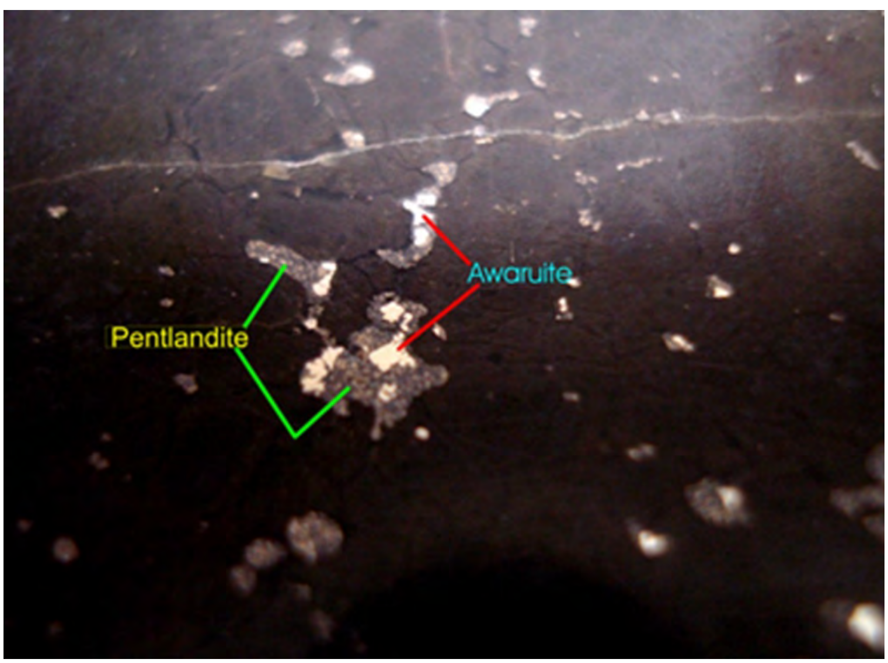
7.3 Disseminated Nickel Mineralization

Nickel-bearing sulphides and a nickel-iron alloy are enriched (grades > 0.35% nickel) in stratiform bands within the dunite subzone and are also broadly disseminated at lower concentrations throughout the dunite and lower peridotite subzones. The number and thickness of these bands varies from place to place in the deposit. Nickel sulphide and alloy concentrations decrease gradationally away from the centre of these bands toward the interband zones where mineralization continues at lower concentrations. The total nickel contained in these rocks occurs in variable proportions in sulphides, alloy and silicates depending on primary magmatic nickel mineralogy and the degree of serpentinization of the rock.

7.3.1 Nickel Mineralogy

Disseminated nickel mineralization is characterized by disseminated blebs of pentlandite ($(Ni,Fe)_9S_8$), heazlewoodite (Ni_3S_2), and the ferronickel alloy, awaruite ($Ni_{2.5}Fe$), occurring in various proportions throughout the sill. These minerals can occur together as coarse agglomerates, predominantly associated with magnetite, up to 10,000 µm (10 mm), or as individual disseminated grains ranging from 2 to 1,000 µm (0.002 to 1 mm). Figure 7.5 shows nickel mineralization in core from the Dumont property. Nickel can also occur in the crystal structure of several silicate minerals including olivine and serpentine.

Figure 7.5: Photo of the Dumont Mineralization in Core (Field of View is 5 cm wide)



Source: RNC.

The observed mineralogy of the Dumont deposit is a result of the serpentinization of a dunite protolith, which locally hosted a primary, disseminated (intercumulus) magmatic sulphide assemblage. The serpentinization process whereby olivine reacts with water to produce serpentine, magnetite and brucite creates a strongly reducing environment where the nickel released from the decomposition of olivine is partitioned into low-sulphur sulphides and newly formed awaruite. Nickel also occurs in remnant olivine and newly formed serpentine with the concentration of nickel in these minerals being dependent on the degree of serpentinization of the rock. The serpentinization process as it relates to nickel mineralogy is described in Section 7.3.3.1.

Millerite (NiS) is rare, but can be present in lesser amounts near host rock contact zones and in major fault zones. It typically occurs as fine secondary overgrowths, characteristically overprinting pentlandite and heazlewoodite in intercumulus blebs (Figure 7.19 H).

7.3.1.1 Nickel Mineralization Assemblages

Mineralized zones containing pentlandite, awaruite, and heazlewoodite, are classified into the following mineralization assemblages; sulphide dominant, alloy dominant and mixed. RNC's mineralogical sampling program (described in Section 9.3.1.) provides a quantitative analytical measure of the whole-rock mineralogy on a crushed and homogenized 1.5 m core sample, which is the basis for understanding the combination of nickel mineral phases that constitutes these three assemblages:

- Alloy mineralization is dominantly awaruite ± lesser heazlewoodite ± lesser pentlandite.
- Mixed mineralization consists of sulphides and alloy in similar proportions. Specific sub-types are heazlewoodite and awaruite in similar proportions; pentlandite and awaruite in similar proportions; or heazlewoodite + pentlandite and awaruite in similar proportions.
- Sulphide mineralization is dominantly heazlewoodite and/or pentlandite, with or without lesser awaruite.

As noted above, these assemblages contain variable proportions of nickel in silicates. These mineralization assemblages are described in detail below with the aid of EXPLOMIN™ QEMSCAN images and backscattered electrons (BSE) images.

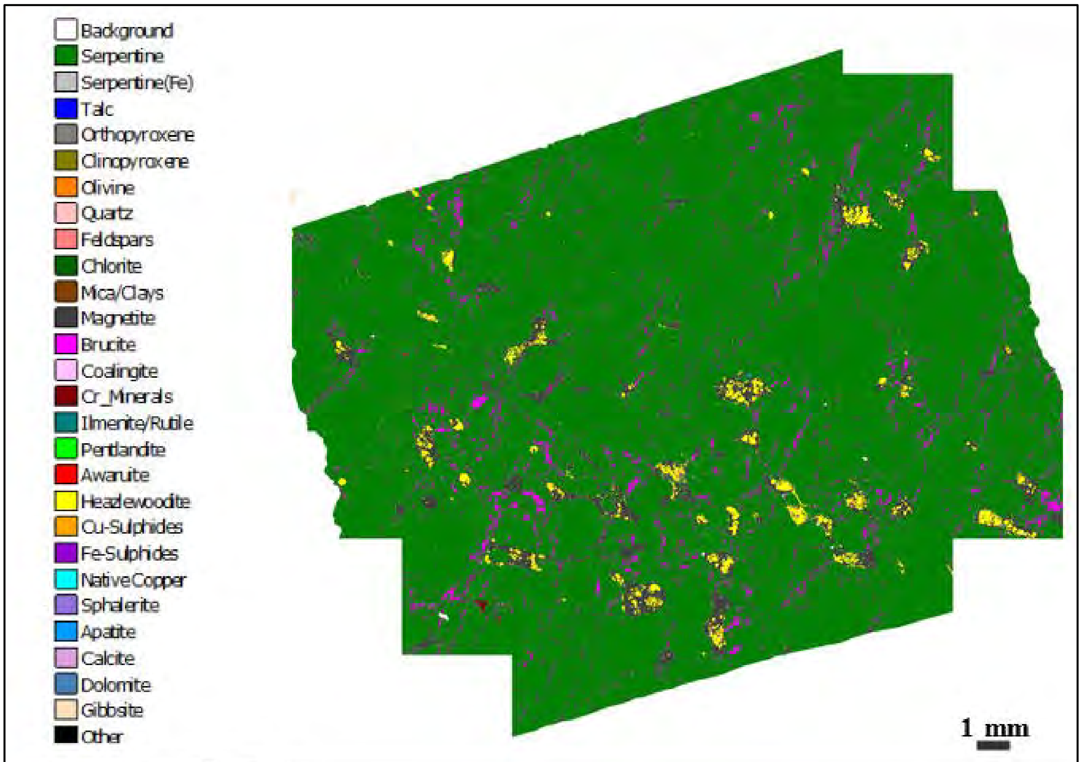
7.3.1.2 Sulphide Mineralization Assemblage

The sulphide mineralization assemblage occurs in higher-grade bands (grades > 0.35% nickel) that are subparallel to the dip of, and principally in the centre of, the sill (Figure 7.4). Sulphide mineralization is dominated by pentlandite (Pn) and/or heazlewoodite (Hz) with lesser awaruite (Aw). Pentlandite and heazlewoodite occur as medium to coarse-grained blebs occupying intercumulus spaces in a primary magmatic texture, sometime exhibiting secondary overgrows within magnetic blebs. These blebs are often intimately associated with magnetite ± brucite ± chromite ± awaruite, in intercumulus spaces (Figures 7.6 and 7.7). Where awaruite is present with sulphides, it often observed to be a secondary overgrowth on pentlandite within the primarily textures intercumulus magnetite blebs. Up to three sulphide bands are found within the dunite where it is the thickest in the central southeast region of the sill.

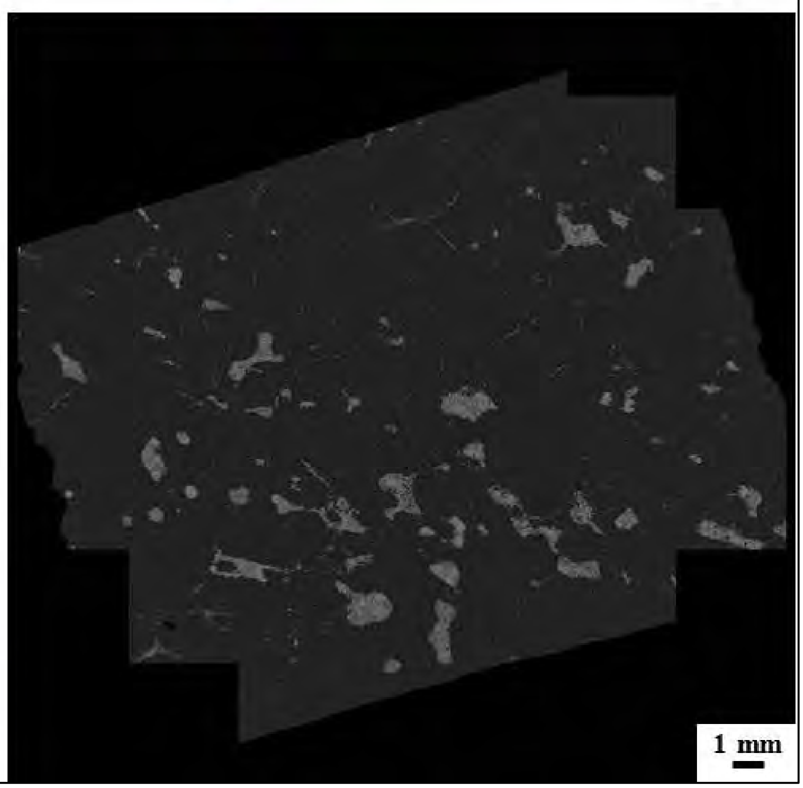
7.3.1.3 Alloy Mineralization Assemblage

The alloy mineralization assemblage is characterized by the presence of awaruite with little to no sulphides. Awaruite occurs as fine grains (generally <1 mm) associated with small intercumulus magnetite or chromite blebs. Awaruite can also be observed as a secondary overgrowth on serpentine within the pseudomorphed grain. Alloy mineralization zones occur where primary sulphides are not present and serpentinization is near complete. Figure 7.8 shows an example of the mineralogical textures in the alloy mineralization assemblage.

Figure 7.6: Sulphide Mineralization Assemblage. Heazlewoodite Dominant Sample (EXP_204)

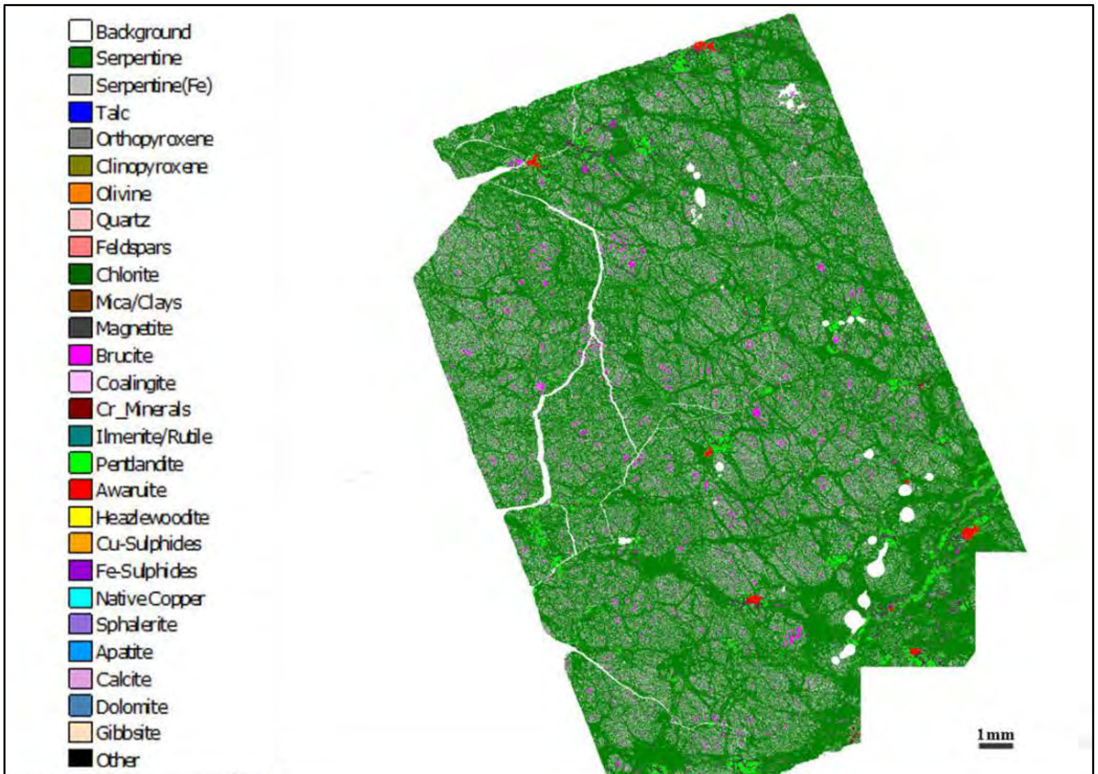


Modal Abundance	%
Serpentine (Total)	82.89
Fe-Serpentine	0.64
Talc	0.00
Orthopyroxene	0.00
Clinopyroxene	0.00
Olivine	0.04
Quartz	0.00
Feldspars	0.00
Chlorite	0.07
Mica/Clays	0.00
Magnetite	9.06
Brucite	4.65
Coalingite	0.08
Cr_Minerals	0.04
Ilmenite/Rutile	0.00
Pentlandite	0.19
Awaruite	0.02
Heazlewoodite	2.92
Cu-Sulphides	0.00
Fe-Sulphides	0.01
Native Copper	0.00
Sphalerite	0.00
Apatite	0.01
Calcite	0.00
Dolomite	0.00
Gibbsite	0.00
Total	100.00

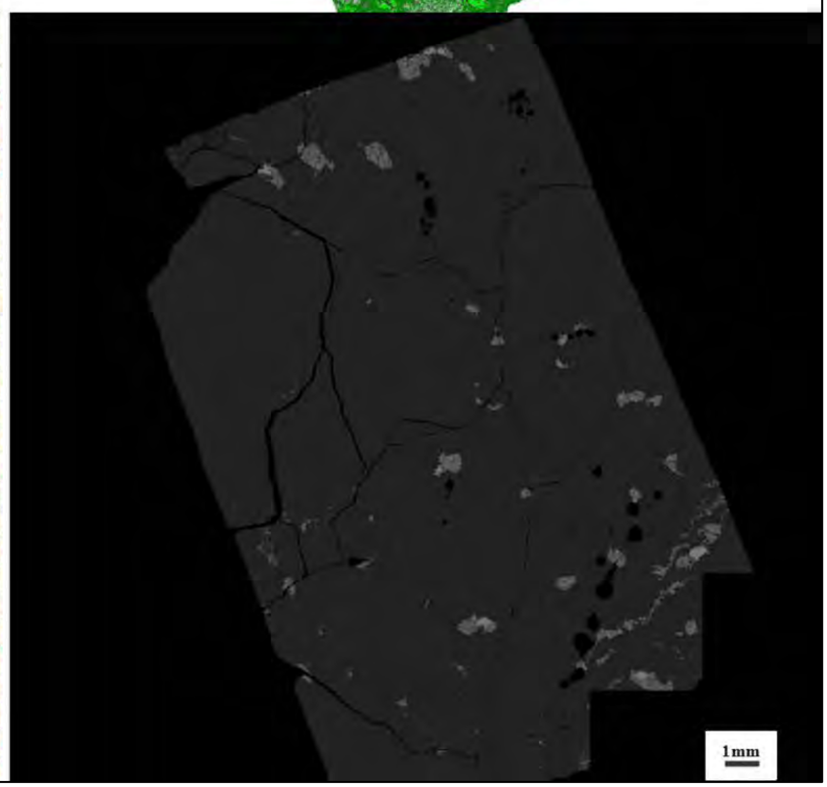


Note: Top: False-colour EXPLOMIN™ field stitch image. Bottom: Equivalent BSE image. (Heazlewoodite to pentlandite ratio 17.7). Modal Abundances as reported from EXPLOMIN™: 0.19% Pn, 2.92% Hz, 0.02% Aw, Metallic Ni 2.16% [(0.02%Aw*0.731%Ni) + (2.92%Hz*0.714Ni%) + (0.19%Pn*0.32%Ni)]. Sample contains coarse intercumulus magnetite blebs, intimately associated with heazlewoodite. Former brucite rings and pseudomorphed olivine grains in a 100% serpentinized matrix exhibit a directional fabric. **Source:** RNC.

Figure 7.7: Sulphide Mineralization Assemblage. Typical Pentlandite Dominant Sample (EXP_287)

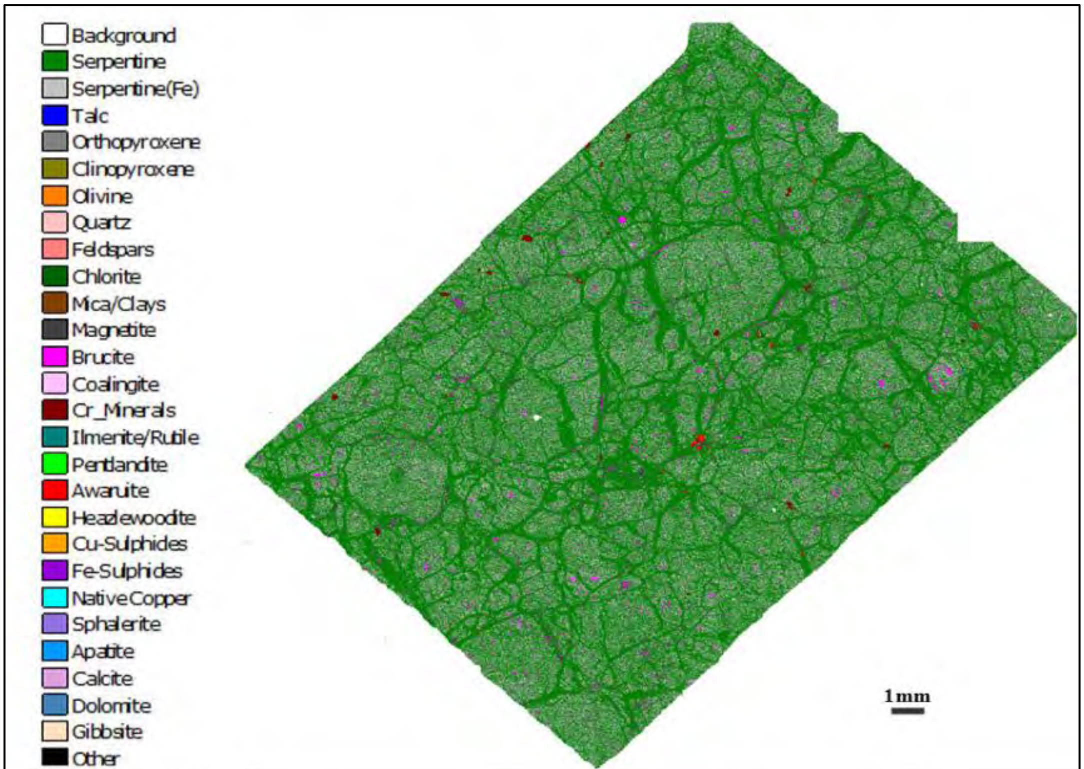


Modal Abundance	%
Serpentine (Total)	92.05
Fe-Serpentine	24.48
Talc	0.00
Orthopyroxene	0.00
Clinopyroxene	0.00
Olivine	0.00
Quartz	0.00
Feldspars	0.00
Chlorite	0.08
Mica/Clays	0.00
Magnetite	1.88
Brucite	2.15
Coalingite	0.40
Cr_Minerals	0.02
Ilmenite/Rutile	0.00
Pentlandite	2.70
Awaruite	0.68
Heazlewoodite	0.02
Cu-Sulphides	0.00
Fe-Sulphides	0.01
Native Copper	0.00
Sphalerite	0.00
Apatite	0.00
Calcite	0.00
Dolomite	0.00
Gibbsite	0.00
Total	100.00

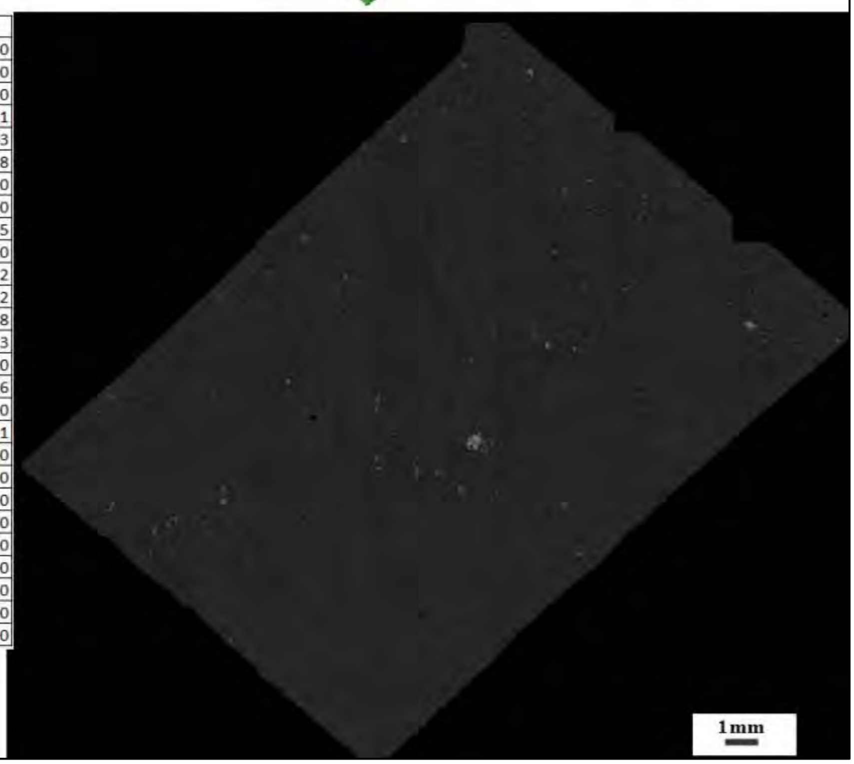


Note: Top: False colour EXPLOMIN^{TM} field stitch image. Bottom: Equivalent BSE image. (Heazlewoodite to pentlandite ratio 0.003). Field Stitch Modal Abundances as reported from EXPLOMIN^{TM}: 2.7% Pn, 0.02% Hz, 0.68% Aw, Metallic Ni 1.38% [(0.68%Aw*0.731%Ni) + (0.02%Hz*0.714Ni%) + (2.7%Pn*0.32%Ni)]. Samples contain pentlandite and awaruite somewhat associated with magnetite in intercumulus blebs. Pseudomorphed olivine grains are preserved and accentuated by iron serpentine centres. **Source:** RNC.

Figure 7.8: Alloy Mineralization Assemblage. Sample (EXP_221)

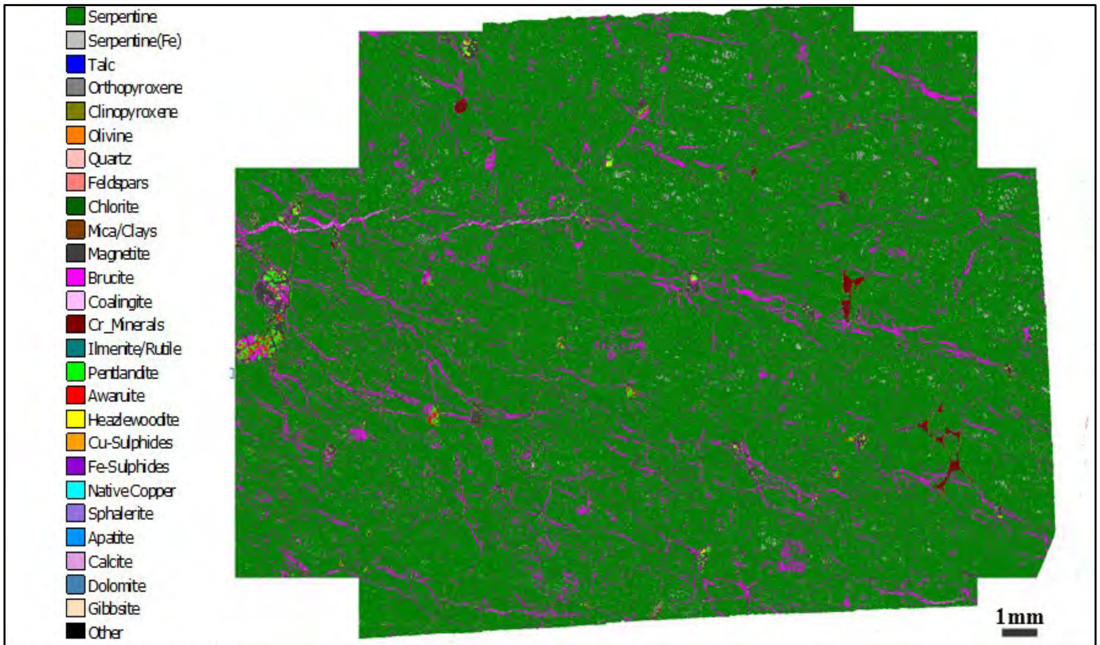


Modal Abundance	%
Serpentine (Total)	93.80
Fe-Serpentine	25.20
Talc	0.00
Orthopyroxene	0.01
Clinopyroxene	0.03
Olivine	0.58
Quartz	0.00
Feldspars	0.00
Chlorite	0.05
Mica/Clays	0.00
Magnetite	1.52
Brucite	3.02
Coalingite	0.38
Cr_Minerals	0.33
Ilmenite/Rutile	0.00
Pentlandite	0.06
Awaruite	0.20
Heazlewoodite	0.01
Cu-Sulphides	0.00
Fe-Sulphides	0.00
Native Copper	0.00
Sphalerite	0.00
Apatite	0.00
Calcite	0.00
Dolomite	0.00
Gibbsite	0.00
Total	100.00

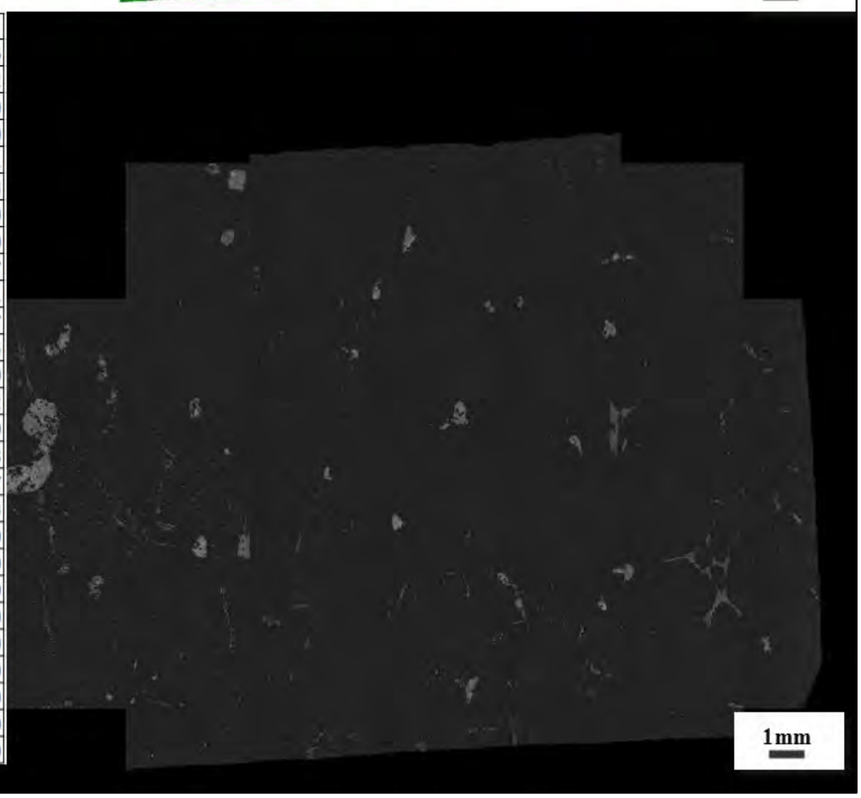


Note: Top: False colour EXPLOMIN™ field stitch image. Bottom: Equivalent BSE image. Modal Abundances as reported from EXPLOMIN™ :(0.06% Pn, 0.01% Hz, 0.20% Aw) Metallic Ni 0.17% [(0.2%Aw*0.731%Ni) + (0.01%Hz*0.714Ni%) + (0.06%Pn*0.32%Ni)]. Sample contains awaruite associated with magnetite and chromite in small intercumulus spaces. Pseudomorphed olivine grains are clearly visible, accentuated by iron serpentine and brucite centres in complete serpentinization. **Source:** RNC.

Figure 7.9: Mixed Mineralization Assemblage. Sample (EXP_256)



Modal Abundance	%
Serpentine (Total)	88.36
Fe-Serpentine	1.72
Talc	0.00
Orthopyroxene	0.00
Clinopyroxene	0.01
Olivine	0.03
Quartz	0.00
Feldspars	0.00
Chlorite	0.07
Mica/Clays	0.01
Magnetite	3.27
Brucite	6.75
Coalingite	0.20
Cr_Minerals	0.32
Ilmenite/Rutile	0.00
Pentlandite	0.48
Awaruite	0.37
Heazlewoodite	0.13
Cu-Sulphides	0.00
Fe-Sulphides	0.00
Native Copper	0.00
Sphalerite	0.00
Apatite	0.00
Calcite	0.00
Dolomite	0.00
Gibbsite	0.00
Total	100.00



Note: Top: False colour EXPLOMIN™ field stitch image. Bottom: Equivalent BSE image. Modal Abundances as reported from EXPLOMIN™ (0.48% Pn, 0.13% Hz, 0.37% Aw) Metallic Ni 0.52% [(0.37%Aw*0.731%Ni) + (0.13%Hz*0.714Ni%) + (0.48%Pn*0.32%Ni)]. Sample contains pentlandite and awaruite associated with magnetite in intercumulus spaces. Pseudomorphed olivine grains are outlined by brucite mesh rims exhibiting a directional fabric. **Source:** RNC.

7.3.1.4 Mixed Mineralization Assemblage

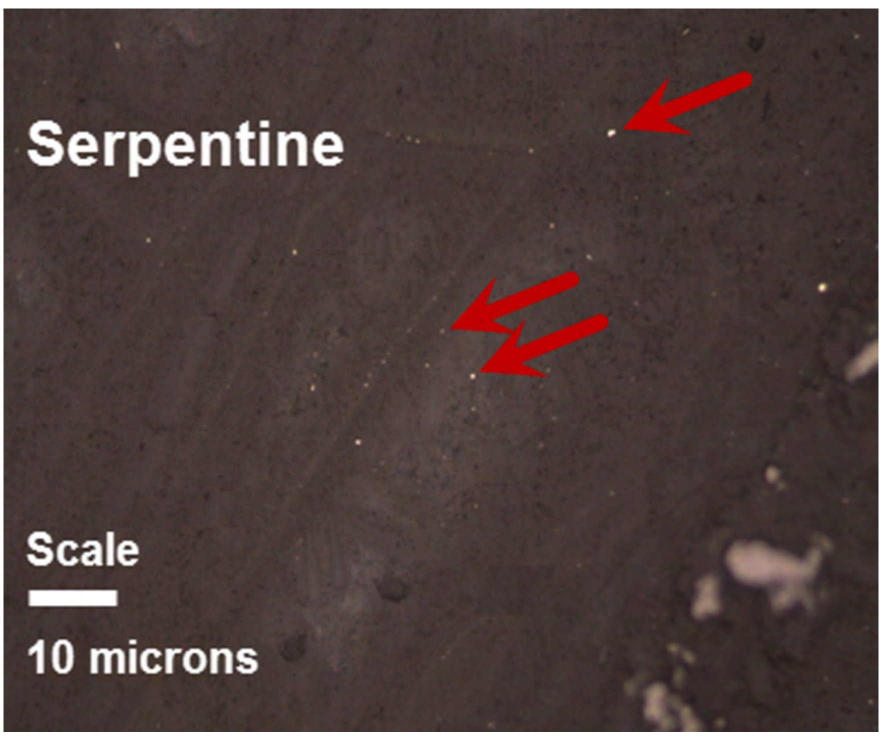
The mixed mineralization assemblage typically represents a transition from sulphide to alloy or sulphide (pentlandite) to sulphide (heazlewoodite) mineralization. The mixed mineralization assemblage contains varying amounts of sulphide (pentlandite and heazlewoodite) along with similar quantities of awaruite. Mineralization can occur as coarse sulphide-magnetite blebs associated with awaruite or as finely disseminated discrete grains. Figure 7.9 (above) shows an example of the mineralogical textures in the mixed mineralization assemblage.

7.3.1.5 Non-Mineralized Ultramafic Zones: Nickel in Silicates

As noted above, nickel in silicates occurs in varying proportions throughout the deposit. In certain portions of the deposit, a very low proportion of the nickel in the rock is contained in sulphide or alloy minerals. In these areas, the nickel in the rock occurs primarily in silicate minerals such as serpentine or olivine. These non-mineralized areas are generally low-grade (<0.25% Ni), and contain no sulphides. Usually these are areas where serpentinization is incomplete and nickel remains held within the crystal structure of olivine ($((Mg,Fe,Ni)_2SiO_4)$ and/or serpentine $(Mg,Fe,Ni)_3Si_2O_5(OH)_4$. Nickel occurring in this mode would not be recoverable through the flotation and magnetic separation methods considered by RNC for Dumont.

In some of these zones, the nickel is not actually contained in the crystal structure of the serpentine, but occurs as very fine (<1 µm) sulphide or awaruite inclusions within the serpentine matrix (Figure 7.10).

Figure 7.10: BSE Image of Fine Nickel Inclusions in a Serpentine Matrix



Note: 500x magnification: Fine Ni-mineral inclusions (<1 µm, indicated by red arrows) in host matrix of serpentine (dark grey). **Source:** RNC.

The term “nickel in silicates” as used herein refers to nickel contained within minerals other than pentlandite (Pn), awaruite (Aw) and heazlewoodite (Hz), either as very fine inclusions of the three minerals too small to be classified as Pn, Hz or Aw by EXPLOMIN™, or within the mineral structure of the silicate minerals. The proportion of nickel in silicates varies throughout the sill (Table 7-1) and is dependent on the strength or state of serpentinization. Zones of the intrusion that are partially or weakly serpentinized generally have a larger proportion of nickel contained in silicates (High Iron Serpentine Domain, Table 7-1), compared to those that have been strongly serpentinized (Heazlewoodite Dominant and Mixed Sulphide, Table 7-1). Zones bearing sulphides generally have a lower proportion of nickel in silicates than those containing no sulphide (Table 7-1). These zones correlate with metallurgical recovery as discussed in Section 7.7.

Table 7-1: Average % Ni in Silicates of EXPLOMIN™ Samples by Serpentinization Domain (as defined in Section 7.7)

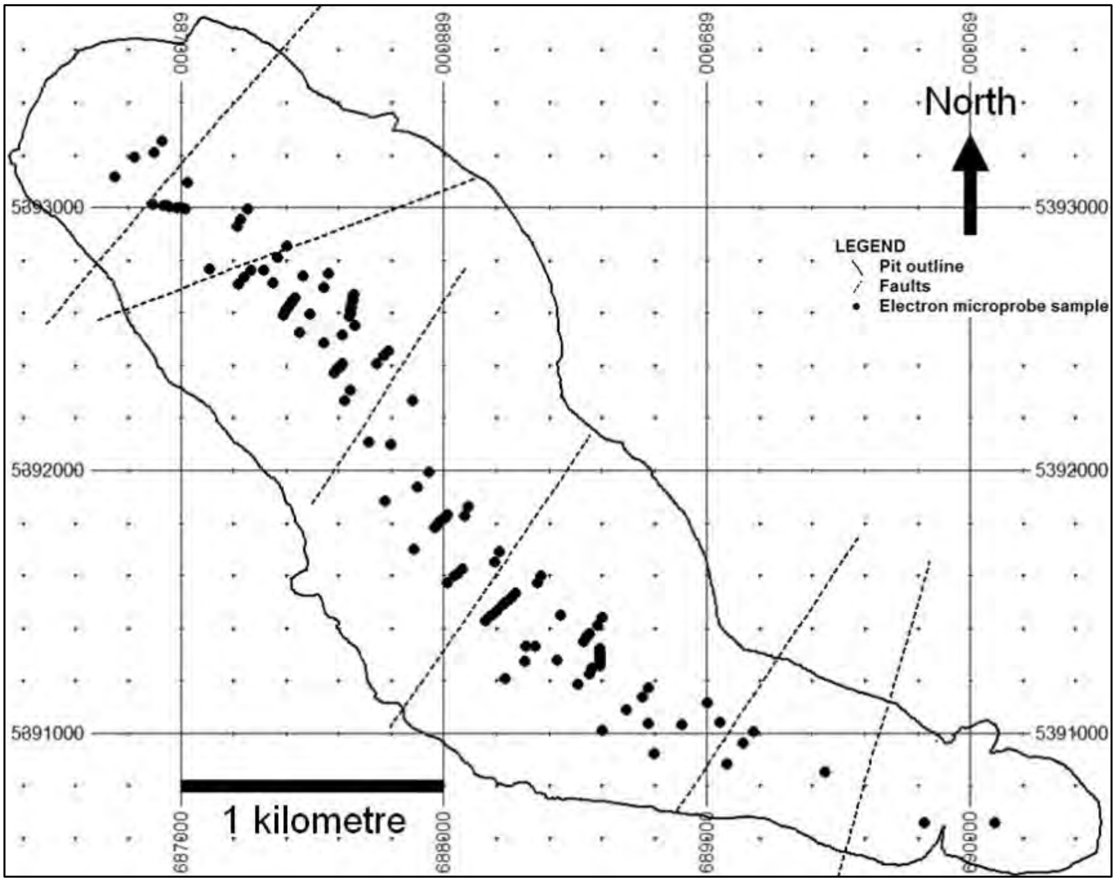
	All Samples in Domain		Sulphide Samples		Non-Sulphide Samples	
Domain	# Samples	Average Nickel in Silicates %	# Samples	Average Nickel in Silicates %	# Samples	Average Nickel in Silicates %
Heazlewoodite Dominant	521	37.3	124	15.54	397	44.06
Mixed Sulphide	162	34.1	64	16.4	98	45.8
Pentlandite Dominant	390	31.1	203	20.19	187	42.9
High Iron Serpentine	347	55.8	135	39.5	212	66.1

Note: The “# Samples” refers to the number of EXPLOMIN™ samples within each serpentinization domain described in Figure 7.15. “% Ni in silicates” is a calculated value based on the modal abundances of pentlandite (Pn), heazlewoodite (Hz) and awaruite (Aw) in the sample. % Ni in silicates = [(Nickel Assay - Metallic Nickel)/Nickel Assay], where the metallic nickel = % Modal abundance of Pn * %Ni in Pn + % Modal abundance of Hz * %Ni in Hz + % Modal abundance of Aw * %Ni in Aw. Where heazlewoodite modal abundance <0.1%, the average value of 27.3% Ni in Pn from electron microprobe data was used, for heazlewoodite modal abundance >=0.1, 32% Nickel was used for pentlandite. 73.1% and 71.4% Ni was used for Aw and Hz respectively across all domains. “Non-sulphide” is considered to be samples with sulphur <0.07% **Source:** RNC.

7.3.1.6 Nickel Tenor & Compositional Variability of Recoverable Minerals

Electron microprobe analyses were performed to quantify the variability of nickel content (tenor) in key minerals of interest for samples from locations throughout the Dumont deposit (Figure 7.11). All minerals analysed showed low variability in nickel tenor throughout the sill with the exception for pentlandite and serpentine (Table 7-2).

Figure 7.11: Location of Electron Microprobe Samples



Source: RNC.

Table 7-2: Electron Microprobe Results

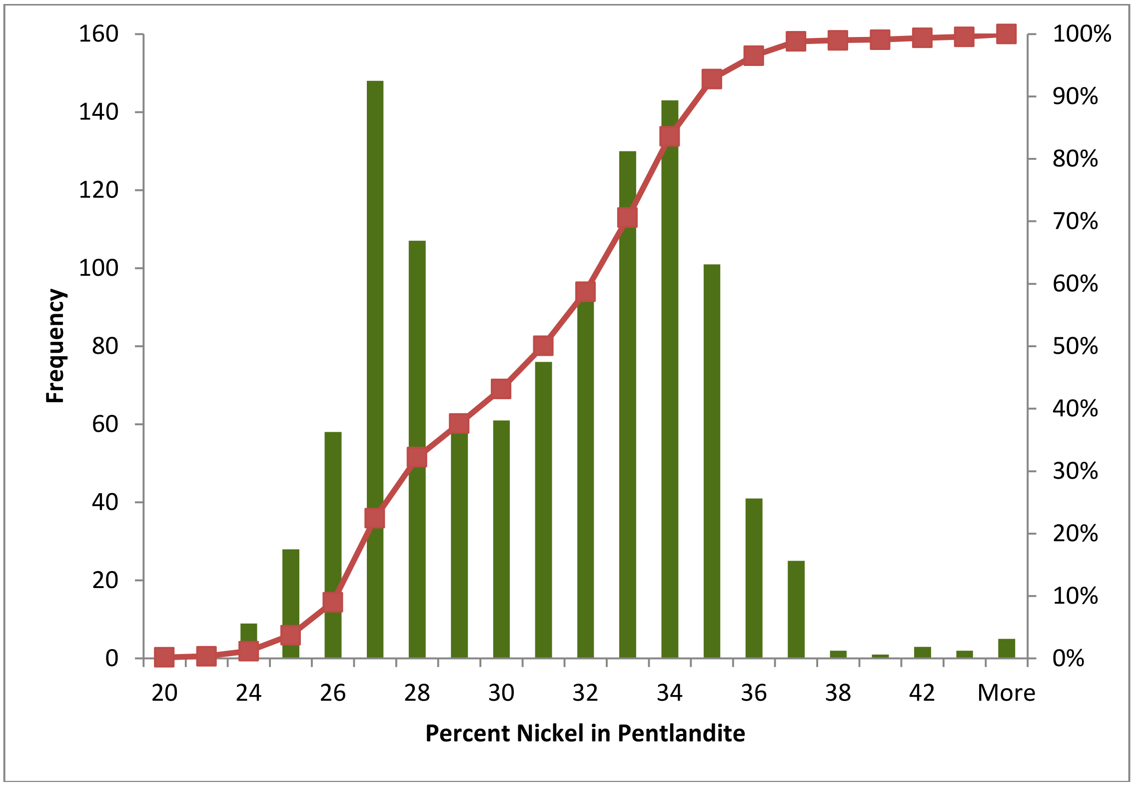
	Minimum Value (% Ni)	Maximum Value (% Ni)	Average (% Ni)	Number of Points	Standard Deviation	Number of Samples
Pentlandite	18.21	52.58	30.54	1103	3.65	117
Awaruite	59.03	89.86	72.85	699	3.10	118
Heazlewoodite	61.14	74.31	72.08	641	1.01	99
Olivine	0.124	0.4	0.29	131	0.06	7
Serpentine	0.00	1.31	0.13	917	0.14	51
Chromite	0.056	0.090	0.071	14	0.009	2
Magnetite	0	1.604	0.072	893	0.162	144

Note: Statistics for point data collected within mineral grains from various locations across the Dumont dunite. **Source:** RNC.

Sulphide and Awaruite

Pentlandite shows the most variability of the metallic minerals and exhibits a bimodal population (Figure 7.12). For samples where nickel tenor in pentlandite is lower, the lower nickel values are mostly associated with an increase in iron, and less so, sulphur. Within each subgroup, nickel tenor variability is low (Table 7-3).

Figure 7.12: Frequency Distribution for Percent Nickel in Pentlandite



Source: RNC.

The bimodal distribution suggests that two populations are present. These populations correspond to spatially continuous zones within the deposit. Pentlandite, which is hosted by weakly serpentinized rock (Zones 3a, 4 in Figure 7.21), exhibits lower Ni tenors, compared to the higher Ni tenors of pentlandite in strongly serpentinized dunite (Zones 1, 2 &3b, Figure 7.21).

Table 7-3: Statistics for High & Low Ni Pentlandite Groups

	Minimum Value (% Ni)	Maximum Value (% Ni)	Average (% Ni)	Number of Points	Standard Deviation
Low Ni Pentlandite	18.21	29.99	27.01	474	1.58
Hi Ni Pentlandite	29.96	52.58	33.23	624	2.23

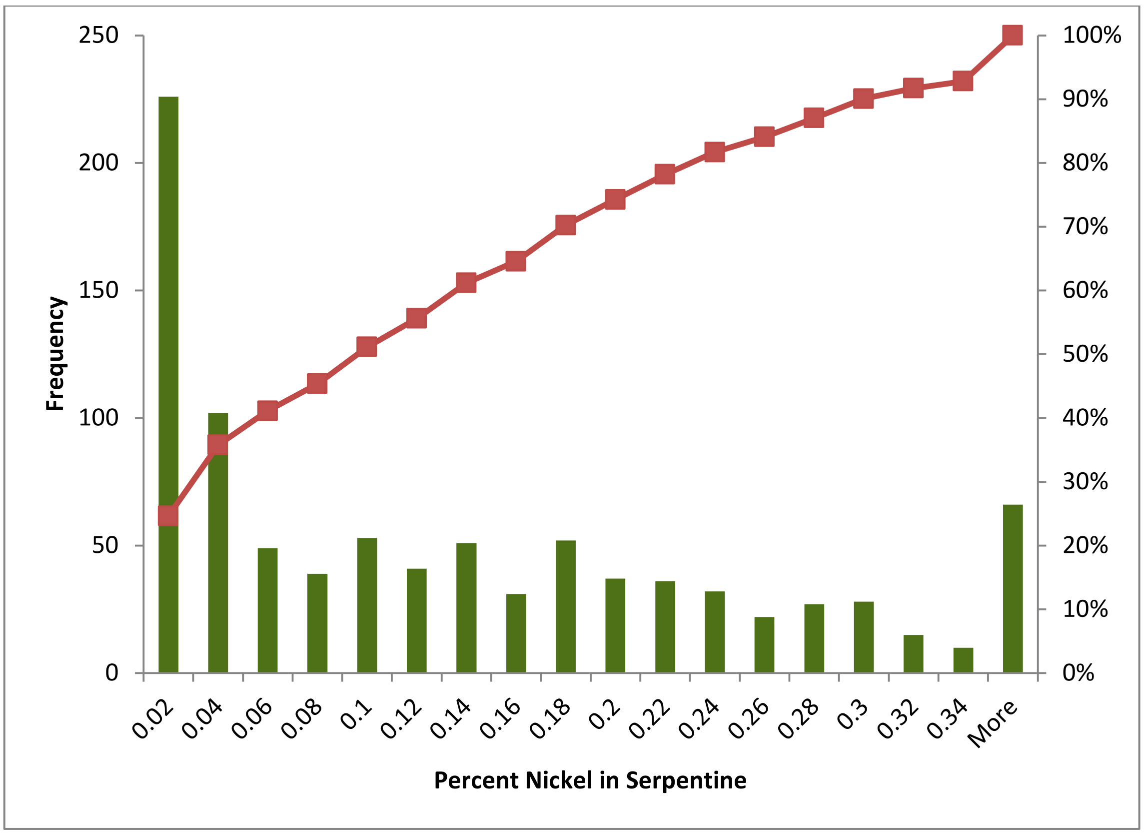
Source: RNC.

Table 7-2 shows that heazlewoodite is the least variable of the three main nickel bearing minerals of interest, followed by awaruite. Eighty percent of the microprobe values measured for awaruite are between 71% and 75%.

Serpentine

As expected, serpentines show a wide range of nickel tenors (Figure 7.13). At some analysis points nickel is reported at values higher than commonly expected within the serpentine structure **(**(Mg, Fe, Ni)$_3$Si$_2$O$_5$(OH)$_4$)**.** As shown in Figure 7.10, serpentine can host inclusions of very fine-grained awaruite in its matrix. Those points where the nickel content in serpentine is reported as uncommonly high by the microprobe are likely measurements of sulphide or alloy inclusions finer than the width of the electron beam (Stephanie Downing, Senior Mineralogist, SGS Lakefield, pers. com.). The presence of fine nickel inclusions tends to be more common in samples that are higher in iron-serpentine content.

Figure 7.13: Frequency Distribution & Cumulative Frequency Plot for Percent Nickel in Serpentine

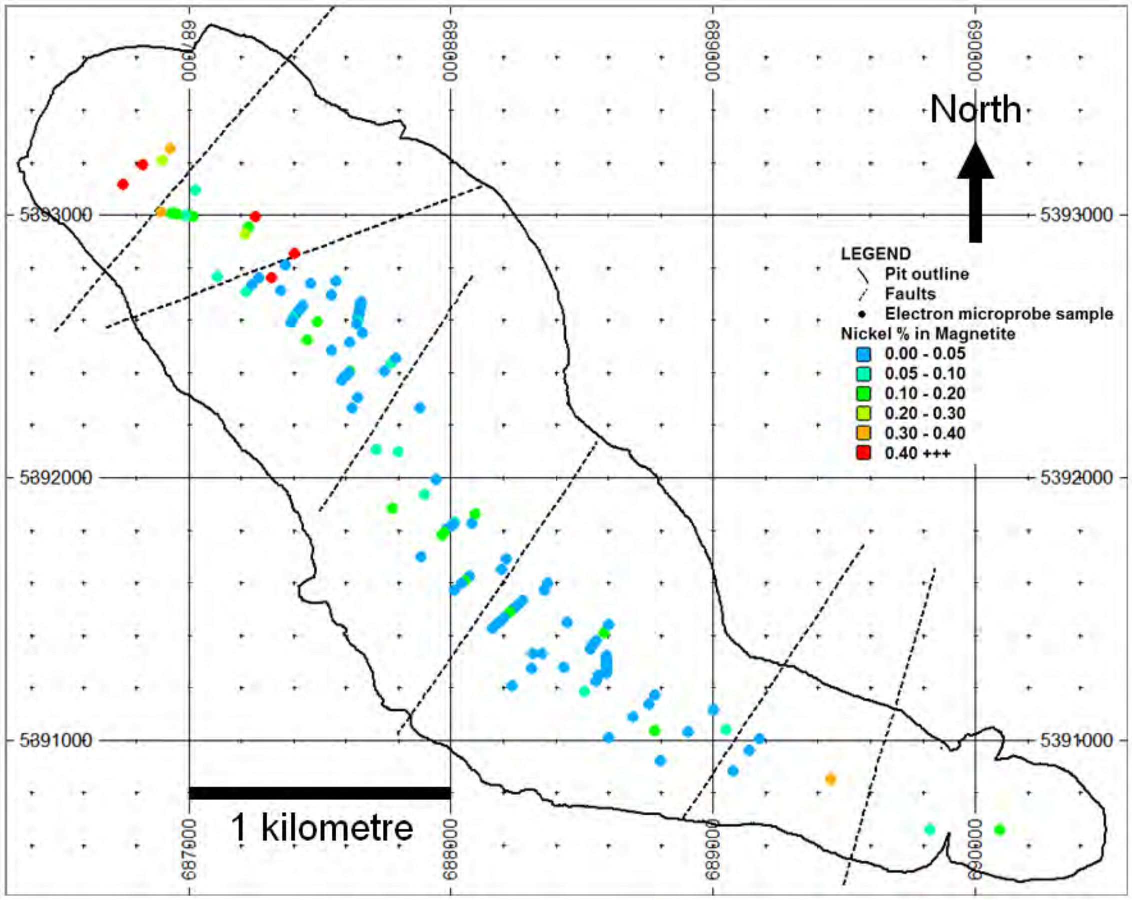


Note: 51% of data has less than 0.1% Ni in Serpentine. **Source:** RNC.

Magnetite

Magnetite was analysed over 893 points in 144 samples (Figure 7.14) by electron microprobe for the elements listed in Table 7-4.

Figure 7.14: Location of Magnetite Electron Microprobe Samples (Coloured by Ni% in Magnetite)



Source: RNC.

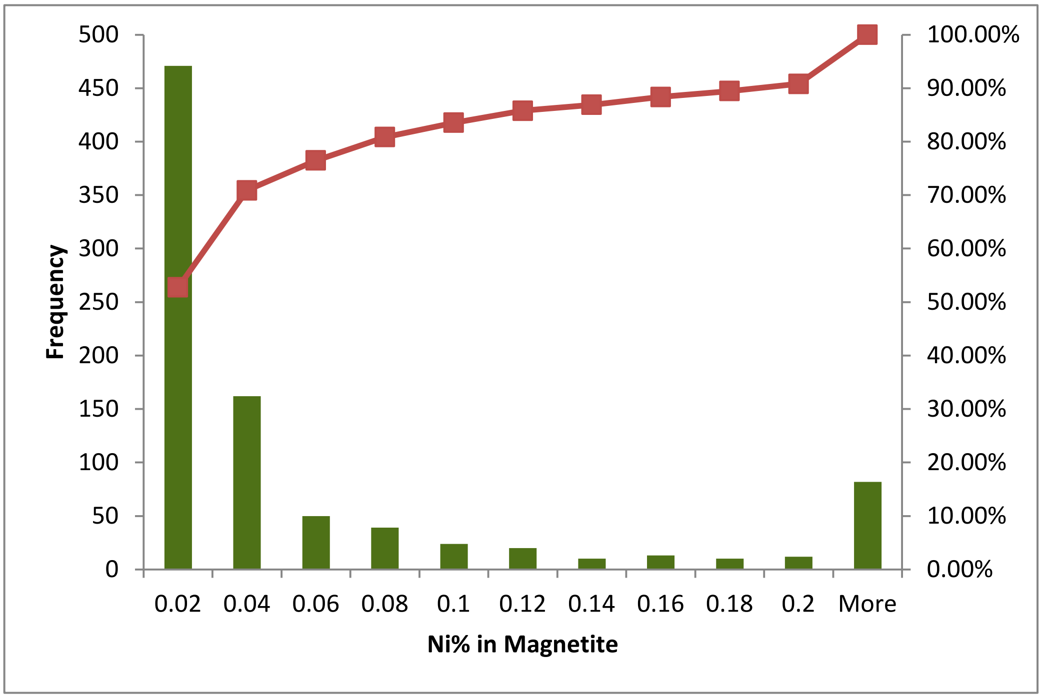
Table 7-4: Electron Microprobe Analyses for Magnetite

	Si	Mg	Fe	Cr	Ni	Al	Mn	Ti	Co	Zn	V	Ca	Na	P	K
Avg.	0.07	0.22	71.2	0.14	0.07	0.00	0.09	0.01	0.04	0.01	0.01	0.00	0.00	0.00	0.01
Max	1.66	5.72	73.1	10.27	1.60	0.17	1.42	0.29	0.16	0.37	0.40	0.30	0.14	0.02	0.03
Min	0.00	0.00	59.6	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Stdev	0.16	0.33	1.3	0.55	0.16	0.01	0.10	0.03	0.03	0.02	0.02	0.01	0.01	0.00	0.01
N	893	893	893	893	893	893	893	893	864	853	853	853	824	824	824

Source: RNC.

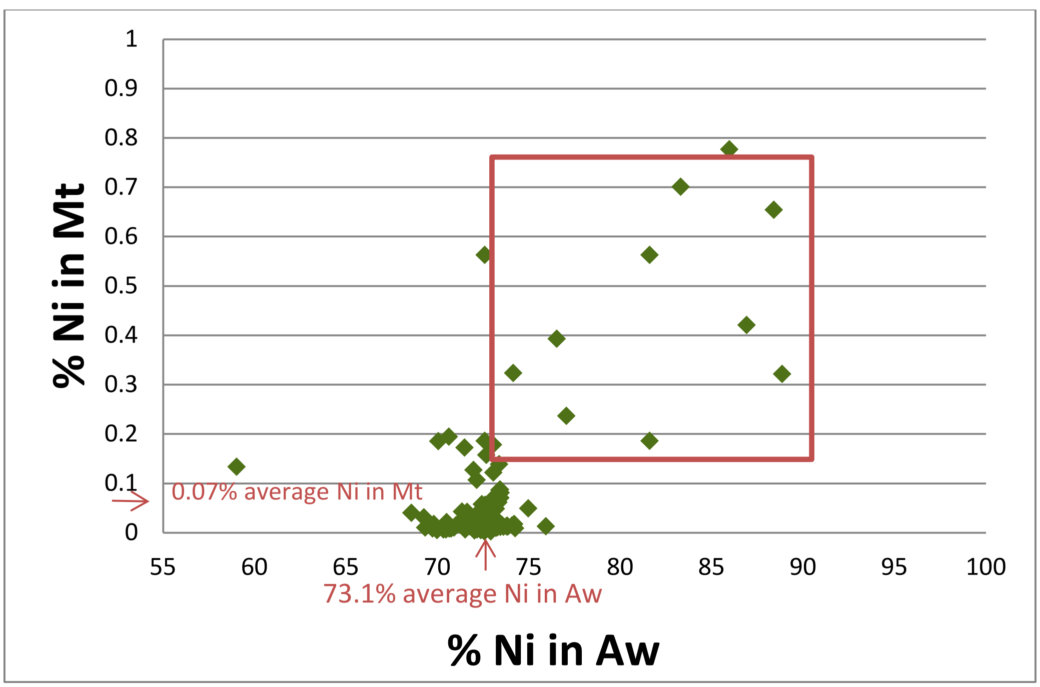
Magnetite on average contains 0.07% Ni by weight. 77% of 893 points analysed have values less than 0.06% (Figure 7.15). The group in Figure 7.15 with greater than 0.2% Ni in magnetite is associated with a zones containing Aw with higher than expected Ni (Figure 7.16).

Figure 7.15: Percent Nickel in Magnetite Distribution



Source: RNC.

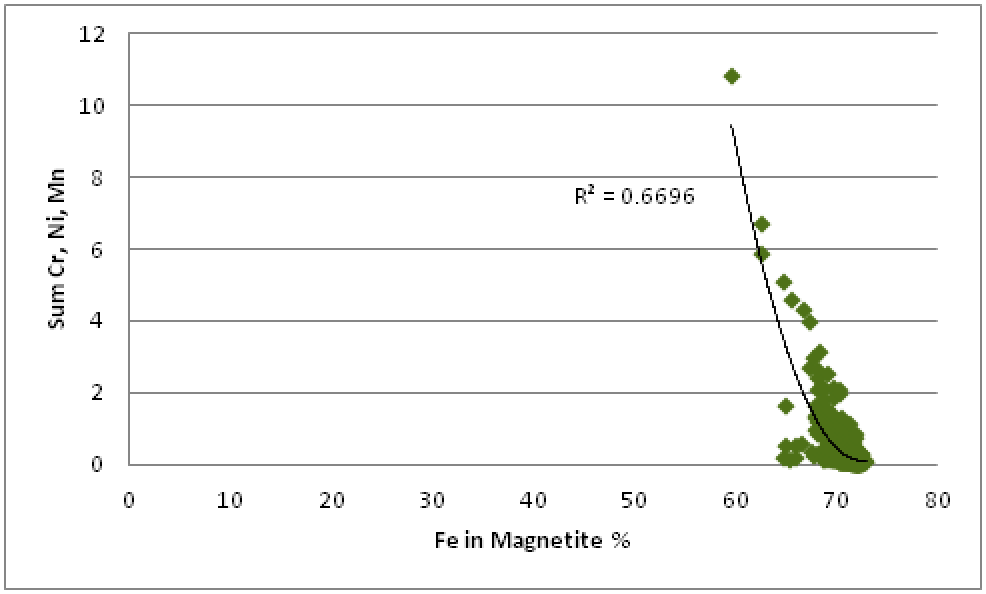
Figure 7.16: Percent Nickel in Awaruite vs. Percent Nickel in Magnetite



Source: RNC.

The iron content in magnetite shows low variability with an average of 71.2% Fe and standard deviation of 1.3. Ni, Cr, Mn are variable at the expense of changes in Fe content. The sum of the weight percent of Ni, Cr, Mn accounts for approximately 67% of the variability of Fe seen in magnetite EMP data (Figure 7.17). The remaining variability in Fe is due to spikes in Mg and Si which are attributed to edge effects. As a result of the secondary nature of magnetite, it is often intimately associated with serpentine in intercumulus bleb spaces; therefore the decreases in iron content which are associated with spikes in Mg and Si and are thought to be magnetite and serpentine associated on a scale of that of the electron beam.

Figure 7.17: Fe % vs. the Sum of Cr, Mn & Ni; Fe Content Increases with Decreases in Cr, Ni, Mn



Source: RNC.

Cobalt

Cobalt can be hosted in various quantities in each of the previously discussed minerals; pentlandite $(Co,Ni,Fe)_9S_8$, heazlewoodite $(Co,Ni)_3S_2$, awaruite$(Co,Ni)_3Fe$, serpentine($(Co,Mg,Fe,Ni)_3Si_2O_5(OH)_4$ and magnetite ($Fe_{3-x}Co_xO_4$). Pentlandite hosts the most cobalt by weight percent with an average of 3.96% Co, followed by awaruite with an average of 1% Co. (Table 7-5).

Table 7-5: Cobalt Weight % in Pentlandite, Heazlewoodite, Awaruite, Serpentine & Magnetite as per Microprobe Data

	Average (%)	Maximum	Minimum	Standard Deviation	Number of Points
Pentlandite	3.96	40.53	0.34	4.96	1098
Heazlewoodite	0.06	2.95	0.00	0.25	646
Awaruite	1.00	5.05	0.02	0.91	699
Serpentine	0.00	0.05	0.00	1.62	917
Magnetite	0.04	0.16	0.00	0.03	864

7.3.2 Controls on Nickel Distribution & Mineralization – Serpentinization

The variability in the final mineral assemblage and texture of the disseminated nickel mineralization in the Dumont deposit has been controlled primarily by the variable degree of serpentinization that the host dunite has undergone.

Serpentinization is a metamorphic process involving heat and water in which low-silica mafic and ultramafic rocks are oxidized and hydrolysed with water into serpentinite. Peridotites and dunites are converted to serpentine, brucite and magnetite. In the process, large amounts of water are absorbed into the rock increasing the volume and destroying the structure. The density changes from 3.3 to 2.7 g/cm3 with a concurrent volume increase of approximately 40%. The reaction is exothermic and large amounts of heat energy are produced in the process. Rock temperatures can be raised by nearly 260°C. The chemical reactions producing the magnetite produce hydrogen gas. Sulphates and carbonates are reduced and form methane and hydrogen sulphide.

Generalized reactions for the serpentinization of olivine:

Olivine + Water = Serpentine + Brucite

$$2Mg_2SiO_4 + 3H_2O = Mg_3Si_2O_5(OH)_4 + Mg(OH)_2$$

Olivine + Water + Oxygen = Serpentine + Magnetite

$$6Mg_{1.5}Fe_{0.5}SiO_4 + 6H_2O + 0.5O_2 = 3Mg_3Si_2O_5(OH)_4 + Fe_3O_4$$

In the early stages of the serpentinization process, water reacts with the primary phases hosting ferrous iron, a strong reducing environment is created where the ferrous iron is oxidized resulting in the production of dihydrogen and magnetite (Klein & Bach, 2009). In these early stages of serpentinization, olivine is decomposed to form iron and magnesium serpentine, iron brucite, magnetite and enough hydrogen so that the nickel-iron alloy awaruite is stable (Frost and Beard, 2007). Under such conditions, awaruite is produced by the desulphurization of pentlandite by equation (1) (Figure 7.19, B and C).

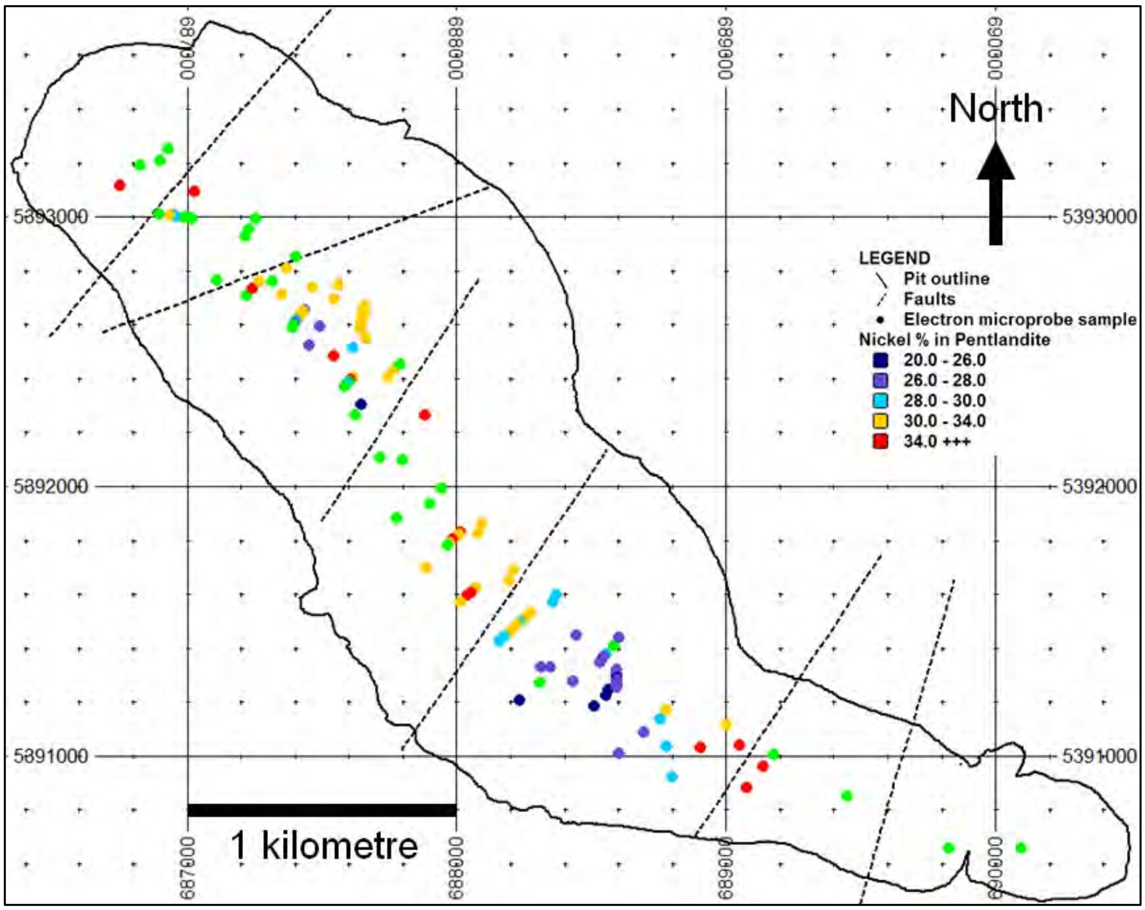
Pentlandite + Hydrogen + Water = Awaruite + Magnetite + Hydrogen Sulphide

$$Ni_{4.5}Fe_{4.5}S_8 + 4H_2(aq) + 4H_2O = 1.5Ni_3Fe + Fe_3O_4 + 8H_2S(aq) \quad \textbf{(1)}$$

In zones of the Dumont dunite where the dominant assemblage is iron (Fe) and magnesium (Mg) serpentine + MgFe brucite ± magnetite ± olivine, serpentinization is incipient (Figure 7.21, Zones 3a, 4 and 5). Here the dominant nickel-bearing phases are pentlandite and awaruite (Figure 7.19, A to D). In the stratigraphically higher dunite in the central southeast (Figure 7.21, Zone 3a and 4), where olivine has almost been exhausted but the iron-rich phases of serpentine and brucite remain, awaruite grains are the coarsest observed in the Dumont sill and are clearly secondary overgrowths on pentlandite. (Figure 7.19, B to D). Since most of the iron is tied up in serpentine and brucite, the modal abundance of magnetite is low (<2%), thus the intercumulus blebs of pentlandite and awaruite are low in magnetite and can be devoid of magnetite altogether. In zones of the stratigraphically lower dunite (Figure 7.21, High Iron Serpentine Domain), where the olivine content can increase to as much as 40%, awaruite is almost never present and pentlandite is the dominant metallic nickel-bearing phase (Figure 7.19, A). Intercumulus blebs are often without magnetite as significant amounts of olivine and iron serpentine are the major reservoirs of iron in this early stage.

Serpentinization is considered to take place on a grain-by-grain scale, whereby nickel is removed from the olivine and serpentine structure and mobilized from silicates to the metallic phases of the intercumulus blebs, resulting in an increase in the tenor of the nickel bearing phases in the intercumulus blebs (Duke, 1986). As a result, in zones where serpentinization is incomplete (Figure 7.21, High Iron Serpentine Domain), the percentage of nickel in silicates existing within the silicates structure or as microscopic inclusions of alloy and sulphide (Figure 7.10) is generally higher (see Table 7-1). This incomplete remobilization of nickel to the intercumulus blebs has resulted in the population of lower tenor pentlandite (Table 7-5) associated with incomplete serpentinization (Figure 7.18).

Figure 7.18: Distributions of Ni Tenor in Pentlandite

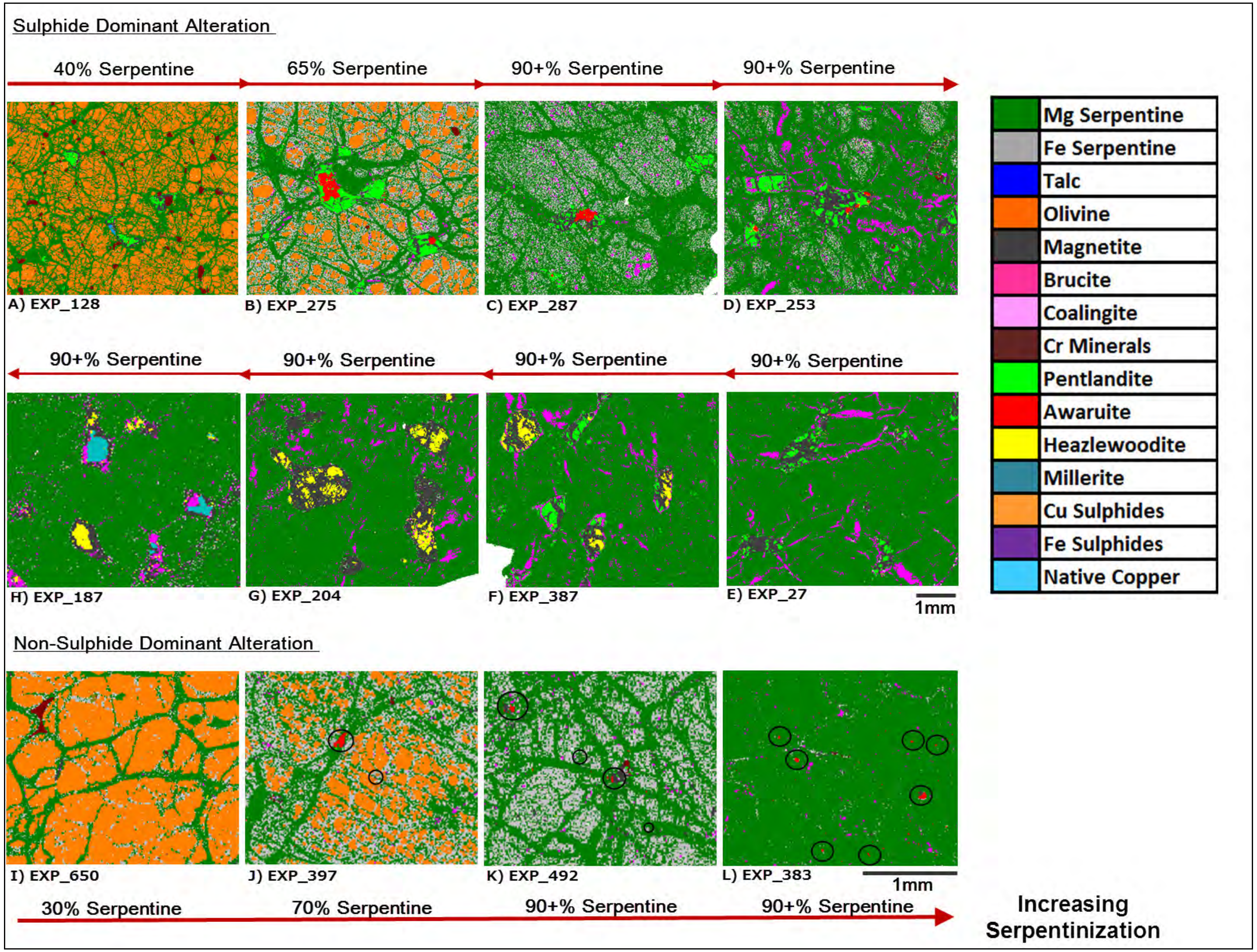


Source: RNC.

The mineralization envelope is cut by faults which define the structural domain boundaries (Figure 7.3). Serpentinization zones correspond to letters A to G descriptions in Figures 7.19 and 7.21. (1) G, (2) E to G, (3a) C-D, (3b) D-E, (4)C-E and (5)A&B. Note that "H" is not displayed, because it does not correspond to broad zones, but is restricted locally to large fault zones. The basal contact where millerite ("H") can sometimes be found is outside of the mineralization envelope shown in this figure.

Ausenco

Figure 7.19: Serpentinization Process of Sulphides Represented by EXPLOMIN™ QEMSCAN Mineralogy Sections within the Dumont Dunite



Note:

A) Incipient serpentinization. A significant amount of olivine remains. The process has not continued enough as to produce reducing conditions where the alloy awaruite is stable.

B) Olivine is decomposed to form Fe and Mg serpentine, Fe brucite, and weak magnetite. Enough hydrogen is produced so that the nickel-iron alloy awaruite is stable. Pentlandite is desulphurized to produce awaruite (equation 1).

C) Olivine is exhausted. Fe and Mg serpentine and Fe brucite remain. The breakdown of the Fe-rich phases has begun to produce more magnetite in intercumulus blebs. Brucite exists as pseudomorphed olivine centres.

D) the near complete of the breakdown of iron-rich silicate phases, mainly Fe-brucite and serpentine, continue to produce mg-rich brucite rims, mg serpentine and more intercumulus magnetite. Awaruite begins to decompose to produce magnetite (equation 3).

E to G) the remaining serpentines and brucites are Mg-rich. At this point the stability fields for serpentine and brucite expand such that they begin to consume previously produced alloys such as awaruite The accompanying increase in oxygen fugacity causes pentlandite and awaruite to continue to break down to produce heazlewoodite (equation 2 and 3).

H) Serpentinization has continued well beyond the total consumption of olivine. The increase in oxygen fugacity has an associated increase in sulphur fugacity as a result magnetite is replaced by sulphur-rich nickel sulphides such as millerite (equation 4). Images

I to J) are the non-sulphide analogues of A to H serpentinization. I is the least serpentinized with little to no awaruite.

L) is the most serpentinized with abundant awaruite. In the non-sulphide process, awaruite appears to be stable/occurs well beyond complete serpentinization.

Black circles highlight awaruite occurrences.

Source: RNC.

As serpentinization continues and olivine is consumed, iron serpentine and iron brucite break down to produce more magnetite, while the remaining serpentine and brucite become increasingly magnesium rich (Figure 7.19 D to G). At this point, the stability fields for serpentine and brucite expand such that they begin to consume previously produced alloys such as awaruite (Figure 7.19 transition from D to E). This results in an accompanying increase in oxygen fugacity (Beard and Frost, 2007). Under such conditions, pentlandite and awaruite continue to break down to produce heazlewoodite by equations (2) and (3) (Klein and Bach, 2009) (Figure 7.19 F and G).

Pentlandite + Water = Heazlewoodite + Magnetite + Hydrogen + Hydrogen Sulphide

$$Ni_{4.5}Fe_{4.5}S_8 + 6H_2O = 1.5Ni_3S_2 + 1.5Fe_3O_4 + H_2(aq) + 5H_2S(aq) \qquad \textbf{(2)}$$

Awaruite + Hydrogen Sulphide + Water = Magnetite + Heazlewoodite + Hydrogen

$$3Ni_3Fe + 6H_2S(aq) + H_2O = Fe_3O_4 + 3Ni_3S_2 + 10H_2(aq)) \qquad \textbf{(3)}$$

In zones where serpentinization is complete (Figure 7.21, Zones 1, 2, and 3b), intercumulus blebs contain abundant magnetite ± pentlandite ± heazlewoodite with little to no awaruite (Figure 7.19, E to H). Generally, heazlewoodite and awaruite exhibit negative correlation on a zone scale. Where heazlewoodite content is high, awaruite is low. Where sulphides are not present, awaruite exists as finely disseminated grains associated with magnetite or brucite mesh rims. However, on the scale of a thin section, heazlewoodite and awaruite can occur together in the same bleb. Nickel remobilization to intercumulus spaces has been completed by late-stage serpentinization, thus the percentage of nickel hosted in silicates is generally lower in Zones 1, 2 and 3b (Figure 7.21) and the nickel tenor of pentlandite is higher. This is represented by the population of higher Ni tenor of 30% to 35% in Figure 7.18.

Locally both early- and late-stage serpentinization features can be present in the same thin section. This effect is thought to be related to regional deformation and faulting. Localized strain may have caused fluid to travel through newly formed stress fractures focusing serpentinization along their route while leaving the relict olivine centres intact. Many of these thin sections exhibit a directional fabric that supports this hypothesis. (Figure 7.20 overleaf)

Serpentinization can continue well beyond the total consumption of olivine. In very late-stage serpentinization where the common assemblage is Mg-serpentine + Mg brucite + magnetite ± heazlewoodite, as serpentinization continues, steatitization can occur where magnetite is replaced by sulphur-rich nickel sulphides such as millerite as per equation 4 (Figure 7.14G). These transitions indicate increasing oxygen and sulphur fugacities (Eckstrand, 1975; Frost 1985). Mg serpentine and brucite can break down to produce talc (Klein and Bach, 2009). This is rare within the Dumont dunite, although observed locally around major structures and with more regularity at the basal contact of the intrusion (outside of resource envelope) where fluid flux was probably high.

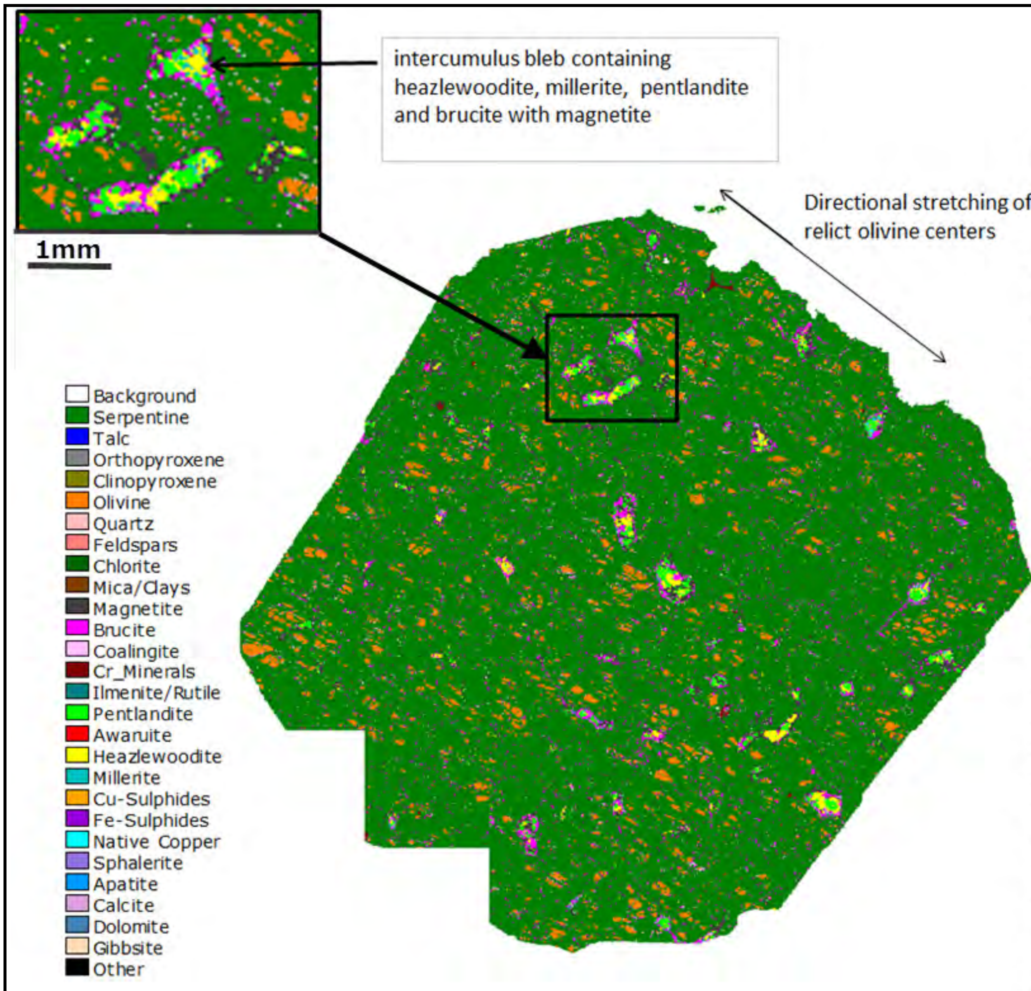
Heazlewoodite + Hydrogen Sulphide = Hydrogen + Millerite

$$Ni_3S_2 + H_2S_{(aq)} = H_{2(aq))} + 3NiS \qquad \textbf{(4)}$$

In areas of the deposit where low concentrations of sulphur occur, the above serpentinization scheme is modified by the absence of sulphide phases. Where intercumulus sulphide blebs are not present, sulphur assay values are on average less than 0.05%, and awaruite is the dominant metallic nickel-bearing phase (Figure 7.19 K to L). This suggests that awaruite formation is not controlled by the desulphurization of primary sulphides.

For non-sulphide zones, where serpentinization is weak, the nickel in silicates is higher (Table 7-1), which in turn is associated with low values of awaruite modal abundances (Table 7-6). In non-sulphide zones where serpentinization is complete, the nickel in silicates values are lower (Table 7-1) and the modal abundances of awaruite is highest (Table 7-6). This evidence suggests that where serpentinization is incomplete or weak, and the remobilization of nickel is not complete, more nickel is hosted in silicates as opposed to creating awaruite. Where serpentinization and the associated nickel re-mobilization are complete, larger amounts of nickel have gone into forming awaruite, leaving less nickel in silicates. At later stages of serpentinization, well beyond the exhaustion of olivine, awaruite does break down, as modal abundances are low in domain 1 which corresponds to late stage serpentinization in Figure 7.19, G and H. This suggests awaruite has a range of stability which is associated with partial to complete serpentinization. Samples with more awaruite have less nickel in silicates.

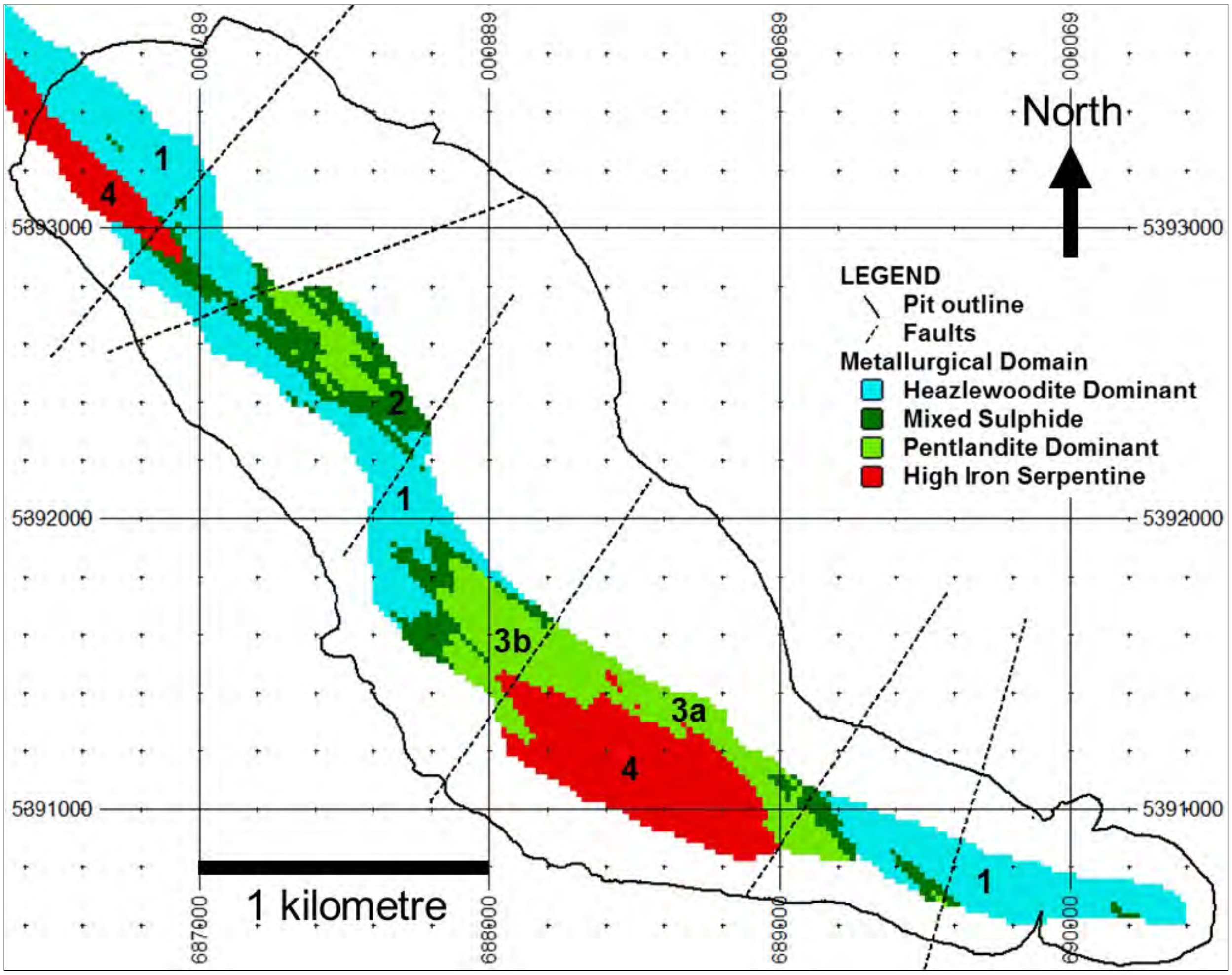
Figure 7.20: Early & Late Stage Serpentinization Features



Serpentine	73.68
Serpentine(Fe)	3.96
Talc	0.00
Orthopyroxene	0.00
Clinopyroxene	0.01
Olivine	8.59
Quartz	0.00
Feldspars	0.00
Chlorite	0.06
Mica/Clays	0.00
Magnetite	5.90
Brucite	3.67
Coalingite	0.53
Cr_Minerals	0.14
Ilmenite/Rutile	0.00
Pentlandite	1.58
Awaruite	0.03
Heazlewoodite	1.47
Millerite	0.14
Cu-Sulphides	0.03
Fe-Sulphides	0.00
Native Copper	0.00
Sphalerite	0.00
Apatite	0.00
Calcite	0.07
Dolomite	0.13
Gibbsite	0.00
Other	0.00
Total	100.00

Note: False-colour EXPLOMIN^{TM} field stitch image (EXP_214). (Heazlewoodite to pentlandite ratio 1.01). Modal Abundances as reported from EXPLOMIN^{TM}: 1.58% Pn, 1.6% Hz, 0.03% Aw, 0.11% Millerite, 8.5% Olivine, 4% Iron Serpentine. Sample contains relict olivine centres stretched along a directions fabric. Relict olivine (an early stage serpentinization feature) is juxtaposed against Mg serpentine (missing intermediary Fe-serpentine phase) along with coarse intercumulus magnetite blebs, intimately associated with heazlewoodite, pentlandite and millerite which are late stage features. Pseudomorphed olivine grains are encircled with fine magnetite-brucite rims. **Source:** RNC.

Figure 7.21: Modelled Distributions of Serpentinization Strengths & Associated Mineralogy



Note: Serpentinization zones which are analogous to metallurgical domains correspond to letters A to G descriptions in Figure 7.19. (1) G, heazlewoodite dominant, fully serpentinized +/- awaruite (metallurgical domain heazlewoodite dominant, Hz/Pn>=5, SPFE<14, (2) E to G, low iron serpentine, mixed sulphide, pentlandite and heazlewoodite +/- awaruite (metallurgical domain: mixed sulphide,1<HZ/PN<5, SPFE<14, (3a) C-D , low –moderate iron serpentine, pentlandite dominate commonly with coarse awaruite (3b) D-E low iron serpentine, pentlandite dominate +/- awaruite (metallurgical domain: Pentlandite Dominant HZPN<=1, SPFE<14 (4)A&B pentlandite dominate with high iron serpentine +/- relict olivine (Metallurgical domain high iron serpentine SPFE>14 . Note that “H” is not displayed because it does not corresponds to broad zones but is restricted locally to large fault zones and the basal contact (which is found) outside of the mineralization envelope. (Note: Hz/Pn is the heazlewoodite to pentlandite ratio, Hz+Pn is the sum of the modal abundance of heazlewoodite and pentlandite and SPFE is high iron serpentine). Block model intersection at 237.5 metre elevation shown. **Source:** RNC.

Table 7-6: Awaruite Sample Populations for Non-Sulphide Samples

Domain	% Awaruite Average for non-sulphide samples by EXPLOMIN TM Modal Abundance
Heazlewoodite Dominant	0.08
Mixed Sulphide	0.13
Pentlandite Dominant	0.17
High Iron Serpentine	0.08

Note: "Domains" refers to serpentinization domains described here. Awaruite modal abundances are reported from mineralogical sampling program. Non-sulphide samples have sulphur<0.07% **Source: RNC.**

7.4 Contact-type Nickel-Copper-PGE Mineralization

Magmatic nickel-copper-platinum group element (PGE) analyses were not performed during the initial drilling program that defined the Dumont deposit in the early seventies. In 1987, a drilling program (Oswald, 1988) was conducted to test the sill contacts for platinum and palladium at two locations. The best intersection from this program was drill hole 87-7, located in the east near drill hole E-7, inside and adjacent to the sill contact. This drill hole graded 0.61% nickel, 0.10% copper, 190 ppb palladium and 900 ppb palladium over 6.4 m. Drill holes 87-12 to 14 in the main zone did not reach the contact.

Drilling by RNC has confirmed the occurrence and grade of the historically identified mineralization at the basal contact at the eastern end of the Dumont sill. Drill hole 08-RN-71 intersected 0.8 m of semi-massive pyrrhotite grading 0.99% nickel, 0.19% copper, 0.3 g/t platinum, 1.0 g/t palladium and 0.07 g/t gold at the contact between the Dumont intrusive and footwall volcanics.

7.5 2011 Discovery of Massive Sulphides at Basal Contact

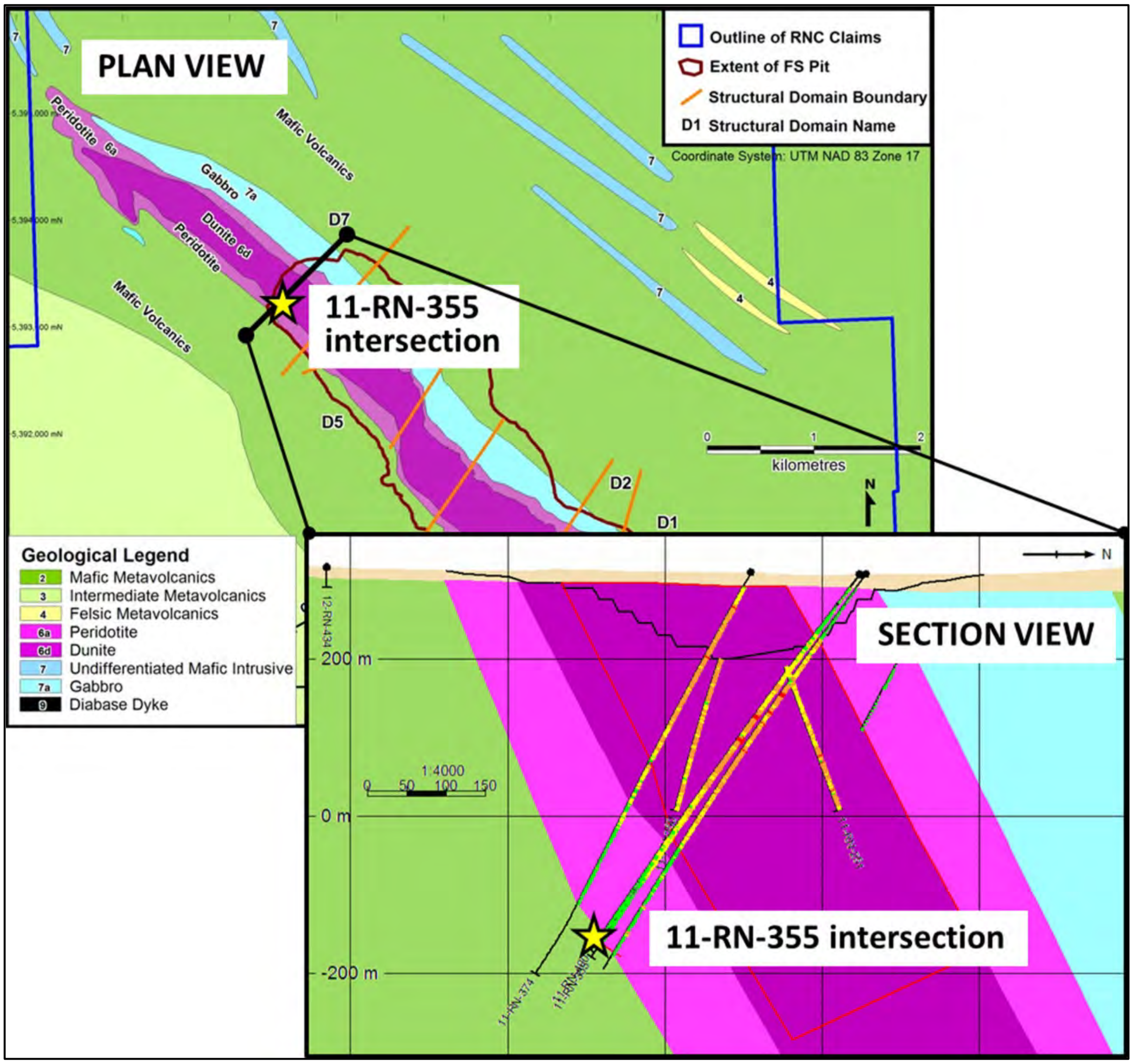
A hole drilled on section 5500E, passing through the Dumont intrusion and penetrating the footwall contact between the peridotite and the footwall mafic volcanic rock just to the northwest of the FS pit intersected a 1.25 m core-length of massive sulphide mineralization (Figure 7.22). The massive sulphide was composed of >60% sulphides containing primarily pyrrhotite with up to 10% centimetre-scale pentlandite crystals and trace chalcopyrite. Assuming that this massive sulphide body is coplanar with the footwall contact (dipping 65˚ toward 025˚ azimuth), the true thickness of the mineralization would be 1.07 m. Borehole geophysical surveying (electromagnetic) and follow-up drilling have not defined any significant extent to this mineralization to date.

Table 7-7: Assay Results for the Massive Sulphide Interval in 11-RN-355

From (m)	To (m)	Interval (m)	Palladium (ppm)	Platinum (ppm)	Sulphur %	Nickel %	Specific Gravity
572.95	573.55	0.60	3.26	1.94	38.8	4.25	4.79
573.55	574.20	0.65	3.75	2.15	38.1	4.49	4.80

Source: RNC.

Figure 7.22: Plan & section view of massive sulphide interval in drill hole 11-RN-355



Source: RNC.

This is the first time that such elevated concentrations of sulphides with high metal grades have been encountered anywhere in the Dumont intrusion. This discovery demonstrates that mineralizing processes capable of producing high-grade massive sulphide mineralization have operated, at least locally, within the Dumont setting, particularly at the basal contact of the intrusion. Further work will focus on following up this intersection and on developing exploration vectors to explore the rest of the 7.5 km long basal contact for similar occurrences.

7.6 Other Types of PGE Mineralization

RNC's drilling has further delineated three anomalous PGE horizons other than the basal contact type described above. In 2008, a PGE horizon associated with the pyroxenite layer overlying the upper peridotite was identified. This zone varies in thickness from 0.4 to 51 m with grades ranging 0.08 to 1.46 g/t platinum, and 0.04 to 2.39 g/t palladium. The second PGE horizon, lies under the main sulphide body, was previously identified during research on the historical drilling (Brügmann, 1990). This zone ranges from 0.4 to 34.5 m thick with grades ranging from 0.1 to 1.4% nickel, trace to 0.75 g/t platinum, and trace to 0.2 g/t palladium. The third PGE horizon was discovered by RNC in 2008 and is located approximately 100 m below the lowest sulphide body near the dunite contact with the lower peridotite. This horizon ranges from 1.0 to 140 m thick with grades ranging from 0.1 to 0.5% nickel, trace to 0.9 g/t platinum, and trace to 2 g/t palladium. These horizons generally are observed to be continuous along strike and dip where drilling is present. Samples from each PGE horizon were sent to Memorial University for analysis using scanning electron microscope. This work identified that the PGE phases are similar in all horizons and consist of three alloys: palladium/tin (Pd/Sn), platinum/copper (Pt/Cu), and platinum/nickel (Pt/Nickel) which are intimately associated with nickel sulphides.

7.7 Metallurgical Domaining of Nickel Mineralization

Sections 7.1 and 7.2 describe the geological controls on nickel mineralization assemblages and their distribution as well as the controls on the nickel content of the pay and gangue minerals. Metallurgical test results (Section 13) show a clear correlation between mineralogical variations related to degree of serpentinization (described in Section 7.2.2 and illustrated in Figure 7.19) and metallurgical recovery of nickel. Four metallurgical domains have therefore been established that correspond to these serpentinization domains. They are defined mineralogically on the basis of heazlewoodite to pentlandite ratio (Hz/Pn) and iron-rich serpentine abundance as follows:

- Heazlewoodite Dominant Domain: Samples with heazlewoodite to pentlandite ratios (Hz/Pn) greater than 5, and contain an iron rich serpentine abundance less than 14% are considered to be heazlewoodite dominant (Figure 7.6).
- Mixed Sulphide Domain: Samples having a heazlewoodite to pentlandite ratio between 1 and 5, and contain an iron rich serpentine abundance less than 14% are considered to be a combination of heazlewoodite and pentlandite (Figure 7.9).
- Pentlandite Dominant Domain: Samples with heazlewoodite to pentlandite ratios less than 1, and contain an iron rich serpentine abundance less than 14% are considered to be pentlandite dominant (Figure 7.7).
- High Iron Serpentine Domain: Samples that contain more than 14% iron rich serpentine. (FESP) as shown in Table 7-8.

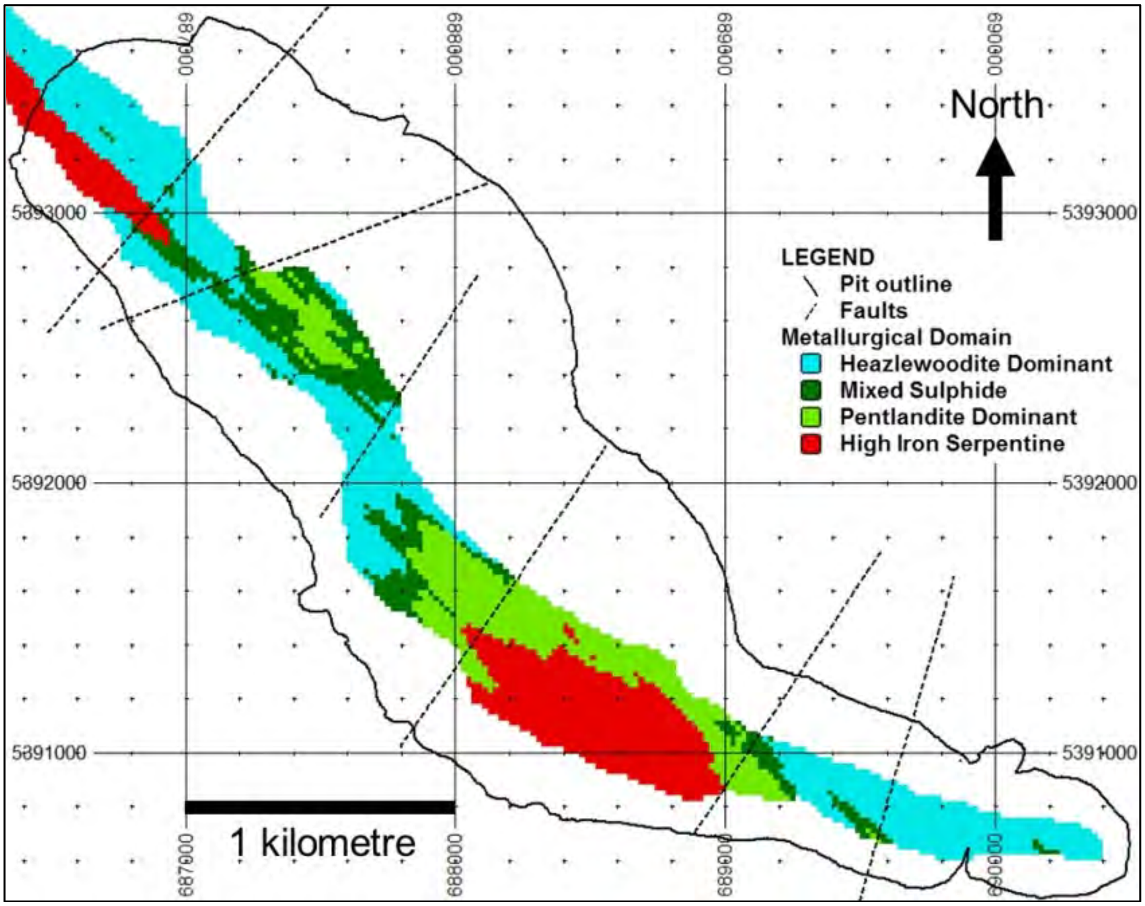
Pentlandite dominant samples (Hz/Pn<1, FESP<14) are most common making up a significant proportion of the reserve.

Table 7-8 gives the abundance of each metallurgical domain as calculated from estimated mineral abundances in the resource block model for the measured and indicated resource within the pre-feasibility pit. Figure 7.23 shows the distribution of these domains within the Dumont deposit.

Table 7-8: Proportion of Reserve in Each Metallurgical Domain

	Average Nickel Grade (%)	M&I Resource (Mt)
Total in-situ reserve	0.27	1,179
Heazlewoodite Dominant (Hz/Pn>=5, FESP<14)	0.25	348
Mixed Sulphides (1<Hz/Pn<5, FESP<14)	0.27	223
Pentlandite Dominant (Hz/Pn<=1, FESP<14)	0.29	358
High Iron Serpentine FESP>=14)	0.27	250

Figure 7.23: Distribution of Metallurgical Domains in Block Model



Note: Block model intersection at 237.5 metre elevation shown. **Source:** RNC.

8 DEPOSIT TYPES

Magmatic nickel-copper-platinum group element (PGE) deposits occur as sulphide concentrations associated with a variety of mafic and ultramafic magmatic rocks. The magmas originate in the upper mantle, and an immiscible sulphide phase occasionally separates from the magma as a result of the processes occurring during emplacement into the crust. The sulphide phase generally partitions and concentrates nickel, copper and PGE elements from the surrounding magma. The heavy sulphide droplets once concentrated and separated from the magma tend to sink towards the base of the magma, and form concentrated pockets or layers of sulphides that crystallize upon cooling to form mineral deposits.

The Dumont mineral deposit comprises olivine + sulphide cumulates that comprise differentiated layers of the Dumont sill, an Archean komatiitic intrusion contained within the Archean Abitibi Greenstone Belt of northwestern Quebec. As such, it is usually classified (Naldrett, 1989) with its most analogous counterpart, the Mt. Keith mineral deposit located in the Agnew-Wiluna Greenstone Belt within the Archean Yilgarn craton of West Australia.

Greenstone belts are typical terranes found in many Archean cratons, and may represent intracratonic rift zones. The greenstone belts are generally composed of strongly folded, basaltic/andesitic volcanics and related sills, siliciclastic sediments, and granitoid intrusions that have been metamorphosed to greenschist and amphibolite facies, and typically adjoin tonalitic gneiss terranes. Komatiitic rocks form an integral part of some of these greenstone belts.

Both the Dumont and Mt. Keith deposits have undergone pervasive serpentinization and local talc-carbonate alteration due to metamorphism to mid-upper greenschist facies. This alteration history has resulted in liberation of much of the nickel from nickel silicates (olivine) and consequent upgrading of the primary magmatic nickel-sulphide and nickel-alloy minerals through partitioning of nickel. However, the Dumont deposit is differentiated from the Mt. Keith deposit by the abundance of the nickel-iron alloy awaruite and by the restricted extent of talc-carbonate alteration, which is limited to the basal contact of the intrusion and occurs outside the resource envelope. Also, the Dumont deposit has not been subjected to the extensive supergene weathering alteration present at Mt. Keith.

9 EXPLORATION

Exploration for nickel mineralization on the Dumont property has been completed primarily by diamond drilling due to the lack of outcrop over the ultramafic portions of the Dumont intrusive which host the nickel mineralization. This drilling was initially targeted using data from historical drilling and airborne electromagnetic and magnetic surveys. Drilling programs and results are described in Section 10.

No continuous trench samples were taken from the Dumont deposit. Non-drilling exploration work carried out on the Dumont property is described below.

9.1 Geophysics

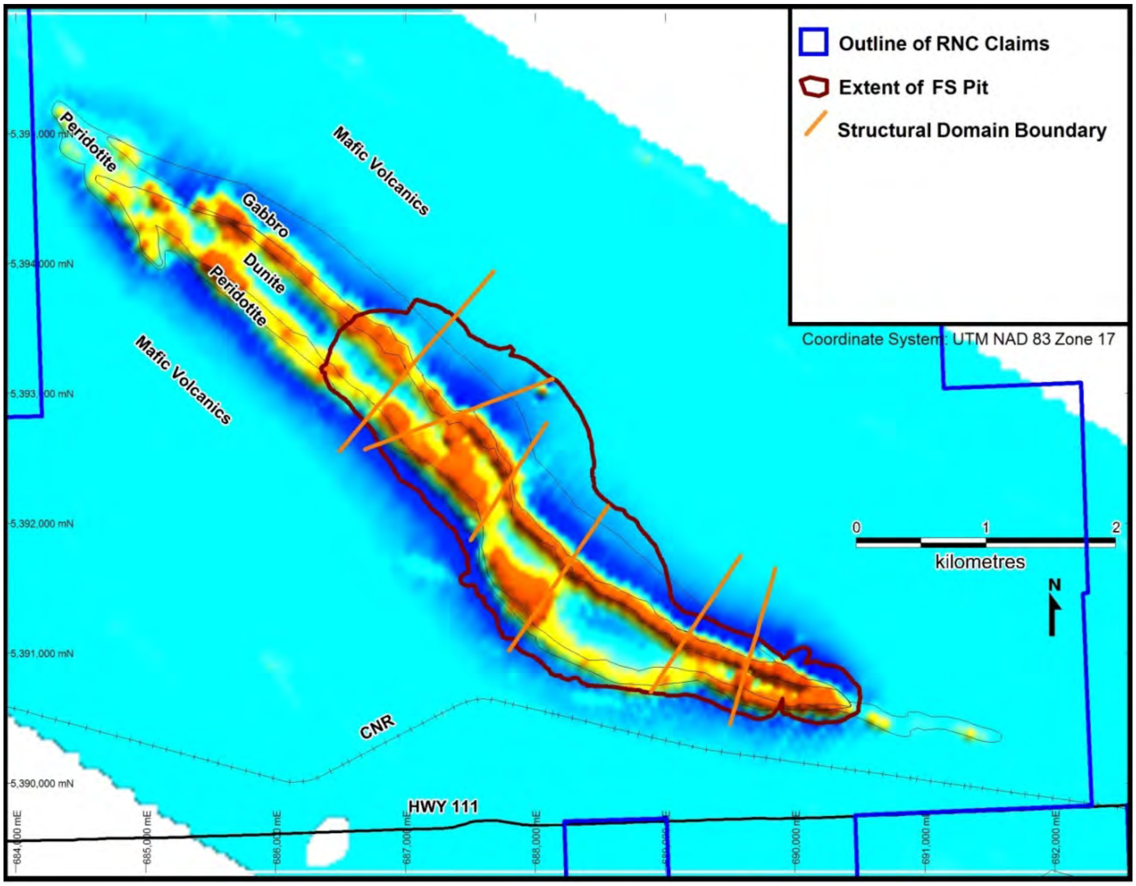
9.1.1 Airborne Geophysics

A helicopter-borne versatile time domain electromagnetic (VTEM) and magnetometer survey was completed by Geotech Ltd. over the Dumont intrusive and adjacent areas at 100 metre line spacing in 2007 as follow up to an earlier helicopter-borne magnetometer-only survey conducted by Geophysics GPR International Inc. in February 2007. Figure 9.1 shows a gridded plot of the first vertical derivative of total magnetic intensity.

The magnetic survey has outlined the limits of the Dumont sill which exhibits a strong contrast between its magnetic susceptibility and that of the surrounding country rocks. The survey has also defined stratiform bands of varying magnetic intensity which reflect varying magnetite content within these rocks which is related to the igneous layering within the sill and to varying degrees of serpentinization within a given layer. The magnetic pattern also allows the interpretation of major structures that cross-cut the intrusion.

The VTEM survey detected several weak electromagnetic anomalies along the footwall contact of the Dumont intrusive. Several of these anomalies were drill-tested. Anomalies tested to date were primarily due to barren pyritic interflow sediments within the footwall volcanic.

Figure 9.1: First Vertical Derivative Magnetics Map of Dumont Property



Source: RNC.

9.1.2 Ground & Drill hole Geophysics

In February 2013, a ground time-domain electromagnetic survey was completed over a portion of the footwall of the Dumont intrusion. The purpose of this survey was to evaluate the potential for massive sulphide similar to the occurrence intersected in drill hole 11-RN-355 (see Section 7.5) in an orientation subparallel to the basal contact of the intrusion. A 100-metre spaced grid was established between lines 5300E and 7000E and an InfinTEM time-domain electromagnetic survey was completed over the grid. Interpretation of the results indicated weak to moderate large-scale conductive horizons coincident with the footwall contact, but did not indicate discrete conductors consistent with significant accumulations of massive nickel sulphides. These results are consistent with results from drill hole geophysical surveys (UTEM time domain electromagnetics) conducted on several drill holes in the vicinity of hole 11-RN–355 from September to November 2011. Follow-up work will consist of conducting another survey using a loop configuration that will evaluate potential massive sulphide accumulations controlled by cross-cutting structures.

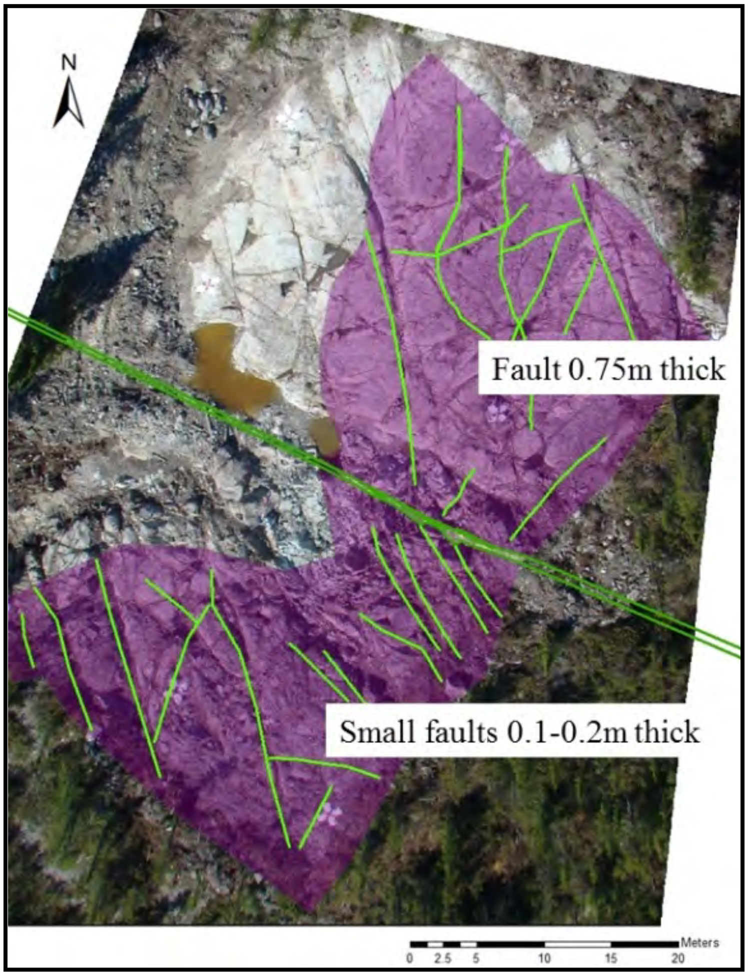
9.2 Geological Mapping

Surface mapping programs have been carried out over the Dumont property, primarily to provide a structural geology framework for the modelling of the Dumont deposit.

Several geological mapping programs have been completed over the Dumont property beginning in the summer of 2008. Given the poor exposure over the Dumont sill, the mapping programs have focused on outcrops in the country rocks outside the sill, in order to gain an understanding on the local structural geology. A secondary purpose for these programs has been to identify outcrop in areas of potential mining infrastructure development. Information collected during these programs was interpreted in association with airborne magnetics and LIDAR topography data and was used to update historic geological maps and to provide constraints for subsurface fault modelling. Outcrop locations were also used to assist in modelling of the bedrock surface and overburden thickness.

In 2012, detailed structural mapping of several outcrops, including the 57 m x 27 m exposure of dunite cleared for the purpose of bulk sampling described in Section 9.4 was completed in support of the structural modelling of the deposit (Fedorowich, 2012). A structural mapping example from the outcrop bulk sample location is shown in Figure 9.2 and the location is labelled in Figure 9.3 as “Outcrop Bulk Sample Location.”

Figure 9.2: Aerial View of the Outcrop Bulk Sample Location with Outline of Exposed Dunite & Fault Traces



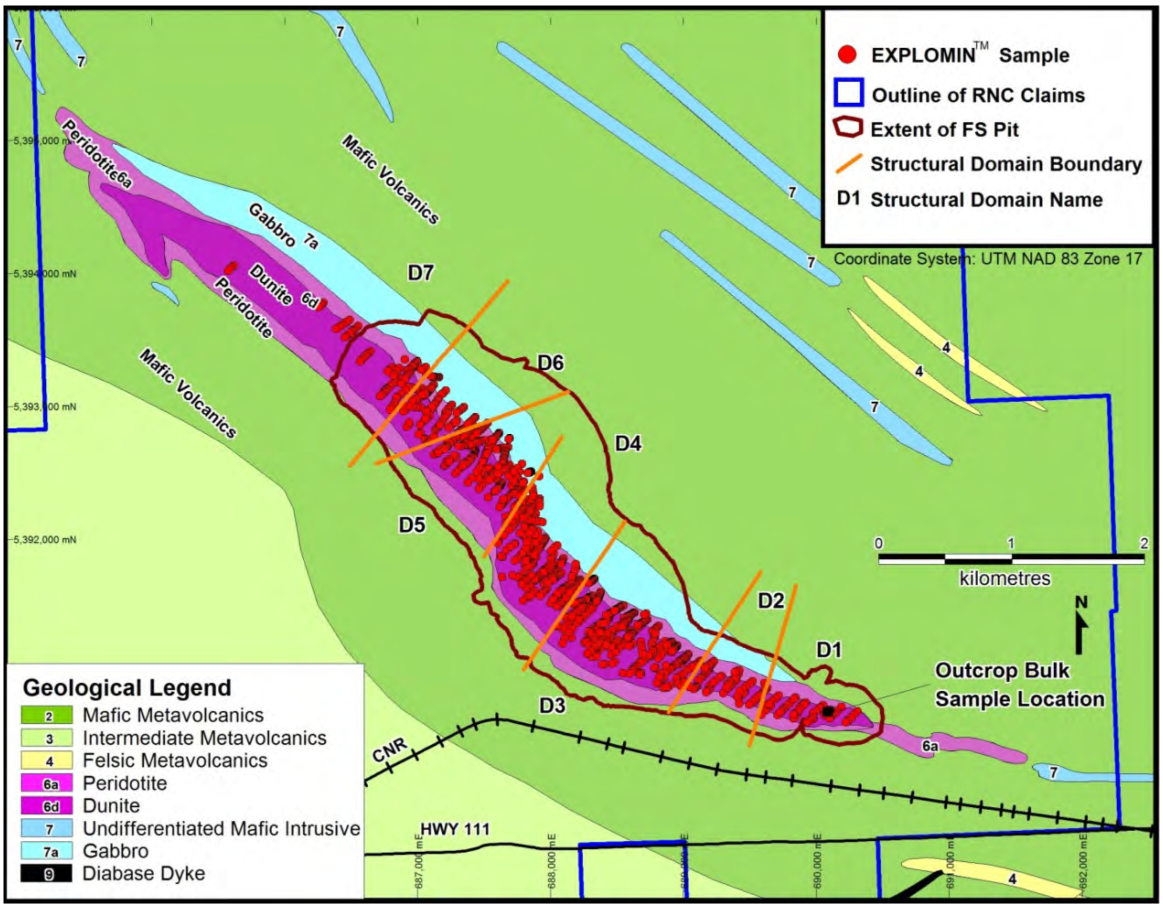
Source: Itasca Consulting. Note that the image is best fit, and not a true orthophoto, therefore there are mismatches.

9.3 Mineralogical Sampling

Mineralogical sampling of Dumont core began in 2009. The mineralogical sampling program uses the SGS Minerals Services' EXPLOMIN™ analysis to provide detailed mineralogical information on mineral assemblages, nickel deportment, liberation, alteration and the variability of these factors. Mineralogical samples were taken for the purpose of metallurgical domain composite characterization and for the purpose of mineralogical mapping of the Dumont deposit.

Mineralogical mapping sample locations were planned so as to provide spatially and compositionally representative data down drill hole traces for holes on even numbered sections along the length of the deposit as shown in Figure 9.3, with the goal of providing comprehensive representation of the mineralogical variability of the deposit. A total of 1561 mineralogical mapping samples were collected as of 25 November 2012, 1420 of which occur within the mineralized envelope and were used for mineralogical modelling of the deposit as described in Section 14.

Figure 9.3: Location of Mineralogical Samples



Source: RNC.

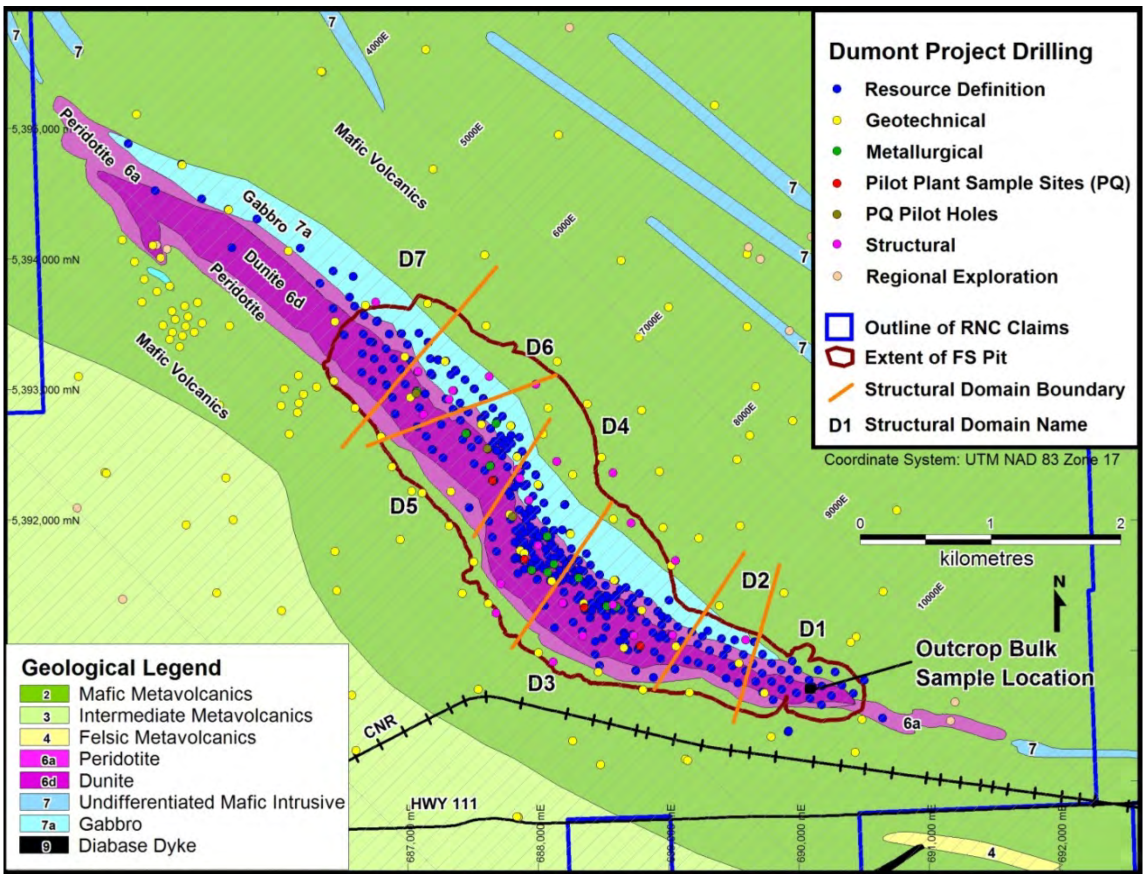
Metallurgical domain composite characterization samples were selected on an ongoing basis to represent the mineralogy of each metallurgical domain composite as defined for testwork. This includes all domain composites described in Section 13, as well as all metallurgical composites defined in the mini pilot plant test (PQ) drill holes.

The sampling and analytical procedures for both types of samples are identical and described in Section 11.1.2.

9.4 Outcrop Bulk Sampling

In the spring of 2011 a mineralized serpentinized dunite outcrop located in the eastern portion of the deposit on line 9850E was prepared for bulk sampling (Figure 9.4). Nickel mineralization in the sampled portion of the outcrop is dominated by heazlewoodite.

Figure 9.4: Map Showing Outcrop Bulk Sample Location



Source: RNC.

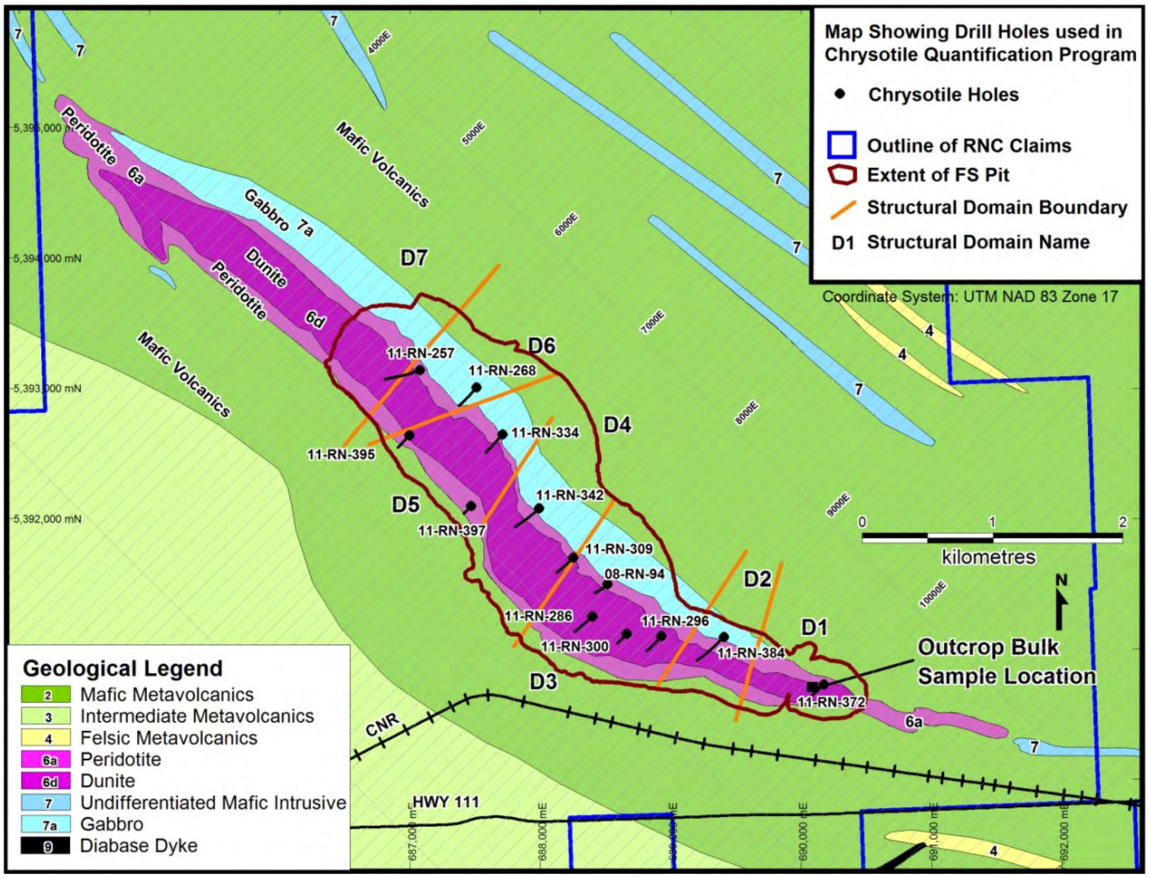
A section of the outcrop measuring approximately 40 m x 55 m was cleared of glacial overburden with an excavator and power washed. A smaller area within this was identified for sampling and subsequently drilled and blasted to a depth of approximately 1.5 m.

Approximately 100 tonnes of this material was used in the in-situ environmental geochemistry characterization cells described in Section 20. Approximately 3 tonnes of this material were used for metallurgical testing as described in Section 13.

9.5 Chrysotile Quantification

A logging program to quantify the bulk chrysotile content of dunite and peridotite from the Dumont deposit was completed from January to March 2013. This program involved relogging a representative sample of 13 holes (Figure 9.5). RNC has developed a standard logging procedure for the quantitative visual estimation of chrysotile in drill core. This method has been validated by independent external experts (Verschelden and Jourdain, 2013) and provides reproducible and quantifiable results and is described in Section 11.1.7. Results of the determinations are presented in Tables 9-1 and 9-2 (overleaf). Table 9-2 reveals that the 95% confidence interval for the average bulk chrysotile content for dunite and peridotite is between 1.6% and 1.9%.

Figure 9.5: Map Showing Drill Holes used in Chrysotile Quantification Program



Source: RNC.

Table 9-1: Chrysotile Quantification Results

Hole	Upper Peridotite	Dunite	Lower Peridotite	Subtotal of Rock	Non Recovered Core	TOTAL
11-RN-372	91.9 m	152.8 m	20.6 m	265.3 m	3.6 m	268.8 m
	1.9%	0.9%	0.6%	1.2%	8.75%	1.3%
11-RN-384	178.0 m	41.5 m	177.8 m	397.3 m	7.7 m	405.0 m
	1.6%	3.4%	1.5%	1.7%	8.75%	1.9%
08-RN-94	100.7 m	351.2 m	0 m	451.8 m	0 m	451.8 m
	1.9%	1.7%		1.7%	0%	1.7%
11-RN-286	0 m	221.6 m	0 m	221.6 m	1.8 m	223.4 m
		1.2%		1.2%	8.75%	1.3%
11-RN-300	0 m	226.6 m	0 m	226.6 m	1.4 m	228.0 m
		1.1%		1.1%	8.75%	1.1%
11-RN-296	0 m	258.5 m	86.1 m	344.6 m	4.4 m	349.0 m
		1.1%	1.5%	1.2%	8.75%	1.3%
11-RN-342	110.2 m	247.1 m	0 m	357.2 m	0 m	357.2 m
	1.8%	2.5%		2.3%	0%	2.3%
11-RN-309	125.2 m	362.8 m	0 m	488.0 m	7.0 m	495.0 m
	1.7%	2.6%		2.4%	8.75%	2.5%
11-RN-334	104.4 m	204.9 m	0 m	309.2 m	10.8 m	320.0 m
	1.3%	3.1%		2.5%	8.75%	2.7%
11-RN-395	0 m	0 m	83.8 m	83.8 m	1.1 m	84.9 m
			0.8%	0.8%	8.75%	0.9%
11-RN-397	0 m	0 m	46.4 m	46.4 m	1.5 m	47.9 m
			0.5%	0.5%	8.75%	0.7%
11-RN-268	152.7 m	273.7 m	0 m	426.4 m	8.2 m	434.6 m
	1.8%	1.7%		1.7%	8.75%	1.9%
11-RN-257	0 m	380.6 m	65.4 m	446.0	14.8 m	460.8 m
		1.2%	0.8%	1.2%	8.75%	1.4%
Total Length by Lithology	**862.9 m**	**2721.3 m**	**480.1 m**	**4064.3 m**	**62.2 m**	**4126.4 m**
Average % by Lithology	**1.7%**	**1.8%**	**1.1%**	**1.7**	**8.75%**	**1.8%**

Table 9-2: Chrysotile Quantification Percentages obtained over the Dataset & Sorted by Lithology

Lithologies	Weighted Average	Standard Deviation	Number of Values	95^{th} Percentile	95% Confidence Interval	Tolerance Limit 95% for the Mean	
						Lower	Upper
Upper Peridotite	**1.7**	1.1	294	3.8	3.5	**1.6**	**1.8**
Dunite	**1.8**	2.0	917	4.9	5.1	**1.6**	**1.9**
Lower Peridotite	**1.1**	1.5	166	3.3	3.5	**0.9**	**1.4**
Entire project	**1.7**	1.8	1 377	4.4	4.6	**1.6**	**1.8**

10 DRILLING

Upon acquiring the Dumont property, RNC conducted an initial exploration drilling program which consisted of 5 twin holes to confirm the historic drilling results in 2007. Results from this drilling campaign confirmed the historical drilling results and encouraged RNC to embark on an extensive drilling campaign to fully evaluate the Dumont deposit. RNC has since conducted core diamond drilling on the Dumont property for the purposes of exploration, resource definition, metallurgical sampling and bedrock geotechnical investigation. RNC has also conducted core drilling and cone penetration testing for the purpose of overburden geotechnical characterization. A summary of the drilling conducted on the property since 2007 is given in Table 10-1. Figure 10.1 illustrates the location of all diamond drill and sonic holes completed by RNC on the Dumont property classified by type, and Figure 10.2 illustrates the location of all diamond drill and sonic holes completed by RNC on the Dumont property classified by year of drilling. Figure 10.3 illustrates the locations of all overburden testing sites.

No continuous trench samples were taken from the Dumont deposit.

RNC contracted Forages M. Rouillier (Rouillier) of Amos, Quebec to conduct core diamond drilling. Rouillier used custom built diamond drill rigs mounted on skids or self-propelled tracked vehicles with NQ diameter diamond drill coring tools. On occasion, HQ and PQ diameter core was drilled. Rouillier is an independent diamond drilling contractor that holds no interest in RNC.

For the purpose of establishing sections and for easy location reference in the context of the strike of the deposit, a local grid coordinate system has been established with a baseline approximately parallel to the strike of the Dumont sill and the general trend of the mineralized zones. Grid lines are oriented at an azimuth of 045° and the origin of the grid (grid coordinates 0E, 0N) is located at UTM NAD83 Zone 17 coordinates 678,160E, 5,392,714N. This grid was established for ease of reference and section plotting only and is shown in Figure 10.1. This is a virtual grid and no physical grid lines have been cut in the field. Drill collar coordinates continue to be recorded and reported in UTM NAD83 Zone 17 coordinates and drill hole directional data are recorded and reported relative to astronomic (true) north.

Drill hole directional surveys were conducted using a Maxibor down-hole survey tool which calculates the spatial coordinates along the drill hole path based on optical measurements of direction changes and gravimetric measurements of dip changes. Drill holes are subsequently subject to a differential global positioning system (DGPS) location and deviation surveys using a north-seeking gyro by a certified surveyor before integration of the drilling data into the resource estimation database. Core recovery is very good and is generally greater than 95% with no statistical difference along strike or by geological or metallurgical domain.

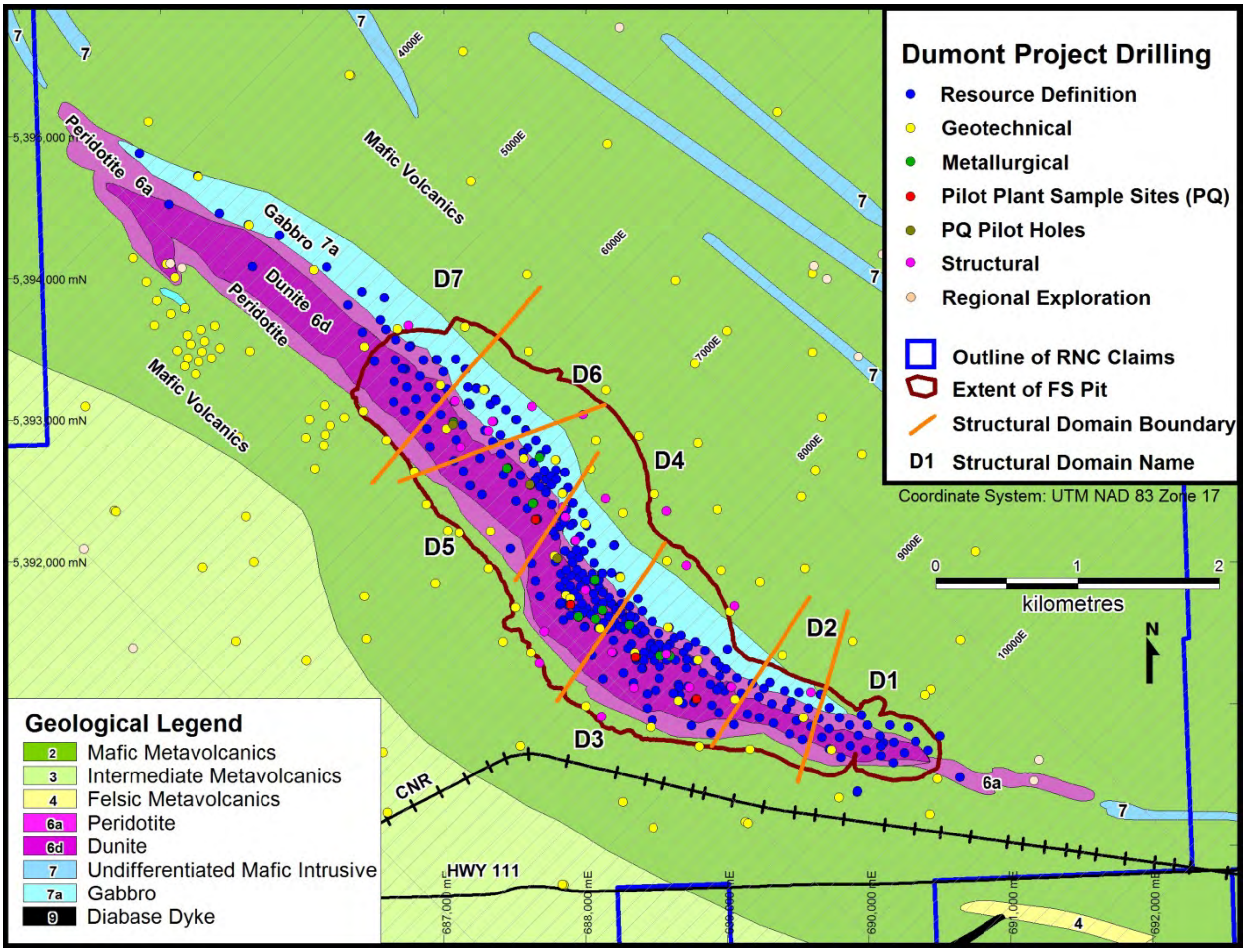
All geological, engineering and supervision portions of the drilling program were overseen by geological staff of RNC, principally Mr. John Korczak, P.Geo., Mr. Lorne Burden, P.Geo., and Mr Robert Cloutier, Geo., OGQ supervised by Mr. Alger St-Jean, P.Geo., Vice-President Exploration for RNC.

Table 10-1: Summary of Drilling Conducted on the Dumont Property

	2007 to 2010		2011		2012		2013		TOTAL	
	Number of Holes	Total Metres	Number of Holes	Total Metres	Number of Holes	Total Metres	Number of Holes	Total Metres	Number of Holes	Total Metres
Twin Hole	5	1,681							5	1,681
Sectional Resource Definition	216	86,986	157	56,527					373	143,513
Structural	4	1,359							4	1,359
Geotechnical (Bedrock)	3	1,503	13	6,503	35	5,387			51	13,393
Mini pilot plant Test Holes (NQ)	7	1,757							7	1,757
Total Drilling Included in the Current Resource Estimate									440	161,703
Metallurgical Domain Composites	10	3,194							10	3,194
Crushing Testwork Sample	3	406							3	406
Geotechnical (Overburden)	5	104	66	1,452	64	1,055			135	2,611
Mini Pilot Plant Sample (PQ)	13	2,774							13	2,774
Regional Exploration							13	3,392	13	3,392
TOTAL	266	99,764	236	64,482	99	6,442	13	3,392	614	174,080

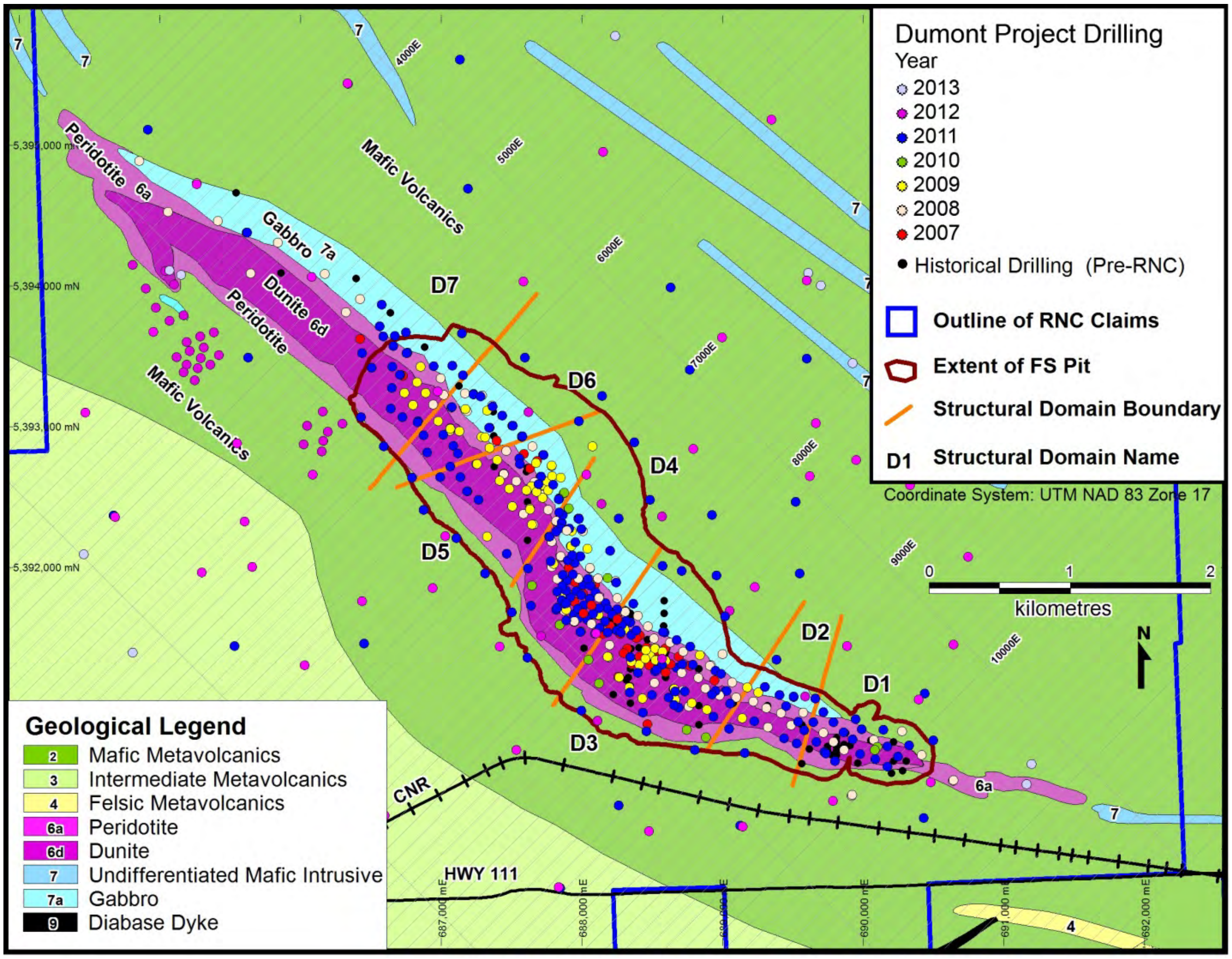
Source: RNC.

Figure 10.1: Location of Drill Holes on the Dumont Property



Source: RNC.

Figure 10.2: Drill Holes on the Dumont Property – Drilling Year



Source: RNC.

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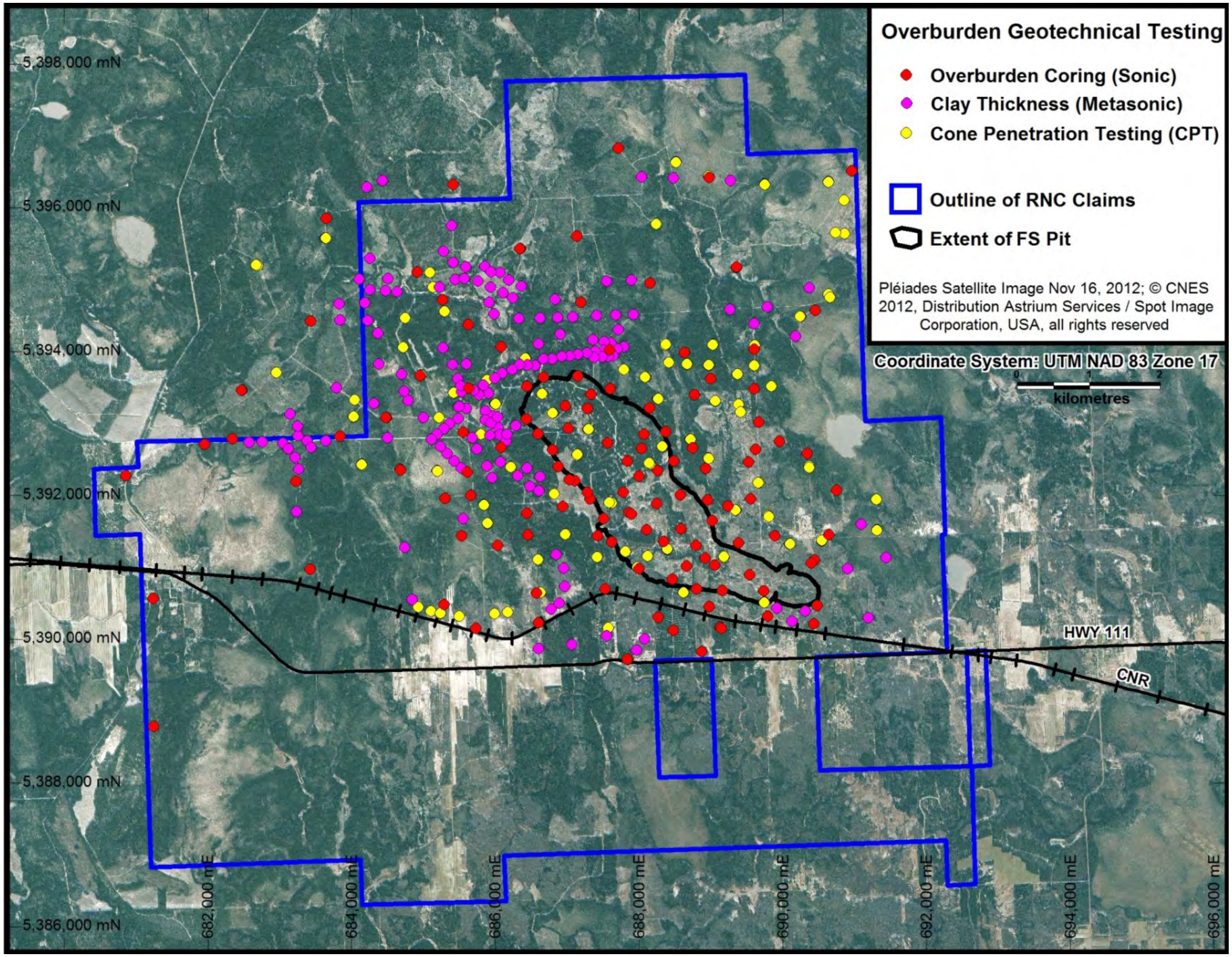
Figure 10.3: Overburden Drilling & Cone Penetration Test (CPT) Sites



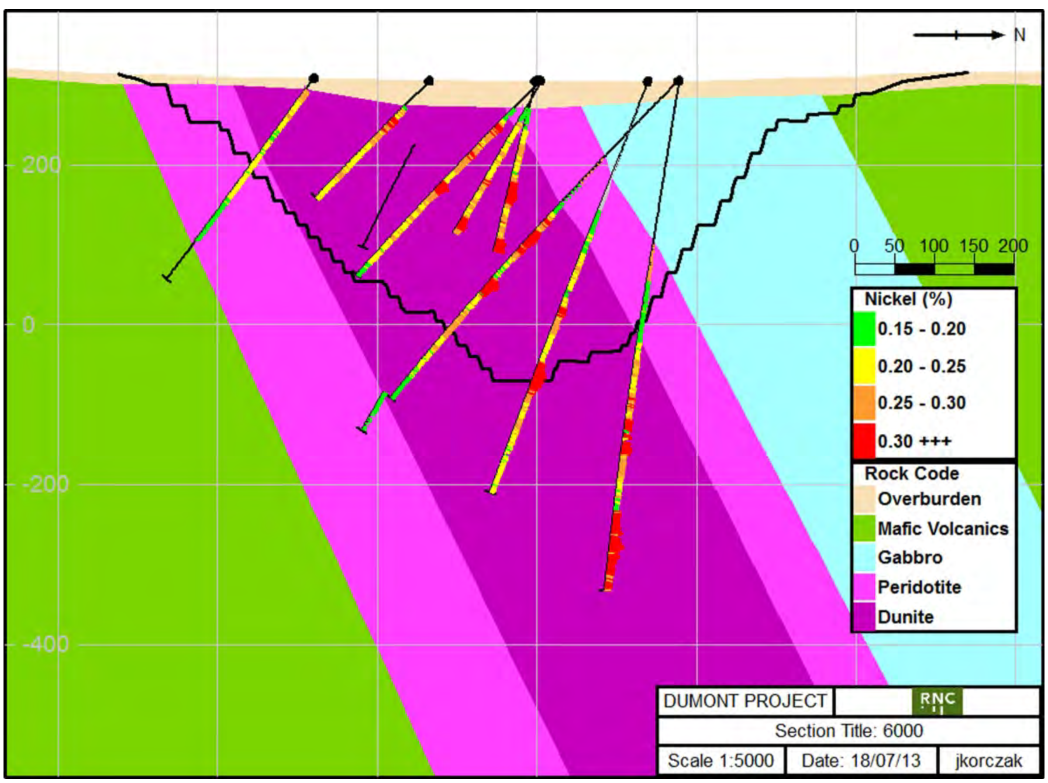
Source: RNC.

10.1 Resource Definition & Exploration Drilling

The sectional resource definition drilling program, initiated in 2007, was designed to maintain a nominal 100 m spacing between holes within the plane of the section and along strike between sections from section 5600E to Section 10000E. Drill spacing was decreased to 50 m by 50 m in two selected variability testing blocks centred on section 8250E and on section 6850E. Outside of the 10000E to 5600E range, exploration drilling was conducted along the trend of the Dumont intrusion, usually at wider spacing. Several exploration holes were drilled where conductive anomalies detected by the VTEM airborne geophysical survey conducted in 2007 coincided with the basal contact of the intrusion.

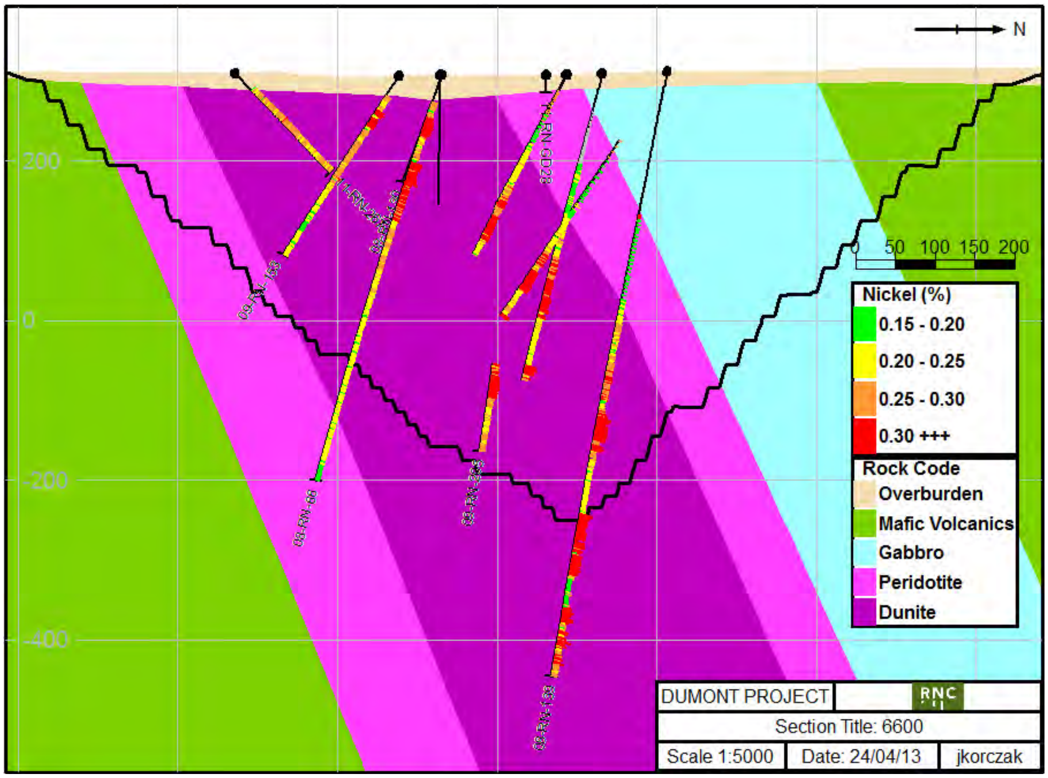
The program was designed to define mineralization down to a nominal depth of 500 m from surface (-200 m elevation). In places, drilling has investigated mineralization down to a depth of 700 m (-400 m elevation). Figure 10.1 illustrates the location of all holes completed during the sectional resource definition and exploration drilling program. Representative examples of drill sections through the Dumont deposit are given in Figure 10.4 (section 6000E), Figure 10.5 (section 6600E), Figure 10.6 (section 7600E), and Figure 10.7 (section 8350E). See Figure 10.1 for location of section lines. In general, the core recovery for the diamond drill holes on the Dumont property has been better than 95% and very little core loss due to poor drilling methods or procedures has been experienced. Core recovery does not vary along strike or by geological or metallurgical domain.

Figure 10.4: Drill Section 6000 E showing Outline of FS Pit



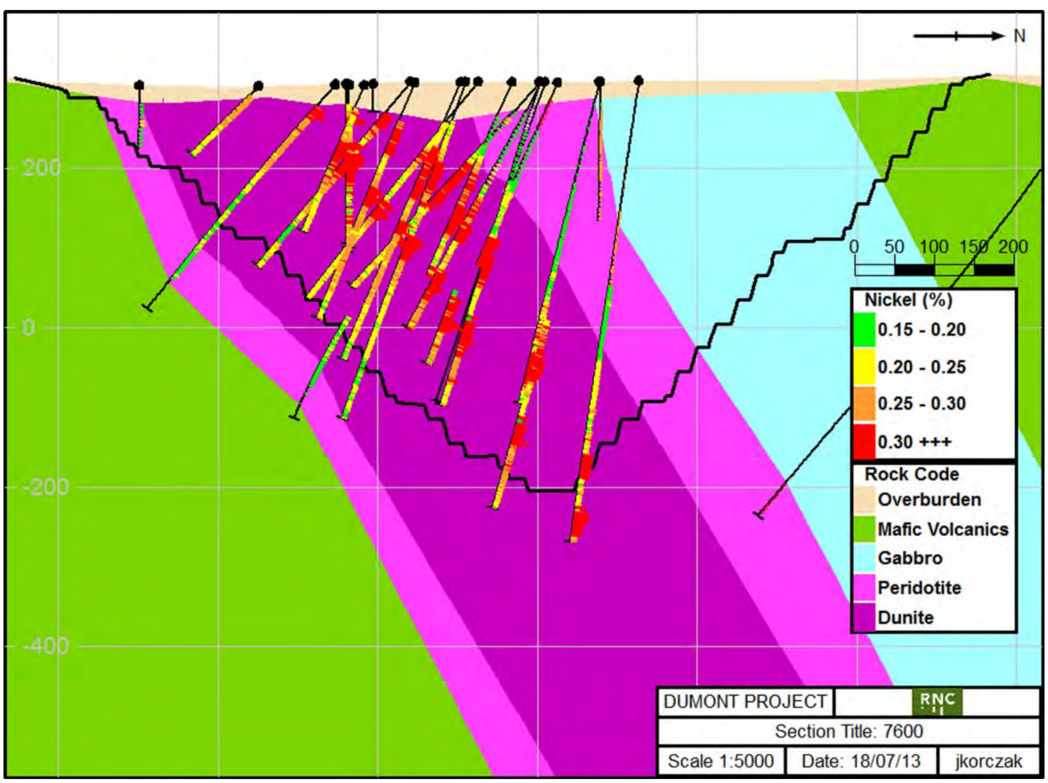
Source: RNC. Note that the scale is given in metres. Section shown is 100 m wide.

Figure 10.5: Drill Section 6600 E showing Outline of FS Pit



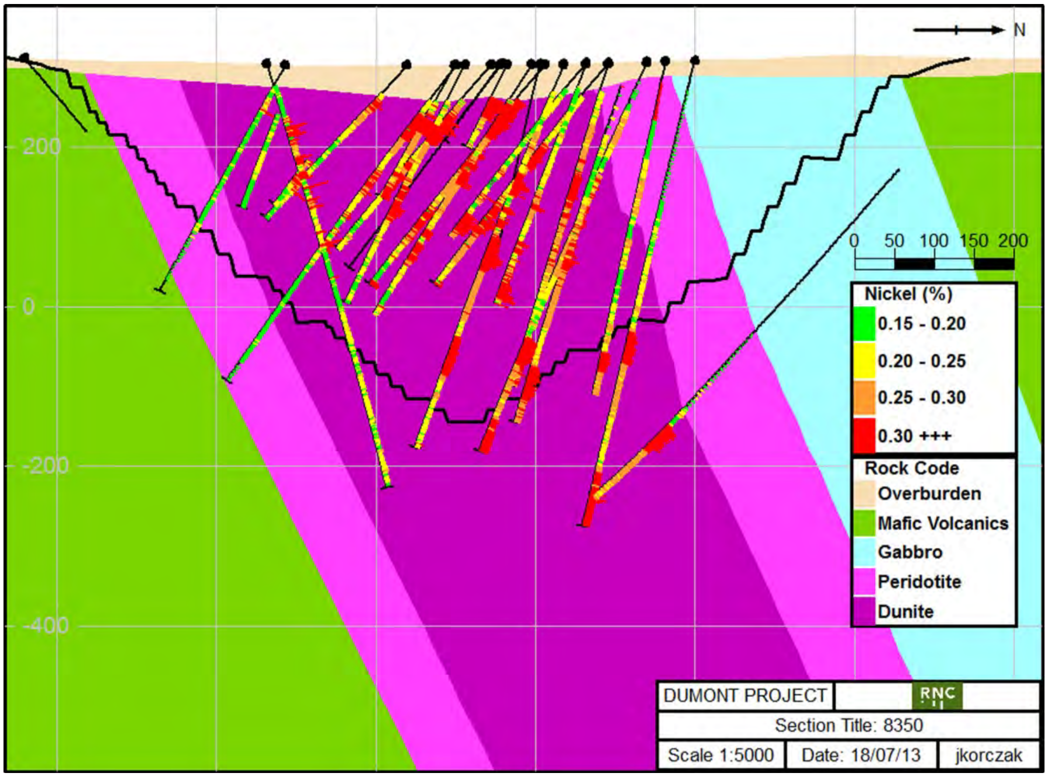
Source: RNC. Note that the scale is given in metres. Section shown is 100 m wide.

Figure 10.6: Drill Section 7600 E showing Outline of FS Pit



Source: RNC. Note that the scale is given in metres. Section shown is 100 m wide.

Figure 10.7: Drill Section 8350 E showing Outline of FS Pit



Source: RNC. Note that the scale is given in metres. Section shown is 100 m wide.

10.2 Structural Drilling

For the purpose of defining major geological structures (faults) in the central portion of the deposit, 1,359 m were drilled in 4 oriented core holes in 2009. These holes were drilled parallel to the strike of the deposit and at high angles to the major structures that cross-cut the deposit. Data from these structural holes were combined with the global drill hole database and surface mapping by John Fedorowich, Ph.D., P.Geo., of Itasca Consulting, to produce a first order structural model (Fedorowich, 2010) for the deposit that was used to delimit structural domains and help constrain the resource block model (see Section 9.2). Since 2009, several resource definition and exploration holes in zones of structural complexity have also been oriented to augment the structural model.

The structural model was revised and updated by SRK in 2011 (SRK Consulting Canada Inc., 2011) using oriented core data collected during the 2011 geotechnical drilling campaign (see Section 10.3). Itasca Consulting further updated the structural model using data collected during the 2012 geotechnical drilling campaign, data from detailed surface mapping, and regional geophysical surveys (Fedorowich, 2012).

10.3 Bedrock Geotechnical Drilling

In order to define rock mass characteristics and evaluate open-pit wall slope angles on an indicative basis, data collection for a preliminary geotechnical study was carried out in 2009. Work associated with this study included the measurement and analysis of 1,503 m of NQ size core from drilling 3 oriented core holes near section 6800E (GENIVAR, 2010b), and a limited hydrogeological study between sections 6500E and 7500E (GENIVAR, 2009b). This data helped define the open pit wall slope angles used in the Preliminary Assessment (Lewis et al, 2010).

Upon initiation of the pre-feasibility study, a geotechnical investigation program was designed by SRK and implemented by RNC staff under the supervision of SRK in 2011. The program consisted of 5,050 m of oriented HQ size core in 10 drill holes. Data from this drilling program was utilized by SRK in order to complete a pre-feasibility level geotechnical assessment for slope design as described in Section 16.2.1. The assessed parameters include rock quality designation (RQD), fracture frequency per metre (FF/m), empirical field estimates of intact rock strength (IRS), field (point load) and laboratory (uniaxial compressive and triaxial) strength, and RMR89 (Bieniawski, 1989). Hydraulic test data (49 packer tests) were also collected during this drilling program and used to map the distribution of bedrock hydraulic conductivity across the site and define bedrock hydrogeological domains.

An additional geotechnical investigation program designed by SRK was implemented by RNC staff under the supervision of SRK starting in December 2011 and was completed in May 2012. The program consisted of 6,163 m of oriented NQ size core in 11 drill holes. Data from this drilling program has been used by SRK to complete further FS level geotechnical assessment for slope design.

10.4 Overburden Geotechnical Drilling

Overburden geotechnical drilling was carried out in three phases. A limited overburden characterization program was carried as part of the preliminary evaluation in 2010. This was followed by a more extensive program of overburden coring by sonic drilling and cone penetration testing in support of the pre-feasibility study in 2011. Another more detailed program incorporating sonic drilling, cone penetration testing and metasonic probing to support feasibility level design work was completed in 2012. Locations of overburden geotechnical holes are shown in Figure 10.3.

10.4.1 Preliminary Overburden Characterization

The preliminary geotechnical (overburden) drilling program conducted in 2010 consisted of five holes totalling 104 m (GENIVAR, 2010c). This initial program was designed to characterize the overburden material located above the indicated resources in order to aid engineering work for the preliminary assessment. The program also allowed for the installation of three piezometers for groundwater measurements.

10.4.2 Sonic Drilling Program

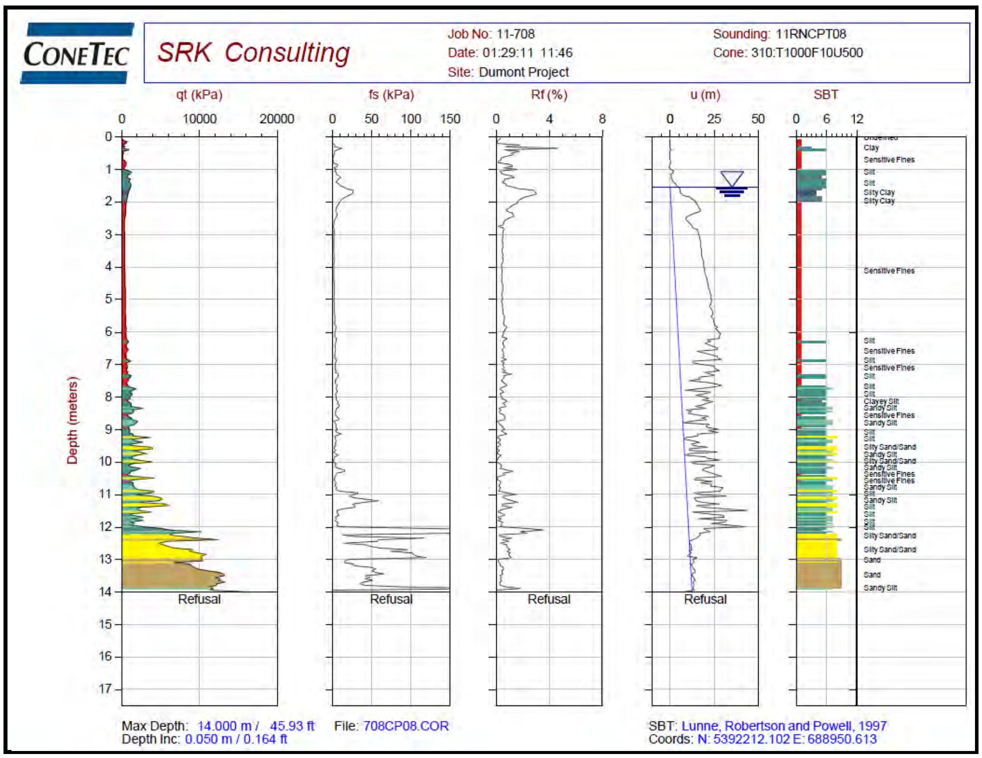
During the winter of 2011, drill holes were completed at 66 locations using a sonic drill rig which employs high frequency, resonate energy to advance the core barrel and casing into the ground. The drill hole plan and locations were strongly influenced by site accessibility, particularly in relation to areas outside the proposed open pit. Core recovery was high and was complemented by various field performance tests to evaluate the geotechnical properties. The groundwater level was measured immediately after completion of each drill hole and a number of monitoring and pumping wells were installed for future field permeability testing. An array of other laboratory tests were subsequently completed on select samples obtained during the drilling program. A drill hole log was prepared for each borehole and the laboratory test results are also included on the respective borehole log.

In the winter of 2012, additional drill holes were completed at 63 locations using a sonic drill rig. Data from this drilling program was incorporated into the geotechnical database and was used by SRK to complete FS level geotechnical assessments and infrastructure designs.

10.4.3 Cone Penetration Testing

During the winter of 2011, cone penetration testing (CPT) was undertaken at 62 probe hole locations using a track-mounted vehicle specifically built for CPT programs. The electronic piezocone measured parameters such as tip pressure, sleeve friction and porewater pressure every five centimetres as the cone was advanced into the ground. Pore pressure dissipation and seismic tests were carried out in select locations to provide additional information on soil characteristics. The CPT was terminated in each hole when the probe met refusal, which occurred in very dense soil or when bedrock was encountered. The CPT results from each probe hole are presented as a series of plots showing the tip pressure, sleeve friction and porewater pressure along with the interpreted soil profile of the hole as shown in Figure 10.8.

Figure 10.8: Example of CPT Results for Hole 11RNCPT08



Source: RNC.

Further to the initial program, in the winter of 2012, 80 additional CPT probe holes were completed. Data from this program was incorporated into the geotechnical database and was used by SRK to complete FS level geotechnical assessments and infrastructure designs.

10.4.4 Metasonic Probing

From June to November 2012, a metasonic probing program was carried out to evaluate the thickness of superficial glacial clay and silt deposits underlying the locations of proposed mine/mill infrastructure development (tailings storage facility, ore stockpiles, waste and overburden dumps, water reservoirs). The metasonic probe is a tool that vibrates NQ or BQ sized rods through soft unconsolidated sediments. The instrument was able to penetrate clay and silt layers through to refusal in a sand and/or gravel horizon at the base of the clay-silt sequence. At most probe locations a 1.5 m sample was taken when the instrument first penetrated into glacial clay and another at the end of the hole (refusal). Metasonic probing was completed at 153 sites as shown in Figure 10.3.

10.5 Metallurgical Drilling

10.5.1 General

Drilling was carried out in 2010 to collect samples for bench-scale metallurgical variability testing and crushing testwork. A total of 2,774 m of drilling in 13 holes was completed for metallurgical domain composite sampling, and 3 holes totalling 406 m were completed for crushing testwork. Additional metallurgical samples were taken from holes drilled as part of the sectional resource drilling program.

10.5.2 Drilling for Mini Pilot Plant Sampling

The objective of the mini pilot plant sampling drilling was to provide representative mineralogical variability in a larger sample size for testwork at RNC's mini pilot plant located in Thetford Mines, Quebec. A series of 7 pilot drill holes totalling 1,757 m were completed to characterize the near-surface mineralization in order to select representative mineralization domains for sampling by large diameter drilling for mini pilot plant testing in 2010. On the basis of the results from these pilot holes, four locations were selected for large diameter (PQ-size) diamond drill coring and thirteen holes totalling 2,785 m were completed. Multiple holes were planned on each site in order to acquire a sufficient sample of each metallurgical domain. Location of the pilot holes and the selected PQ drilling sites is shown in Figure 10.1.

The mini pilot plant sample drill holes (PQ) are sampled according to the variability domain composites defined in the pilot holes. Sampling procedures are described in Section 11.1.3. Samples were stored on site in Amos until required for testwork in the RNC mini pilot plant. Figure 10.9 on the following page shows the site of the 10-RN-218 mini pilot plant drill hole collars.

10.6 Regional Exploration Drilling

A diamond drilling program was designed to evaluate exploration targets that occur on the Dumont property outside the Dumont resource. A total of 3,392 m in 13 holes was drilled from March to April 2013 to evaluate a series of geophysical targets. The location of these holes is shown in Figure 10.1. No significant results were returned from this regional drilling program.

Figure 10.9: Drill Site Showing Collars for 10-RN-218 PQ Mini Pilot Plant Holes

Source: RNC.

11 SAMPLE PREPARATION, ANALYSIS, SECURITY

Descriptions of the historical sampling methods and approaches for the Dumont property have been previously provided in Section 6 of this report. Prior to the initial drilling program conducted in 2007, RNC did not conduct any sample preparation or analysis, as no samples were collected from the property during the period leading up to the drilling program. Since initiating field exploration work in March 2007 RNC has maintained strict sample preparation and security procedures and a Quality Assurance/Quality Control (QA/QC) program following industry best practices.

SRK reviewed sample preparation, analyses, and security procedures and discussed the QA/QC program with RNC staff during the site visit in 2011. SRK also performed independent data analyses verification checks as described in Section 12 and has also reviewed the results of the QA/QC program for the 2008, 2009, 2010, 2011, and 2012 Technical Reports.

In the opinion of SRK the sampling preparation, security and analytical procedures used by RNC are consistent with generally accepted industry best practices and are therefore adequate.

11.1 Sample Preparation & Analyses

There has been no change to core drilling assay/geochemical, mineralogical mapping, mini pilot plant sampling methods, electron microprobe determinations, comminution testwork, and geochemical characterization of Dumont rocks and tailings described below since the last Technical Report entitled "Technical Report on the Dumont project, Launay and Trécesson Townships, Quebec, Canada" (June 2012). A new sampling campaign for chrysotile quantification has since been initiated and is described below.

11.1.1 Drill Core Assay/Geochemical Sampling

11.1.1.1 Sample Collection & Transportation

Diamond drilling sampling controls start after a run has been completed and the rods are pulled out of the drill hole. The core is removed from the core barrel and placed in core boxes. The capacity of each box depends on the diameter of core stored in it (1.5 m for PQ diameter, 3.0 m for HQ diameter or 4.5 m for NQ diameter). This follows standard industry procedures.

Small wooden tags mark the distance drilled in metres at the end of each run. On each filled core box, the drill hole number and sequential box numbers are marked by the drill helper and checked by the geologist. Once the core box is filled at the drill site, the box is covered with a lid to protect the core and the box is sent to the core logging facility in Amos at the end of each shift for further processing. In general, the core recovery for the diamond drill holes on the Dumont property has been better than 95% and little core loss due to poor drilling methods or procedures has been experienced. There is no statistical difference on core recovery along strike or by geological or metallurgical domain.

11.1.1.2 Core Logging & Sampling

Once the core boxes arrive at the logging facility in Amos, the boxes are laid out in order, the lids are removed and the head of the first box is marked in red to denote the starting point of the drill hole. The core is then laid out on the logging table and cleaned to remove any grease and dirt which may have entered the boxes. The core is stored sequentially hole by hole in racks for logging. Core logging consists of two major parts: geotechnical logging and geological logging.

The diamond drill core sampling is conducted by a team of several staff geologists, all geologists in training (GIT) and geological technicians under the close supervision of the RNC geologist in charge of the program on site. The RNC staff geologists are responsible for the integrity of the samples from the time they are taken until they are shipped to the preparation facilities in Rouyn-Noranda or Timmins.

The geotechnical logging is completed first to check the core pieces for best fit and to determine core recovery, Rock Quality Designation (RQD), Index of Rock Strength (IRS) and magnetic susceptibility. The number of open (natural) fractures in the core is counted and the fracture surfaces are evaluated for their joint surface condition.

Geological logging follows and is comprised of recording the lithology, alteration, texture, colour, mineralization, structure and sample intervals. All geotechnical and geological logging and sample data are recorded directly into a computerized database using CAE Mining's (formerly Century Systems) DHLogger data logging software.

During the core logging process the geologists define the sample contacts and designate the axis along which to split the core with special attention paid to the mineralized zones to ensure representative splits. All core which is classified as dunite by the geological logging is marked in 1.5 m intervals for sampling. Any mineralized sections outside the dunite are also marked for sampling. Outside the dunite unit a minimum of one, 1.5 m control sample in every 10 m of core is taken. See Figure 11.1 for a photograph of the core logging facilities in Amos.

Samples are identified by inserting three identical pre-fabricated, sequentially-numbered, weather-resistant sample tags at the end of each sample interval.

Once the core is logged, photographed and the samples are marked, the core boxes are transferred to the cutting room for sampling. Sections marked for sampling are split using a diamond saw. Once the core is split in half, one half is placed into a plastic sample bag and the other half is returned to the core box. The core cutting technicians verify that the interval on the sample tag matches the markings on the core and that the sample tag matches the sample number on the bag. The half of the cut core returned to the core box is then re-marked by the core technician with a grease pencil to indicate the end of the sample interval. The boxes containing the remaining half core are stacked and stored on site in the secure core storage facility.

Duplicate, blank and standard samples are inserted into the sample stream at regular intervals using a sequential numbering scheme set up by RNC.

Figure 11.1: Core Logging Facilities in Amos

Source: RNC.

Once the sample is placed in its plastic sample bag, the bag is secured with electrical tie wraps and the sample bags are placed into large fabrene sacks. Generally, seven sample bags are placed into each fabrene bag and then the bag is secured with an electrical tie wrap. The fabrene sample bags remain secured in the core shack in Amos until they are shipped to the laboratory by courier. The general shipping rate for the samples is once for every 100 to 150 samples.

After-hours access to the core logging, core cutting and core storage facilities, as well as the project office, is controlled by a zoned alarm system with access restrictions based on employee function.

11.1.1.3 Sample Preparation & Analysis

Since 1 June 2008, RNC's samples have been prepared at ALS Minerals' (formerly ALS-Chemex) preparation facility in Timmins, Ontario and analyzed at ALS Minerals' laboratory in Vancouver, British Columbia. Both the preparatory facility and assay laboratory have ISO 9001:2000 certification. Expert Laboratories, located in Rouyn-Noranda, Quebec is not ISO certified; however, it does participate in the CANMET round-robin proficiency testing twice yearly. Prior to 1 June 2008, all samples were assayed at Expert Laboratories and then all the pulps were re-assayed at ALS Minerals. 5% of each assay batch returned from ALS Minerals is randomly selected for check assay. Until June 2011 the check assays occurred at Expert Laboratories, subsequently RNC changed the umpire laboratory to AGAT Laboratories in Mississauga. AGAT is ISO 9001:2000 certified and accredited by the Standards Council of Canada (SCC).

Once the samples reach ALS Minerals' Timmins preparation laboratory, each sample is dried as needed, crushed, and split into "reject" and a 250 g aliquot for pulverization. After pulverization the 250 g pulverized sample aliquot is again split into a 150 g master sample and a 100 g analytical sample. The 150 g master sample is stored in the Timmins facility for reference and the 100 g analytical sample is forwarded to the ALS Minerals analytical laboratory for assaying in Vancouver. On receipt in Vancouver, the specific gravity of the analytical sample material is measured by gas pycnometer, and this is followed by a 35-element analysis using an aqua regia digestion and ICP-AES finish. Where reported nickel values exceed 4,000 ppm, a second analysis is completed from the 100 g analytical sample using a four acid total digestion with an ICP-AES finish. This 4,000 ppm threshold reanalysis was raised to 10,000 ppm on 1 June 2008. In addition, all samples are assayed for precious metals (gold, platinum, palladium) using a standard fire assay with an ICP-AES finish.

After a holding period at the laboratories, all pulps and rejects are returned to RNC in Amos for long-term storage.

All analytical data are reconciled with the drill log sample records and recorded in the project database. For the purpose of geological and resource modelling, the ALS Minerals aqua regia determinations are used for samples under 10,000 ppm nickel and the ALS Minerals total digestion determinations are used for samples over 10,000 ppm nickel.

11.1.1.4 Control Samples

As part of RNC's QA/QC procedures, a set of control samples comprised of a blank, a field duplicate and a standard reference material sample, are inserted sequentially into the sample stream. The cut core samples, along with the inserted control samples, are then shipped to the ALS Minerals assay preparation facility in Timmins.

11.1.1.5 Blank Samples

The blank samples used for the Dumont project consist of local esker sand. The esker sand is collected in 205-L drums by a local Amos construction contractor. Randomly selected samples were collected from the drum and assayed at ALS Minerals to evaluate the composition of the sand and determine its suitability for use as a blank control sample. The assayed nickel grades from these samples range from 30 to 80 ppm. The qualified blank sample drum is sealed and placed at a clean place for further use. RNC sets 100 ppm nickel as the recommended upper limit of the blank sample value.

The blank samples are submitted into the sample stream at the rate of approximately one for every 25 samples.

11.1.1.6 Duplicate Samples

A duplicate sample is submitted into the sample stream at a rate of approximately one for every 25 samples. The sample and its duplicate consist of quartered core from the given sample interval. The remaining half-core is placed back into the core box for future reference.

11.1.1.7 Standard Reference Material Samples

The Standard Reference Material Samples (SRMS) are inserted into the sample stream at the rate of approximately one for every 25 samples. Initially one high-grade SRMS (OREAS 14P) was inserted into the sample stream for every three low-grade SRMS (OREAS 13P) submitted. On the phasing out of OREAS 13P and 14P, OREAS 70P was inserted into the sample stream at the same sample rate of one for every 25 samples. An exception to this occurs where logging personnel visually recognize zones of higher grade mineralization; through these high-grade zones OREAS 72a is inserted. If the situation arises where the twenty-fifth sample is consistently located in between higher grade mineralization zones, a higher grade sample will be inserted outside the one-in-25 sequence to ensure that the higher grade zones are represented by standard reference materials.

Four SRMS have been used in the project. The SRMS were prepared by Ore Research & Exploration Pty. Ltd. of Australia. Table 11-1 summarizes the specifications for the SRMS.

Table 11-1: Summary of the Specifications for the Standard Reference Material Samples

Description	Constituent	Recommended Value	95% Confidence Interval	
			Low	High
OREAS 13P	Cobalt (ppm)	88	85	91
	Copper (ppm)	2,504	2,439	2,569
	Gold (ppb)	47	45	49
	Nickel (ppm)	2,261	2,233	2,289
	Palladium (ppb)	70	68	72
	Platinum (ppb)	47	46	48
OREAS 14P	Cobalt (ppm)	754	739	769
	Copper (%)	0.997	0.979	1.1015
	Gold (ppb)	51	50	52
	Nickel (%)	2.09	2.04	2.14
	Palladium (ppb)	150	147	153
	Platinum (ppb)	99	96	102
OREAS 70P	Cobalt (ppm)	83	76	89
	Copper (ppm)	2.6	1.4	3.8
	Gold (ppb)	13	9	16
	Nickel (ppm) Aqua Regia	2,438	2,222	2,655
	Nickel (ppm) 4 Acid	2,730	2,620	2,841
	Palladium (ppb)	<1	IND	IND
OREAS 72a	Cobalt (ppm)	157	151	164
	Copper (ppm)	316	309	323
	Gold (ppb)	6	5	7
	Nickel (%) 4 Acid	0.693	0.683	0.704
	Palladium (ppb)	41	39	44
	Platinum (ppb)	36	34	38

Note: Table supplied by RNC after Ore Research & Exploration Pty Ltd., (2003, 2004a, 2004b, 2006).

11.1.2 Mineralogical Mapping Sampling

The mineralogical mapping sampling program uses SGS Minerals Services' (formerly SGS Lakefield) EXPLOMIN^{TM} application of Quantitative Evaluation of Minerals by Scanning electron microscopy (QEMSCAN) methods to provide detailed mineralogical information on mineral assemblages, nickel deportment, liberation, alteration and the variability of these factors. Mineralogical samples were taken for the purpose of metallurgical domain composite characterization and for the purpose of mineralogical mapping of the Dumont deposit.

11.1.2.1 Sample Definition & Sampling

The mineralogical mapping sampling program samples a quarter of the NQ core drilled and previously sampled for the resource definition program. In areas of interest, sample length and location are defined to coincide with previous assay sample intervals to ensure that a direct comparison can be made between results obtained from assay/geochemical analyses and mineralogical sampling results.

The selected mineralogical mapping samples are given a unique sample identification number (ID), photographed, and sent to the core cutting area. Mineralogical mapping sampling is usually completed in batches, where multiple samples are selected from each hole, then cut sequentially.

The half-core remaining from the previous assay sampling is quarter-split to produce the mineralogical sample. A portion of the quartered core is cut further to produce a pre-selected portion of rock for thin section field stitch analysis. The selected portion for field stitch analysis and the quartered core are each placed in separate bags, and identified by the same mineralogical mapping sample ID.

For QA/QC purposes, a piece of the quartered core selected for mineralogical particle scan analysis is selected from the sample bag and placed in the RNC mineralogical mapping sampling library.

Once a sample is placed in its plastic bag, the bag is secured with staples. Typically, seven sample bags are placed into a cardboard box and secured with tape. The sealed boxes remain secured in the Amos core logging facilities until they are shipped to the laboratory using a courier service. Samples are shipped at the rate of 50 to 100 samples per shipment. Blanks and standard samples are inserted into the sample stream at regular intervals using a sequential numbering scheme set up by RNC.

The sample bag with the thin section slice is sent directly to SGS Minerals Services for thin section preparation and mineralogical analysis. The sample bag containing the quarter core is sent first to ALS Minerals' Timmins preparation laboratory for stage crushing and assaying, with a split shipped to SGS Minerals Services for mineralogical particle scan analysis.

After-hours access to the core logging, core cutting and core storage facilities, as well as the project office, is controlled by a zoned alarm system with access restrictions based on employee function.

11.1.2.2 Sample Preparation & Analysis

Upon receipt at ALS Minerals' Timmins preparation laboratory the mineralogical samples are prepared according to the procedure summarized in Table 11-2.

Table 11-2: EXPLOMIN™ Mineralogical Sample Preparation Procedure at ALS

Mineralogical Sample Preparation Procedures	
WEI-21	Weigh and log received sample
LOG-22	Log sample
CRU-31	Crush entire sample to > 70% passing 2 mm
SPL-21	Riffle split 100g for pulverizing
PUL-35	Stage pulverize, two 100g splits to 90% passing 106 µm
WSH-22	Wash pulveriser
CRU-QC ≥	Crush to 70% passing 2 mm
PUL-QC ≥	Pulverize to 90% passing 150 mesh

Note: Table supplied by ALS Minerals.

The first 100 g split of pulverized material is sent to SGS Minerals Services where the sample is prepared for EXPLOMIN™ particle scan mineralogy and XRF Borate Fusion assay. The results are forwarded to RNC and imported directly into the database.

The other 100 g split of the pulverized material is retained by ALS Minerals for chemical analyses. The reject material is sent back to the RNC's Amos office for storage. The results are forwarded to RNC and imported directly into the database.

11.1.2.3 Geochemical Preparation & Analysis

Samples are analyzed at the ALS Minerals Laboratory in Vancouver, for specific gravity by gas pycnometer, followed by a 35-element analysis using an aqua regia digestion and ICP-AES finish. Where reported nickel values exceeded 10,000 ppm a second analysis is completed using a four acid total digestion with an ICP-AES finish. In addition, all samples are assayed for precious metals (gold, platinum, palladium) using a standard fire assay with an ICP-AES finish.

Analysis results are forwarded to RNC and imported directly into the project database.

11.1.2.4 Mineralogical Preparation & Analysis

Procedures for EXPLOMIN™ mineralogical analysis and sample preparation internal to SGS were provided to RNC by SGS (A. Karaca, July 26, 2010 email as personal communication). Relevant sections of these procedures are quoted below.

"Upon sample receipt, the Sample Log-on technician verifies the received samples according to the sample list provided by RNC geologists. Any extra sample(s), discrepancies in identification, damage, contamination, unsuitable samples, concerns, or hazards are recorded, and RNC is notified. Once sample receipt is verified, samples are forwarded to the mineralogist for sample login and LIMS [laboratory information management system] reporting. The samples are kept in the same order that they appear on the documentation provided by RNC."

"For sample tracking purposes within SGS Minerals Services, LIMS numbers are assigned to incoming samples. The LIMS number reflects the type of work being performed on the samples, the source of the samples, and secondary information such a Reference, Project, Batch, Quote, Link, Note, Category, Supervisor, Priority, Warning, Charge ID, Date Received, Date Requested."

“When the LIMS log-in has been completed, a project file is created to hold all the paperwork pertaining to the project. The project file is labelled with the project number, LIMS number, and the Client or Company name. A log-in checklist is attached to the project file and completed. A chain of custody is created. Record LIMS information is recorded in Diamond Services/Mineralogy project list.”

“The project file is placed in a red folder and given to the Mineralogy Project Supervisor. Once the folder is checked by the Mineralogy Project Supervisor it is returned to Sample Login. Any additional information is updated in LIMS and the project list. The signed Chain of Custody is photocopied and the original is mailed to the client.”

“Active Mineralogy Samples are stored with labels containing the project number, LIMS number, and test required. All of the samples are placed in one of the LIMS numbered, large plastic bags, placed in the ‘To Do’ box. A copy of the work order accompanies the samples.”

“When all requested analyses have been completed, samples are brought to Sample Tracking for storage. Boxes are stored in the Sample Tracking Room in Mineralogical Services for six months. After six months, the box is inventoried and the mineralogist is contacted for further instructions.”

11.1.2.5 Sample Preparation

“Using a binocular microscope, the Mineralogist or Project Mineralogist identifies the areas of interests previously marked by RNC staff for thin section analysis. One polished section for each sample is prepared for field stitch analysis. Sections are ground and polished then coated with carbon for analysis.”

“Crushed samples that are received later on from ALS Minerals are first riffle-split into two parts (of ~125 g), one for mineralogy and one for assay. Each sample is potted in moulds and the necessary amount of resin and hardener is added. The moulds are placed into the pressure vessel and left under pressure for five hours. The moulds are then labelled and backfilled with resin. Then they are placed in the oven. The sections are ground and polished followed by carbon coating.”

11.1.2.6 QEMSCAN Operation

“The block holder is loaded with the samples. Measurement parameters (for core samples, field scan mode with 10 µm resolution and for crushed samples, PMA mode with 3 µm resolution) are set up. Stage Set-Up, Focus Calibration, Beam optimization and BSE Calibration are performed at the start of each run. After the runs are completed, the daily quality checks are performed as summarized in Table 11-3. Weekly calibration and checks are also performed to verify the following: Stage Initialization, Tilt Check, Rotation Check, X-Ray Detector Check, Gun Set-up, Brightness and Contrast, Filaments and Vacuum. The detectors are checked every three months.”

“The QEMSCAN Data Validation report includes a measurement validation table and an assay reconciliation chart. QEMSCAN data are compared to externally measured chemical assay data to ensure measurement accuracy. Minerals are double-checked optically. A technical check is performed on all data by a senior mineralogist."

Table 11-3: SGS Minerals Services Daily Quality Checks for QEMSCAN Analysis

Task/Duty	Operational Purpose	Management Purpose
Checking correctness of PS placement.	Statistics will readily show if samples and parameters are mismatched.	Proper scheduling and quality control protocols.
Check that analyses have been performed successfully.	Go-, no-go decision to perform sample exchange for next analysis batch.	Keep track of scheduling, processing and project management.
Keep track of the measurement statistics as a matter of record	Optimization of analyses is influenced by the interdependence of PS-packing density and point-spacing	If additional statistics are required for particle or modal accuracy, additional PS's may be required.
To assist in optimizing analysis parameters and analysis times.	For reviewing parameter selection criteria. Resolution vs. speed.	Establishing accuracy and precision of measurement.

Note: Table supplied by SGS Minerals Services.

Analytical results are forwarded to RNC and imported directly into the database.

11.1.2.7 Control Samples

As a part of SGS Minerals Services standard QA/QC procedures for QEMSCAN analysis, a standard sample is run every week. There are currently three standard samples from different projects that are cycled each time. One of the standards used is a RNC data validation sample.

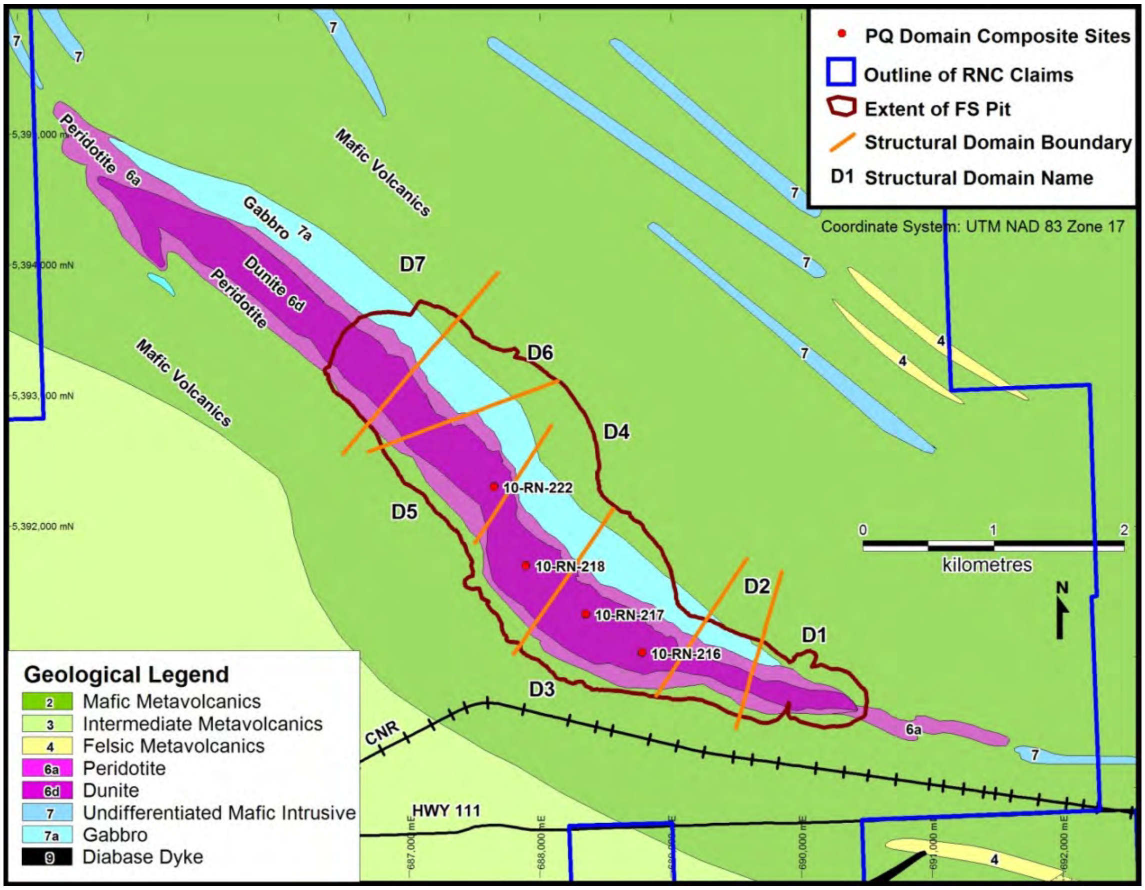
As part of RNC's QA/QC procedures for geochemical assays, a set of control samples comprised of a blank and standard reference material sample, are inserted sequentially into the sample stream. The cut mineralogical samples along with the inserted control samples are then shipped to the ALS Minerals for stage crushing and chemical analysis. The standard reference materials and blanks used are analogous to those described previously with the exception that the frequency of insertion is increased to approximately one in every 15 samples.

11.1.3 Mini Pilot Plant Sampling

PQ core metallurgical domain composite samples are selected based on nickel deportment, grade and alteration of the rocks as determined through assays and mineralogical sampling of an NQ pilot hole drilled at the sampling location. A 1.5 m PQ drilling grid was established around each NQ pilot hole to plan multiple PQ holes on the same site in order to accommodate the sample volume required (approximately 1,800 kg per domain sample) while maintaining domain sample uniformity. As a result of the hole proximity and the inherent difficulty and cost of PQ drilling in overburden, a percussion water well-drilling rig was employed to drive casing into bedrock for the multiple holes required on each of the sites. Once casing was seated in bedrock, the diamond drill returned to drill the PQ core domain samples.

Four locations were chosen, 10-RNC-216 to -218 and 10-RNC-222. Figure 11.2 shows the location of each of the holes along the length of the deposit.

Figure 11.2: Location of the PQ Drill Holes



Source: RNC.

The sampling method for PQ core is identical to that described previously up to and including the geotechnical logging, after which the procedure is different. After geotechnical logging, the core is thoroughly cleaned to remove any drilling additives that may interfere with the metallurgical testwork. The PQ core is then checked for comparability to the pilot hole, by comparing lithological contacts, mineralization, alteration, and structural features. The core is then logged for lithology, and metallurgical domain composite samples are delineated which reflect those established in the pilot NQ hole. The core is then photographed and placed in short-term indoor storage to await sampling. After-hours access to the core logging, core cutting and core storage facilities, as well as the project office, is controlled by a zoned alarm system with access restrictions based on employee function.

The PQ sampling program is supervised by an independent qualified engineer provided by Stavibel Inc. (Stavibel) to ensure quality control of the sampling method and to certify chain of custody. The rock is weighed and transferred by domain sample from the core boxes directly into 200 L plastic barrels fitted with Schrader valves. The domain samples are kept separate and barrels are filled in sequential order. A barrel typically holds from 250 to 270 kg of rock. The engineer seals the full barrel and places a numbered tag on the closure to prevent or identify any possible tampering. The barrels are purged with nitrogen to prevent oxidation and degradation of the rock while the sample awaits metallurgical testwork.

When the sample is required by RNC's metallurgical group, the barrels are shipped directly via road freight to the mini pilot plant in Thetford Mines, Quebec.

11.1.4 Electron Microprobe Sampling

Polished sections from the mineralogical mapping program from locations throughout the Dumont deposit (as described in Section 11.1.2) were selected to quantify the variability of nickel content in key minerals of interest by electron microprobe analysis.

RNC contracted SGS Minerals Services to conduct a detailed electron microprobe analyses on these samples which were already in storage at SGS Minerals Services facilities. SGS subcontracted the analyses to facilities at McGill and Laval University. The McGill University Electron Microprobe Microanalytical Facility is equipped with a JEOL 8900 instrument while the Laval Microanalysis Laboratory is equipped with a CAMECA SX-100. Machine calibrations, replicates and all results passed internal QA/QC procedures used at the facilities and checks as prescribed by SGS Minerals Services.

To further supplement this work in 2012, RNC contracted the Xstrata Process Support (XPS) Mineral Science Laboratory. XPS completed additional quantitative compositional mineral analysis using a Cameca SX-100 electron microprobe. Electron probe microanalysis (EPMA) produces higher electron beam currents and increased beam stability, coupled with higher resolution wavelength dispersive spectrometry (WDS) to produce mineral composition data down to ppm levels. All standard calibrations and QA/QC checks were completed in accordance to XPS Standards and Procedures.

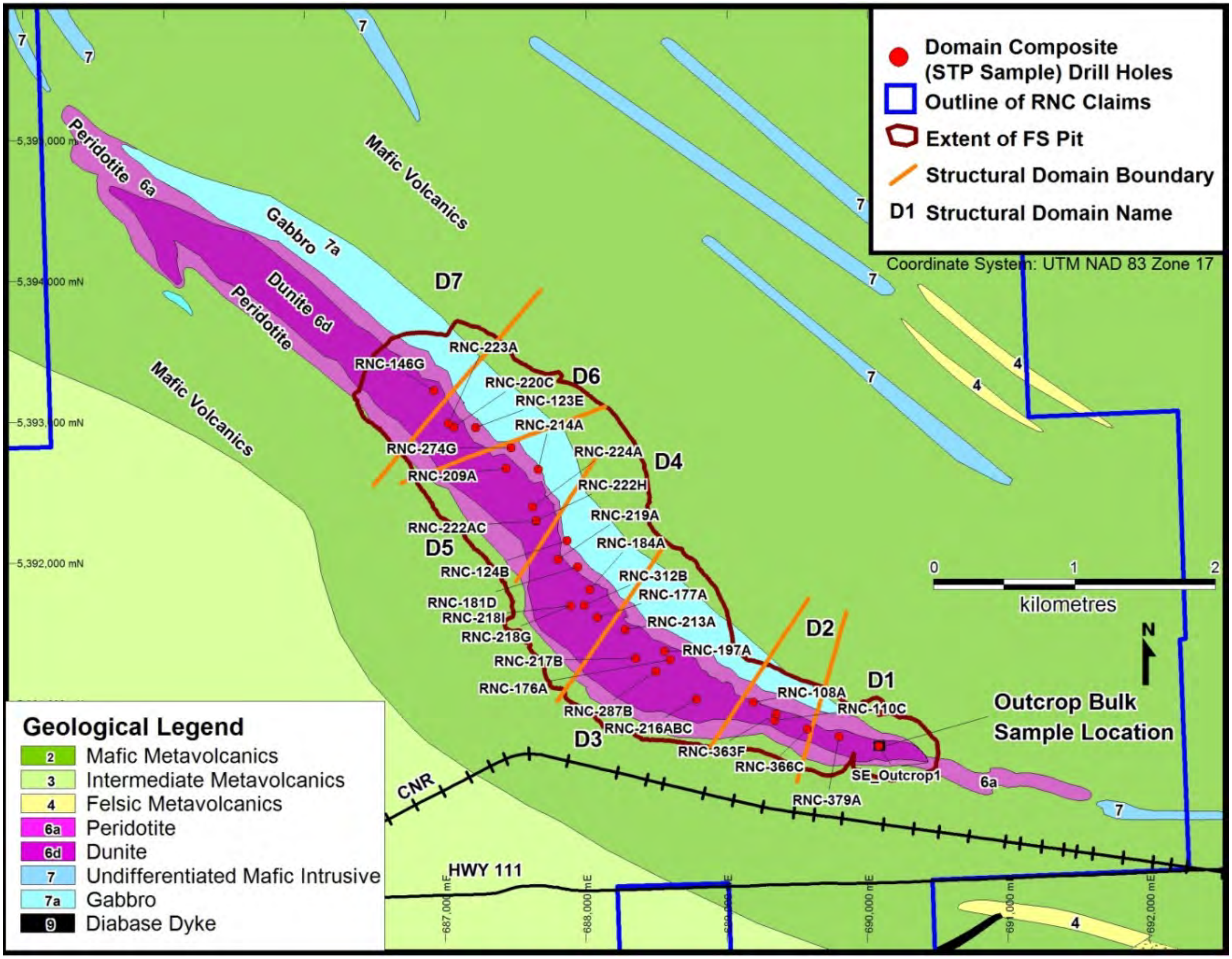
11.1.5 Metallurgical Variability Sample Selection

The metallurgical variability samples were collected from various locations in the deposit as shown in Figure 11.3.

These metallurgical variability samples were chosen to cover the variability in mineralogy and composition across the deposit. Samples were collected in drill holes distributed to be spatially representative both along strike, and across dip (stratigraphy) of the deposit. The major variables examined were nickel grade, nickel deportment, liberation, grain size, association and fibre content. Testwork was completed on 105 individual metallurgical domain composite samples. Testwork includes both metallurgical lab scale recovery tests as well as mineralogical analysis by QEMSCAN and assay.

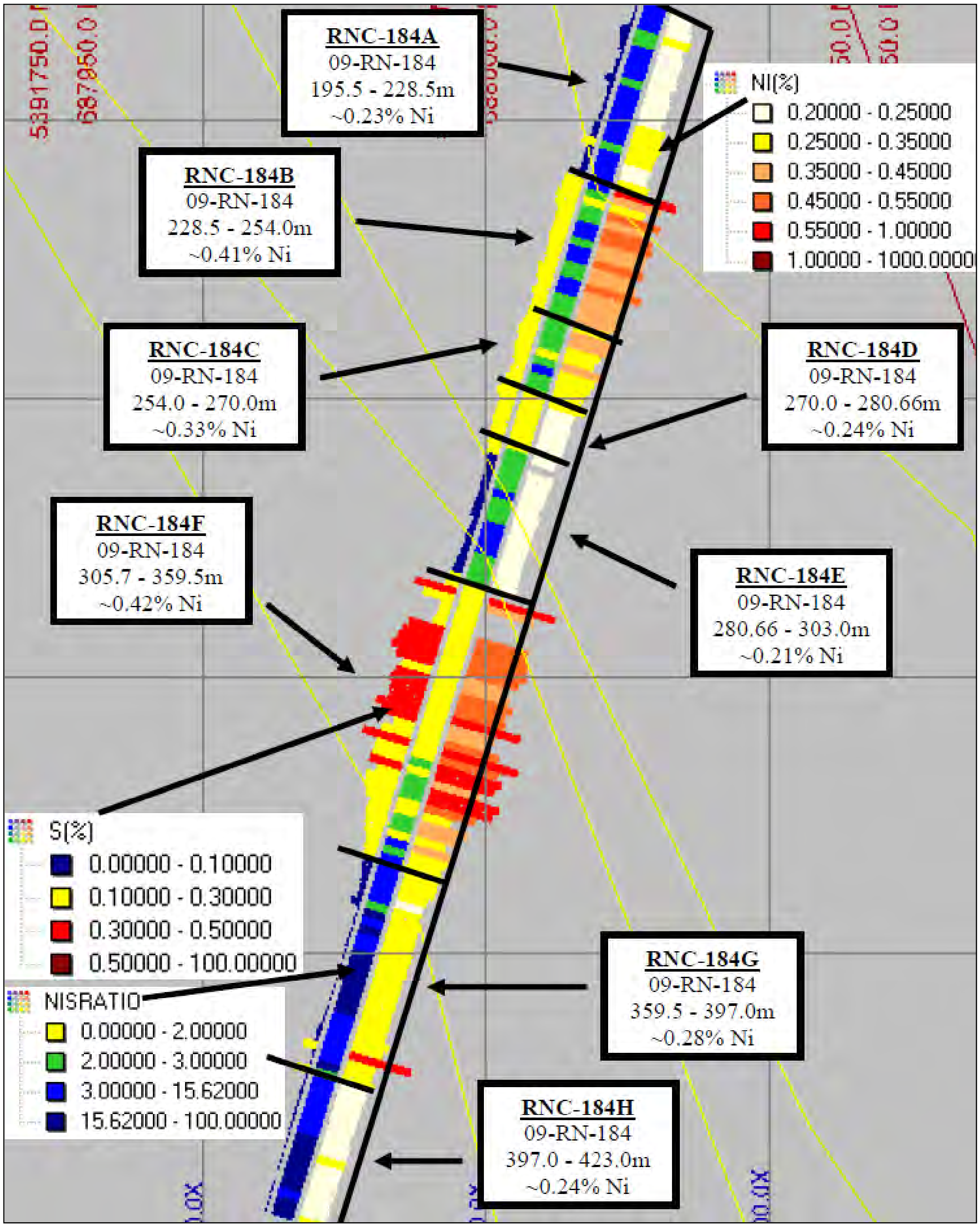
Continuous domain samples were assembled along the continuous length of the drill holes as shown in Figure 11.4. Each of the samples defined a homogeneous domain as characterized by nickel grade, nickel deportment, mineralization grain size and alteration. Any change in these characteristics led to the start of a new sample.

Figure 11.3: Location of Metallurgical Variability Samples (STP Samples)



Source: RNC.

Figure 11.4: Example of Domaining of Each Hole for STP Samples



Source: RNC.

11.1.6 Comminution Sampling

An extensive grindability study was performed on 102 samples from the Dumont deposit. Two types of samples were provided for the testwork, 92 half-NQ and 10 full PQ core samples, corresponding to variability and drop-weight samples, respectively.

11.1.6.1 Sampling Selection

The 92 half-NQ and 10 full PQ core samples have been selected from previously drilled and stored core by RNC. Samples were selected throughout the feasibility pit shell and considered:

- preliminary hardness domains (as indicated from point load testing corresponding to olivine, serpentine, coalingite and faulted domains);
- nickel deportment; and
- distribution throughout feasibility payback shell.

All selected samples are contained within the mineralization envelope to target mineralized dunite of various grades and mineralization types. Half of the selected 92 half-NQ samples (45) were chosen inside the feasibility payback shell. The remaining 47 samples were evenly distributed through the remaining volume of the mineralized envelope within the feasibility pit shell. Selected drill hole intersections were chosen to represent the range of mineralogical and chemical variations with focus on those factors which seem to affect point load strength index (PLSI).

11.1.6.2 Sample Preparation

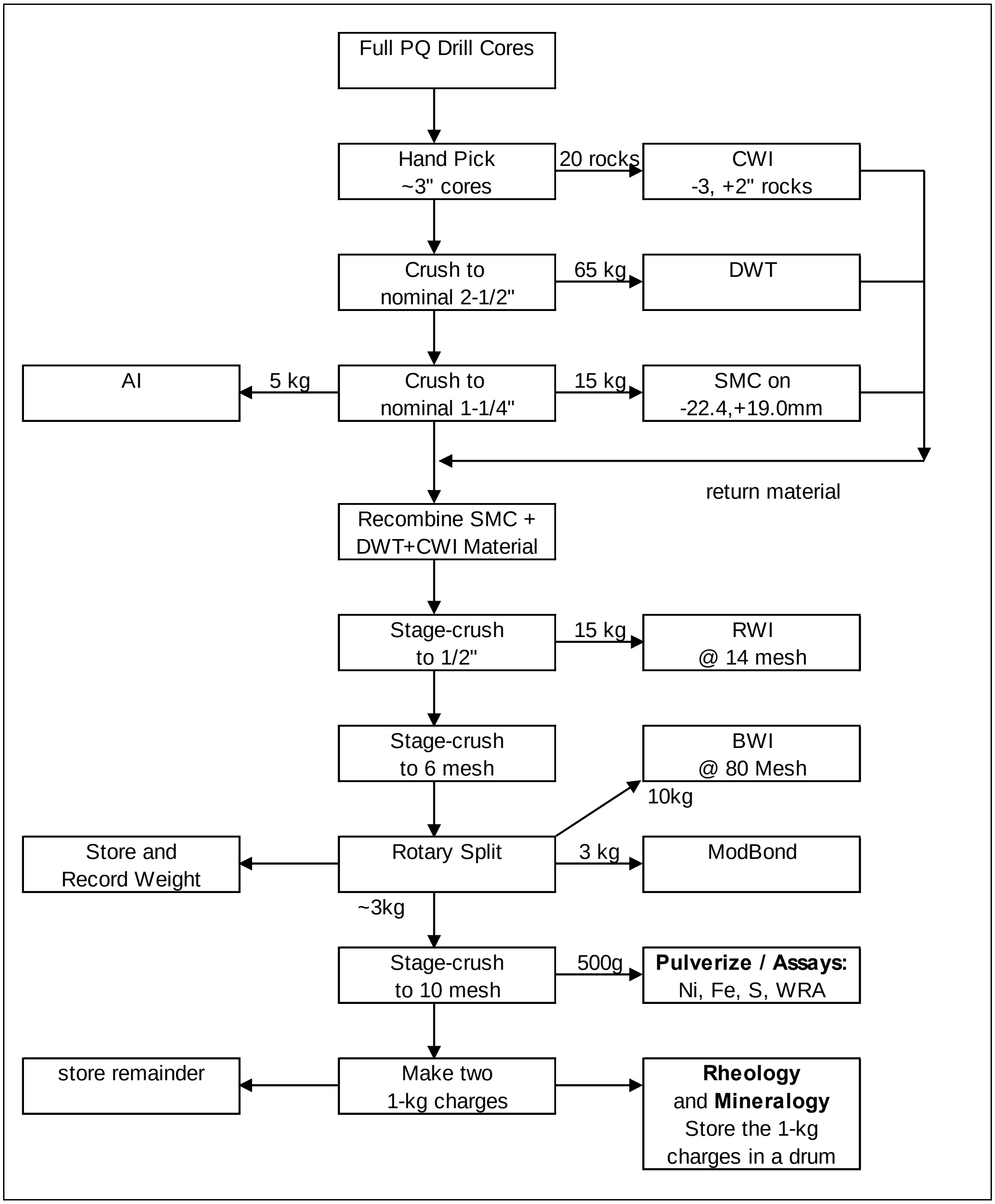
Several shipments of drill core were shipped to the SGS Minerals Lakefield, Ontario site from January to March 2011. The 10 full PQ drill core samples were prepared as shown in Figure 11.5.

These samples underwent the following tests:

- Bond Low-energy Impact Test (CWi);
- Drop-weight Test (DWT);
- SMC Test (SMC);
- Bond Rod Mill Grindability Test (RWi);
- Bond Ball Mill Grindability Test (BWI);
- Bond Abrasion Test (Ai);
- Rheological Characterization; and
- Mineralogical Characterization.

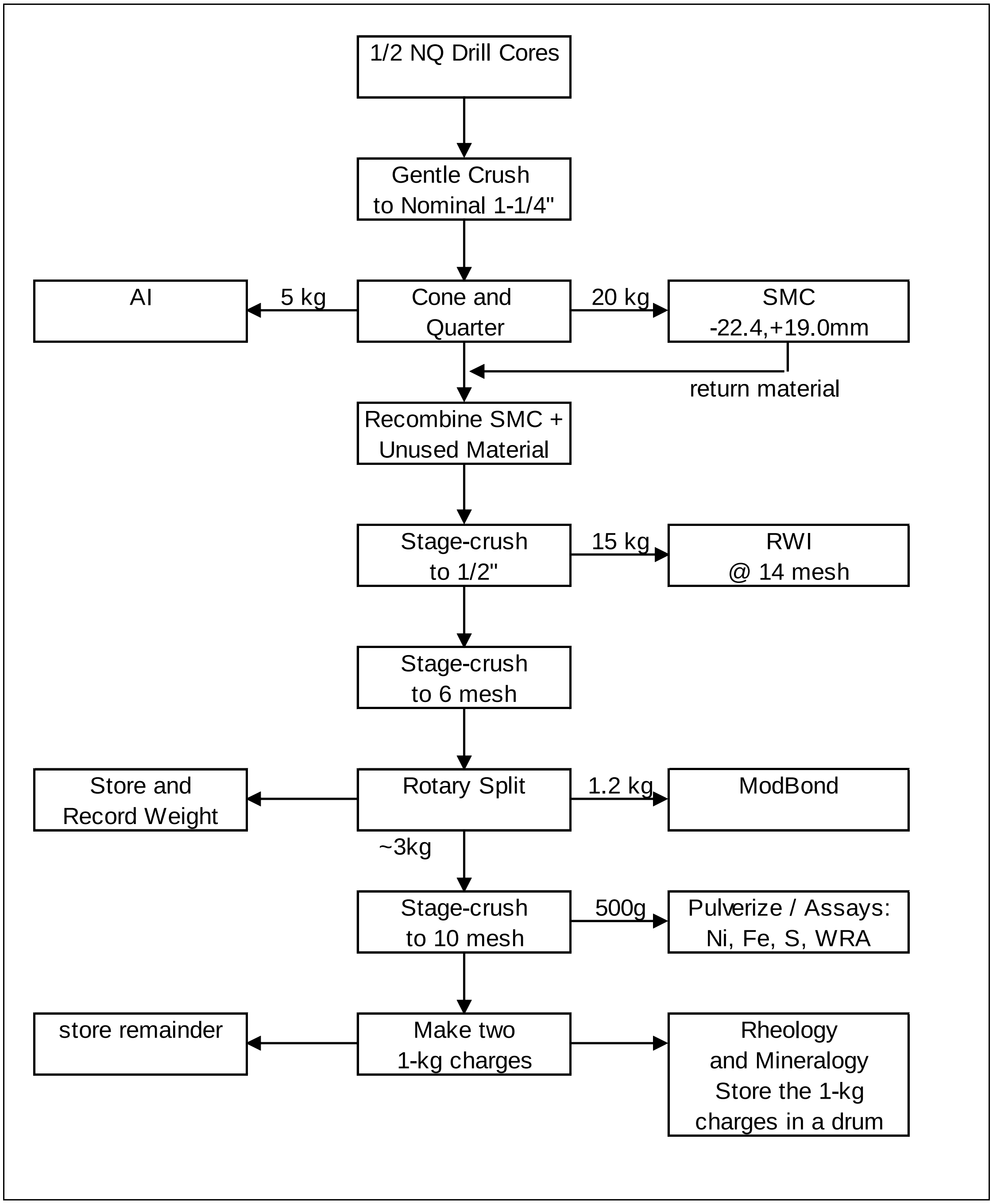
The 92 half-NQ drill core samples were submitted for the same suite of tests with the exception of the Bond low-energy impact test and the drop-weight test. The preparation of the 92 half-NQ drill core samples is shown in Figure 11.6. Three samples selected by RNC were submitted for full rheology benchmark testing in order to establish testing criteria that would be applied to the 89 remaining samples.

Figure 11.5: Sample Preparation Diagram – Full PQ Drill Core



Source: SGS Minerals Services.

Figure 11.6: Sample Preparation Diagram – Half-NQ Drill Core



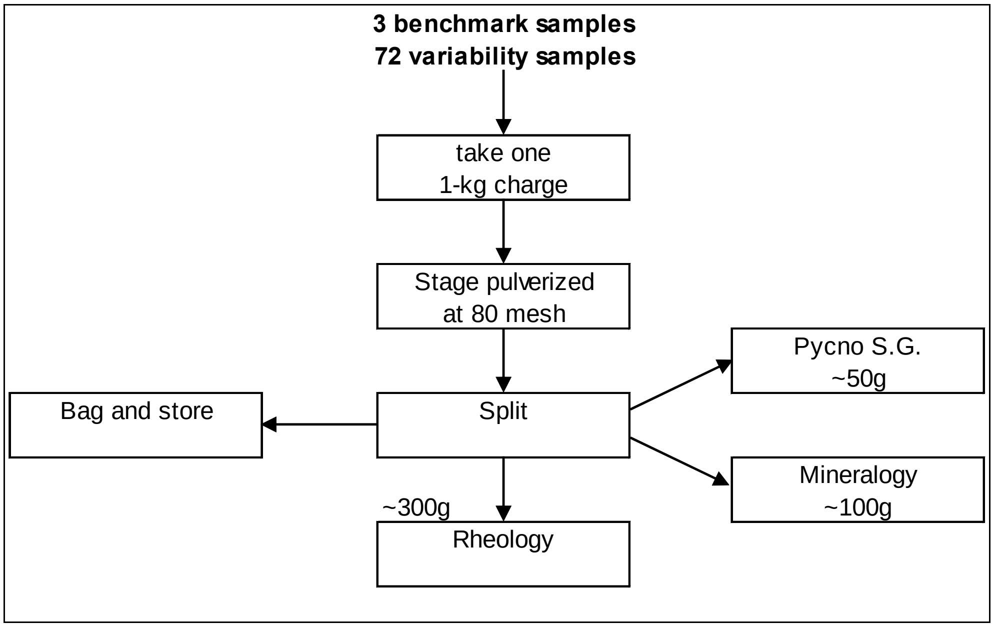
Source: SGS Minerals Services.

The samples submitted for Bond ball mill grindability testing were also submitted for the ModBond test, in order to establish the ModBond – BWI correlation parameters. All the remaining minus 6 mesh material, totalling 4,339 kg in 20 drums, was shipped to a warehouse in Quebec at the request of RNC.

11.1.6.3 Rheology & Mineralogy Preparation

The preparation for the rheological characterization is shown in Figure 11.7. Note that an additional 1 kg charge was used for each of the three benchmark samples.

Figure 11.7: Sample Preparation for Variability Rheology



Source: SGS Minerals Services.

11.1.6.4 Head Assays

The samples were analysed for nickel, sulphur, iron and major elements (Whole Rock Analysis). The iron determinations were performed using two methods, Borate Fusion-XRF (Whole Rock Analysis) and Pyrosulphate Fusion -XRF.

Comminution, rheology and mineralogy results are summarized fully in Section 13 of this report.

11.1.7 Environmental Geochemistry Sampling

11.1.7.1 Sampling for Laboratory Testwork

The objectives of the geochemical characterization program are to: (1) classify mine waste according to Québec Directive 019 sur l'Industrie Minière (Directive 019) for waste management planning, (2) identify chemicals of potential environmental interest in the framework of future mine site water quality and possible water treatment requirements during mine operation, and (3) access the pit lake water quality in an in-pit tailings deposition scenario after mining operations cease. Sampling methodology and analytical procedures are described below. Program design and results are described in Section 20 of this Technical Report.

The phase 1 environmental geochemistry program (GENIVAR, 2010) was completed by GENIVAR in 2009. Samples were selected by one engineer and one geologist of GENIVAR with the help of one geologist of RNC. A total of 21 waste rock samples (three gabbro, ten peridotite, five dunite, two feldspar porphyry and one basalt) were selected for ABA and leaching tests. Six samples from the mineral deposit representing the low (three samples) and the high (three samples) nickel grades were also sent for acid-base accounting (ABA) and leaching tests. In addition, three tailings samples were selected for environmental testing. Five samples of different lithologies and grades (waste: peridotite and dunite, ore: low- and high-grade, tailings) were selected for humidity cell tests. Finally, a composite sample of mineralized rock (low- and high-grade) was created from five different samples for the Meteoric Water Mobility Procedure (MWMP) test.

For the phase 2 environmental geochemistry program (Golder, 2013) in 2011, rock samples were collected by RNC staff supervised by an RNC geologist according to a sampling scheme devised by Golder Associates Ltd. (Golder). A total of 93 samples of core from waste rock areas were collected from existing core of previously drilled exploration boreholes. Samples were collected throughout the deposit and mostly outside the ore shell but within or near the anticipated open pit. Each rock sample consisting of 3 to 5 kg of core was collected over an interval of approximately 5 to 10 m, and some sub-samples were collected at regular intervals of approximately 1 m. Each sample was checked against its log description in terms of rock type, alteration, and staining associated with sulphide mineral oxidation. A consistent sample collection procedure was applied for all rock samples. Each sample was bagged individually to avoid cross-contamination and was labelled with the unique sample identification number. Metallurgical processing wastes (equivalent to tailings) generated at an off-site processing facility were retained for geo-environmental analysis. The tailings were generated from composite samples of ore collected by RNC from each of the main mineralization types including alloy ore, sulphide ore and mixed ore. Three samples of tailings and three samples of associated process water were collected, packaged and shipped to the laboratory by RNC for analysis.

For the phase 3 environmental geochemistry program (Golder, 2013) in 2012, five more metallurgical processing wastes (equivalent to tailings) were generated from composite samples collected by RNC. The five composite tailings samples are representative of the five metallurgical ore types as described in the previous technical report (Ausenco 2012). The composite tailings samples and three samples of associated process water were collected, packaged and shipped to Maxxam Analytics Inc. (Maxxam) in Montréal by RNC for the similar static analysis complimenting the phase 2 program., In addition to the Maxxam work, three metallurgical processing wastes (equivalent to tailings) were generated from a composite of low-grade, non-sulphide ore, by the RNC team, and, packed and shipped by RNC to SGS Mineral Services for analysis. The purpose of these analyses was to assess the potential pit lake water quality in an in-pit tailings deposition scenario after mining is complete.

11.1.7.2 Analytical Methods for Laboratory Testwork (Maxxam)

The static tests completed on mine waste solids are consistent with those recommended by Directive 019 and include acid-base accounting (ABA), chemical composition (whole rock and trace element), and leaching tests (TCLP, SPLP, CTEU9).

ARD Potential

The potential of geologic materials to generate acid rock drainage (ARD) was evaluated through acid-base accounting (ABA) following Québec Method MA.110-ACISOL 1.0. This test includes the determination of the following parameters:

- total sulphur by LECO furnace and Acid Potential (AP) calculated based on total sulphur content
- Neutralization Potential (NP) (following Québec Method MA.110-ACISOL 1.0).

The values of AP and NP are reported as kg equivalent calcium carbonate ($CaCO_3$) per tonne of rock.

Neutralization Potential (NP)

NP is a bulk measurement of the acid-buffering capacity of a sample provided by various minerals of different reactivities and effective neutralization capacity. It is measured by digestion of a pulverized portion of the sample using a strong acid. This process consumes all minerals affected by the acid, including minerals that may not normally be reactive under ambient conditions and minerals that would not neutralize to pH-neutral conditions (such as silicate minerals. This method can overestimate effective NP.

Acid Potential (AP)

The potential of a material to generate acid (acid potential or AP) is calculated from the total sulphur content of the sample in equivalent calcium carbonate ($CaCO_3$). AP is a theoretical value that represents the maximum potential acidity that can be generated by sulphur-bearing minerals in a rock sample assuming that all sulphur is present as pyrite and is available to oxidize completely. This method is generally found to overestimate the AP because total sulphur includes non-reactive sulphur minerals such as sulphates and certain sulphides.

Chemical Composition

The chemical composition of the samples was determined through whole rock and trace element analyses. Major element composition was determined through whole rock analysis by borate fusion and X-ray fluorescence (XRF). Trace element composition was determined through the CEAEQ Method MA200 Mét 1.2 (Québec, 2010).

Metal Leaching Potential

Various short-term leach tests are used to determine the potential of the waste to release readily-soluble metals to the receiving environment. The leach tests performed follow Québec Method MA.100-Lix.com.1.0. They are summarized in Table 11-4 and described below.

11.1.7.3 Analytical Methods for Laboratory Testwork (SGS)

The following analysis/assays were completed to understand the chemical diffusion and transfer interaction between low-grade tailings and process water in the overlying water column: dissolved metals, pH, conductivity, alkalinity, acidity, PO_4, Br, Cl, F, NO_3, SO_4, and Cr(VI).

Table 11-4: Short-term Leach Test Procedures

Leach Test	Purpose	Procedure	Lixiviant
TCLP1 (Toxicity Characteristic Leaching Procedure)	Simulates leaching conditions in municipal landfills	EPA 1311 (USEPA, 1992)	- Crushed sample (<9.5 mm) - 20:1 ratio - acetic acid & sodium hydroxide - initial leachate pH 4.9 to 5.0 - 18-hour agitation
SPLP (Synthetic Precipitation Leaching Procedure)	Simulates acid rain leaching conditions	EPA 1312 (USEPA, 1992)	- Crushed sample (<9.5 mm) - 20:1 ratio - Sulphuric & nitric acids - initial leachate pH 4.2 - 18-hour agitation
CTEU9 (Equilibrium Extraction)	Water leach test to assess readily leachable metals	CTEU9 (CEAEQ, 2006)	- Pulverized sample (<150 µm) - 4:1 ratio - de-ionized water - closed system (no gas exchange) - no pH control - 7-day agitation

Source: RNC.

11.1.7.4 Sampling for In-Situ Experimental Cells

In-situ Low-Grade Ore Cell

A bulk sample of mineralized serpentinized dunite weighing 110 tonnes was collected from outcrop for inclusion in an in-situ experimental environmental characterization cell constructed on the Dumont property. The outcrop was cleared of glacial overburden with an excavator and power washed. The area identified for sampling was then drilled and blasted to a depth of approximately 1.5 m.

The sample was loaded into a dump truck and transported immediately to the in-situ cell site and deposited directly into the in-situ cell.

In-Situ Tailings Cell

A composite sample of tailings produced from the miniplant, weighing 3 tonnes, was prepared for deposition in an in-situ experimental environmental characterization cell constructed on the Dumont property.

The tailings were produced from the miniplant operation from August 2010 to June 2011. The source of the material was from the PQ Domain Composites 218BDF, 218G, 218H, 218I, 222AC, 217B and 216ABC. Both the slimes, fluff and rougher (non-mag) tails produced from the miniplant were used. The slimes had been stored as a low density slurry, the fluff was dry and the rougher tails were a wet filter cake.

The tailings samples was loaded into a cement truck, mixed thoroughly, transported immediately to the in-situ cell site and deposited directly at approximately 50% solids into the in-situ cell.

11.1.8 Chrysotile Quantification Sampling

A logging program to quantify the bulk chrysotile content of dunite and peridotite from the Dumont deposit was completed from January to March 2013 (Cloutier et al., 2013). The program consisted of detailed drill hole logging using half NQ core drilled and previously sampled for the resource definition program. Thirteen drill holes were selected to represent the dunite and peridotite lithologies based on representative lithological, spatial, structural, and metallurgical characteristics. RNC geologists created a standard logging procedure specifically for chrysotile to ensure consistency and reproducibility of results. This method has been validated by independent external experts (Verschelden and Jourdain, 2013; Gauthier, 2013) and provides reproducible and quantifiable results. Sample locations and results are described in Section 9.5.

11.2 Quality Assurance & Quality Control Programs

Quality assurance and quality control programs are typically set in place to ensure the reliability and trustworthiness of exploration data. They include written field procedures and independent verifications of aspects such as drilling, surveying, sampling and assaying, data management and database integrity. Appropriate documentation of quality control measures and regular analysis of quality control data are important as a safeguard for project data and form the basis for the quality assurance program implemented during exploration.

Analytical control measures typically involve internal and external laboratory control measures used to monitor the precision and accuracy of sampling, sample preparation and assaying. They are also important to prevent sample mix-up and to monitor the voluntary or inadvertent contamination of samples. Assaying protocols typically involve regular duplicate and replicate assays and the insertion of quality control samples to monitor the reliability of assaying results throughout the sampling and assaying procedures. Check assaying is typically performed as an additional reliability test of assaying results. Check assaying involves re-assaying a set number of rejects and pulps at a secondary umpire laboratory.

RNC has implemented external analytical control measures since commencing their drilling programs at the Dumont Nickel project in 2007 (Lewis and San Martin, 2010). Analytical control measures consist of the insertion of quality control samples (field blanks, field duplicates and certified reference material samples) in all sample batches submitted for assaying. In addition check assaying to an umpire laboratory was conducted. RNC began regularly inserting quality control samples beginning with drill hole 07-RN-04 (Lewis and San Martin, 2010), the fourth hole drilled by RNC on the Dumont Nickel project.

Field blanks consist of local esker sand and generally range in grade between 0.003 and 0.008 percent nickel (Lewis and San Martin, 2010), with an acceptable upper limit of 0.01% of nickel. Field duplicates consist of quarter core.

RNC used four certified control samples sourced from Ore Research & Exploration Pty Ltd. (ORE) of Victoria, Australia: OREAS 13P, OREAS 14P, OREAS 70P and OREAS 72A. OREAS 13P and OREAS 14P were replaced by OREAS 70P and OREAS 72A in 2008, as they were considered to be unrepresentative of the expected rock type and nickel grades (Lewis and San Martin, 2010).

OREAS 13P and OREAS 14P are both certified for copper, gold, nickel, palladium and platinum values. OREAS 70P is certified for a range of precious and base metals, and major and lithophile trace elements. OREAS 72A is certified for aluminium oxide, arsenic, chromium, cobalt, copper, gold, iron, magnesium oxide, nickel, palladium, platinum, silicon dioxide and sulphur. The certified nickel content of the reference material used on the project and the number of times they were assayed by the primary laboratory is presented in Table 11-5.

A certified reference material sample, a blank or a field duplicate sample were inserted into the sample stream at a rate of one every twenty-five samples (Lewis and San Martin, 2010).

Table 11-5: Specifications of Certified Reference Material Used by RNC between 2007 & 2012

Reference Material	Source	Ni (%)	Std. Dev. (%)	No. of Samples
OREAS 13P	ORE	0.204	0.0115	1,090
OREAS 14P	ORE	2.090	0.0700	159
OREAS 70P	ORE	0.244	0.0193	2,162
OREAS 72A	ORE	0.693	0.0250	243

Prior to June 1, 2008 all pulps prepared by Laboratoire Expert Inc. (Laboratoire Expert) were re-assayed at ALS Chemex Laboratory in Val-d'Or, Quebec (ALS). Since 1 June 2008, 5% of the pulps from ALS are randomly selected and re-assayed at Laboratoire Expert (Lewis and San Martin, 2010). Since June 2011, AGAT Laboratories Ltd. (AGAT Laboratories) in Mississauga is used as umpire laboratory.

Analytical control measures for magnetite as part of the EXPLOMIN^{TM} study involved replicate and duplicate analyzes by SGS Canada Inc. (SGS). Replicate analyzes consisted of re-plotting another sub-sample and re-running the analysis by QEMSCAN (Quantitative Evaluation of Materials by Scanning Electron Microscopy) for each replicate. The results show the reproducibility between sub-samples (including machine reproducibility). Duplicate analyzes consisted of analyzing the same block or polished section again, a second time. The results show the reproducibility of the system or equipment used. However, each time a block or polished section is re-analyzed, a different area on the block or polished section is scanned (i.e. not the exact same particles are scanned). Therefore, the original analyse can never be completely duplicated because the particles within the scanned areas may change due to slight movements in the stage and when setting up the analysis. Analytical control measures were performed on 5% of the EXPLOMIN^{TM} study.

In 2012, upon recommendation from SRK Consulting, RNC had SGS Mineral Services complete 153 Satmagan tests to independently validate the magnetite mineral abundances reported as part of the EXPLOMIN^{TM} mineral mapping program. Satmagan results of the EXPLOMIN^{TM} samples were used to validate the mineral mass percent of magnetite reported by QEMSCAN. Satmagan infers magnetite content by measuring magnetic susceptibility (Fe_3O_4 percent). Satmagan values (or recoverable Fe) can be compared and calibrated with Davis Tube Results. Figure 11.8 on the following page summarizes the Satmagan tests and compares the results to the magnetite mineral abundances reported by EXPLOMIN^{TM} results. Satmagan was performed on 10% of the EXPLOMIN^{TM} study.

11.3 SRK Comments

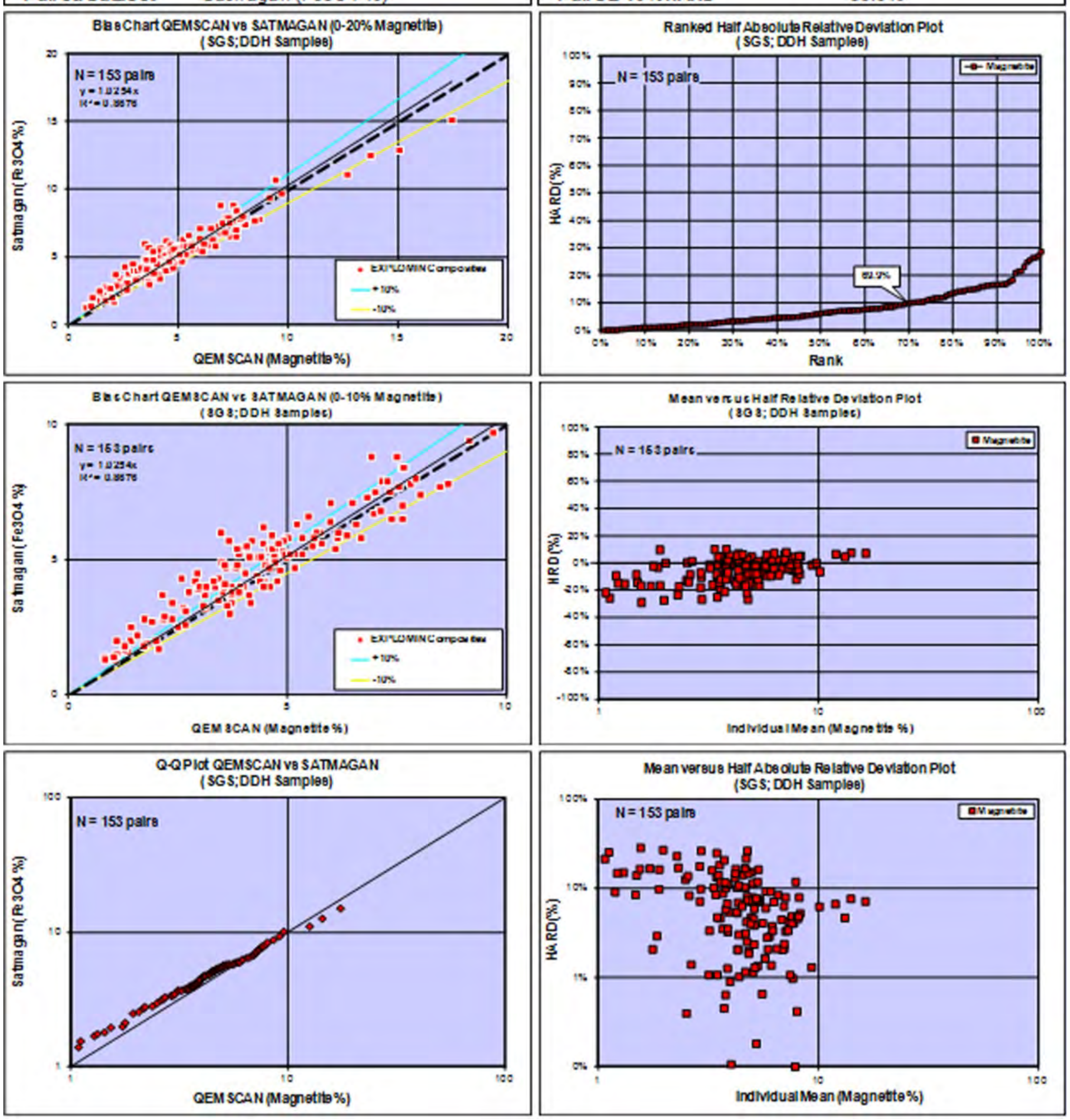
In the opinion of SRK, the sampling preparation, security and analytical procedures used by RNC are consistent with and often exceed generally accepted industry best practices.

Figure 11.8: Bias Charts, Quantile-Quantile & Precision Plots for EXPLOMIN™ Samples (QEMSCAN vs. Satmagan) (SGS) – Magnetite

srk consulting

Project	Dumont
Data Series	EXPLOMIN Composites
Data Type	DDH Samples
Commodity	Magnetite in %
Analytical Method	QEMSCAN and SATMAGAN
Detection Limit	0.05% (QEMSCAN)
Original Dataset	QEMSCAN (Magnetite %)
Paired Dataset	Satmagan (Fe3O4 %)

Statistics	QEMSCAN	Satmagan
Sample Count	153	153
Minimum Value	0.83	1.30
Maximum Value	17.44	15.10
Mean	4.73	5.09
Median	4.38	4.90
Standard Error	0.21	0.18
Standard Deviation	2.55	2.27
Correlation Coefficient	0.9580	
Pairs ≤ 10% HARD	69.9%	



Source: SRK

12 DATA VERIFICATION

12.1 Site Visit

In accordance with NI 43-101 guidelines, Sébastien Bernier from SRK visited the Dumont project between April 27 and May 2, 2011 accompanied by John Korczak, P.Geo; on May 17, 2013 he was accompanied by Robert Cloutier, Geo., OGQ both from RNC. The purpose of the site visit was to ascertain the geological setting of the project, witness the extent of exploration work carried out on the property, and assess logistical aspects and other constraints relating to conducting exploration work in this area.

All aspects that could materially impact the mineral resource evaluation reported herein were reviewed with RNC staff. SRK was given full access to all relevant project data. SRK was able to interview exploration staff to ascertain exploration procedures and protocols.

Borehole collars are clearly marked with metal stakes inscribed with the borehole number on a metal plate. No discrepancies were found between the location, numbering, or orientation of the boreholes verified in the field plans and the database examined by SRK.

The site visit was undertaken during active drilling and SRK examined core from numerous boreholes being processed in the core facility. SRK examined and relogged the nickel mineralized zone from Borehole 11-RN-242. SRK also collected verification samples from this borehole for independent assaying (see below).

On June 21, 2012, Sébastien Bernier and Oy Leuangthong from SRK accompanied by John Korczak and Michelle Sciortino from RNC visited the SGS facilities in Lakefield (Ontario) where EXPLOMIN™ samples are processed and analysed.

Full details of data verification completed by SRK and summarized herein are included in Bernier and Leuangthong (2013), which is available on RNC's website.

12.2 Database Verifications

Exploration data collected by RNC are incorporated directly into a CAE Mining Fusion database using electronic files only. Data collected by the logging geologists are recorded electronically into DHLogger, within the Fusion database management system. Samples tags are automatically and electronically generated by DHLogger. Both DHLogger and Fusion software are equipped with a series of rigorous internal checks that prevent entry errors, including duplications and missing intervals that may occur during logging and/or importing of assay data received electronically from the laboratory.

During the site visit, SRK reviewed and verified the logging procedures with several logging geologists. SRK also performed a series of statistical tests on the database as part of the mineral resource estimation process. No errors were found. SRK is of the opinion that the database is acceptable and sufficiently reliable for mineral resource estimation.

12.3 Verifications of Analytical Quality Control Data

RNC made available to SRK analytical control data as Microsoft Excel spreadsheets containing the assay results for the quality control samples (field blanks, field duplicates, certified reference material, check assays and replicate and duplicate analyses for the EXPLOMIN™ study).

SRK aggregated the assay results for the external quality control samples for further analysis. Eight variables were examined: calcium, cobalt, chromium, iron, nickel, palladium, platinum and sulphur, and specific gravity. Sample blanks and certified reference materials data were summarized on time series plots to highlight the performance of the control samples. Field duplicate, check assay, and replicate and duplicate analyses (as part of the EXPLOMIN™ study) (paired) data were analyzed using bias charts, quantile-quantile and relative precision plots. The analytical quality control data produced by RNC from 2007 to 2011 for the Dumont Nickel project are summarized in Table 12-1 and presented in graphical form (per element and per year) in Bernier and Leuangthong (2013), which is available on RNC's website.

SRK reports only cobalt, magnetite, nickel, palladium and platinum in the mineral resource statement; however, calcium, chromium, iron and sulphur were also modelled because of their correlation with nickel recovery. Although only cobalt, magnetite, nickel, palladium and platinum are discussed here, the comparative charts for all elements and minerals are included in Bernier and Leuangthong (2013), which is available on RNC's website for completion.

The external analytical quality control data produced for this project represents approximately 12% of the total number of samples submitted for assaying (Table 12-1).

Table 12-1: Summary of Analytical Quality Control Data Produced by RNC between 2007 & 2012

	2007	2008	2009	2010	2011	2012	TOTAL	%	Comment
Sample Count							**90,967**		
Quality Control Samples									
Field Blanks	628	966	520	107	1,428	63	3,712	**4.08%**	Esker sand (0.003-0.008% Ni)
Certified Standards	614	945	520	107	1,431	126	3,743	**4.11%**	
OREAS 13P	470	599					1,069	1.18%	ORE (0.2261% Ni)
OREAS 70P		302	456	88	1,310	61	2,217	2.44%	ORE (0.2438% Ni)
OREAS 72A		30	64	19	121	2	236	0.26%	ORE (0.693% Ni)
OREAS 14P	144	14					158	0.17%	ORE (2.09% Ni)
Field Duplicates	550	959	517	101	1,422	63	3,612	**3.97%**	Quarter Core
Total QC Samples	**1,792**	**2,870**	**1,557**	**315**	**4,281**	**252**	**11,067**	**12.17%**	
Check Assays									
Laboratoire Expert & ALS	135	14,411	5,503	182	934		**21,165**	**23.27%**	Pulp Duplicates
AGAT & ALS					761		**761**	**0.84%**	Pulp Duplicates

There are a number of field blanks above the acceptable upper limit of 0.01% nickel. However, SRK notes that this comprises approximately 2% of the total field blanks. Overall, the average value is approximately 0.0038%, indicating that the esker sand used as a blank is not barren in nickel, but sufficiently low for the purpose they are intended.

The field blank is not characterized for cobalt, palladium, or platinum. The cobalt mean of the blank samples is approximately 5 ppm (which is above the detection limit of 2 ppm for Laboratoire Expert and 1 ppm for ALS), indicating that the blank is also not barren in cobalt. Considering the average cobalt grade (at 0% cobalt cut-off) of the deposit is 105 ppm, the blank used is acceptable for cobalt.

The mean for palladium and platinum for the blank samples is less than the detection limit (2 ppm for Laboratoire Expert and 0.001 ppm palladium and 0.005 ppm platinum for ALS).

SRK notes that the blanks analyzed by Expert between 2007 and 2008 have higher means for cobalt, nickel, palladium, and platinum than the blanks analysed by ALS.

The time series plots for field blanks also show a high percentage of spikes above the mean (Bernier and Leuangthong, 2013).

OREAS 13P, OREAS 72A, and OREAS 14P control samples generally display mean grades lower than the expected nickel values. In particular, mean nickel grades for OREAS 13P deviate the most from the expected nickel value with approximately 91% of nickel assays below two standard deviations of the expected value. The exact cause for the poor performance of OREAS 13P is difficult to ascertain by SRK retrospectively. This should be investigated by RNC.

OREAS 13P and OREAS 14P control samples generally performed as expected for palladium and platinum, although between approximately 7% and 29% of samples plot outside of two standard deviations. OREAS 13P and OREAS 14P are not certified for cobalt.

OREAS 72A samples generally display mean grades close to two standard deviations from the expected value for cobalt. The exact cause for poor performance of OREAS 72A is difficult to ascertain by SRK retrospectively, but should be investigated by RNC.

Palladium and platinum performed within the expected ranges, although the mean grades were slightly below the expected value for OREAS 72A. Less than 7% and 2% of platinum and palladium samples plot outside of two standard deviations, respectively.

OREAS 70P generally performed within the expected range for nickel and cobalt. The nickel mean is slightly above the expected value, whereas for cobalt the mean is slightly less than the expected value. Cobalt had less than 1% and nickel had less than 2% outside of two standard deviations.

The mean palladium value for OREAS 70P is below the expected value and less than 1% spiked above the expected value. Platinum values are consistently above the expected range, which is below detection limit, for OREAS 70P, but with less than 2% above the detection limit.

Duplicate assay (paired) data analyzed by SRK show that assay results for cobalt and nickel can be reasonably reproduced by ALS from the same pulp. Rank half absolute difference (HARD) plots for cobalt and nickel show more than 95% of the field duplicate samples have HARD below 10% (Bernier and Leuangthong, 2013). This is expected from re-assaying the same pulp. HARD plots for palladium show between 51% and approximately 61% of the duplicate samples have HARD below 10%. HARD plots for platinum show between 55% to approximately 58% have HARD below 10%.

Check assay (paired) data for nickel analyzed by Laboratoire Expert between 2007 and 2009 generally agree with ALS results (see Bernier and Leuangthong, 2013). For samples assayed in 2010, and in particular 2011, SRK notes that there are significant departures between the two laboratories with Laboratoire Expert yielding consistently lower nickel in the 0.1% and 0.3% nickel grade range (see Bernier and Leuangthong, 2013). Further, there appears to be a gap between 0.2% and 0.3% nickel returned by Laboratoire Expert. Laboratoire Expert assay results are only used as checks and were not considered for resource estimation. It is difficult to analyze retrospectively the variance with the Laboratoire Expert check assay results, which is not accredited. SRK has recommended that RNC further investigates this discrepancy between ALS and Laboratoire Expert and change the umpire laboratory to an accredited facility.

In June 2011, RNC changed the umpire laboratory to AGAT Laboratories in Mississauga. Paired data for check assays show that assay results analysed by AGAT Laboratories generally agree with ALS results for cobalt and nickel with HARD plots showing more than 95% of the check assays have HARD below 10% (Bernier and Leuangthong, 2013). Check assay results since 2011 confirm that ALS results are not biased and are reliable. HARD plots for palladium and platinum show that 47% and 56% of the check assays have HARD below 10%, respectively. No check assays were sent to AGAT during February to December, 2012 because no resource boreholes were completed during this period.

A total of 78 replicate and 13 duplicate samples analyzed for magnetite as part of the EXPLOMIN™ study were analyzed by SRK. The replicate analysis show reasonable reproducibility between subsamples and the duplicate analysis show reasonable reproducibility of the machine. HARD plots for magnetite show that 56% of the replicate analyses and 100% of the duplicate analyses have HARD below 10% (Bernier and Leuangthong, 2013). The lower percentage value of the replicate analyses may indicate a nugget effect of the magnetite particles. The Satmagan data show reasonable reproducibility with the QEMSCAN data with 70% of the samples having HARD below 10%.

Overall, SRK considers that analytical quality control data reviewed by SRK suggest that the assay results delivered by the primary laboratory used by RNC are sufficiently reliable for the purpose of mineral resource estimation. Other than indicated above, the data sets examined by SRK do not present obvious evidence of analytical bias.

12.4 Independent Verification Sampling

As part of the verification process, SRK collected eighteen verification samples during the site visit completed between April 27 and May 2, 2011. The verification samples replicate RNC sample intervals from Borehole 11-RN-242 drilled in 2011. The verification samples comprise of NQ quarter core and were sent to AGAT Laboratories in Mississauga in May 2011 for preparation and assaying. AGAT Laboratories is accredited to Standard ISO/IEC 17025:2005 standards for specific testing procedures by the Standards Council of Canada (SCC) and the Canadian Association for Laboratory Accreditation Inc. (CALA), including those used to assay the samples submitted by SRK (four acid digestion using inductively coupled plasma-optical emission spectroscopy).

Table 12-2 on the following page shows the comparative assay results for the verification samples. The assay certificate for the SRK samples is included in Bernier and Leuangthong (2013), which is available on RNC's website. The verification samples (paired data) were also analyzed using bias charts, quantile-quantile and relative precision plots. The verification samples show that for nickel, sulphur and specific gravity, ALS results can be reasonably reproduced by AGAT. HARD plots show 89% for nickel, 72% for sulphur and 100% for specific gravity, have HARD below 10%.

Such a small sample collection cannot be considered representative to verify the nickel grades obtained by RNC. The purpose of the verification sampling was solely to confirm that there is nickel mineralization and verify that SRK can reproduce nickel grades for the sample intervals independently chosen by SRK.

Table 12-2: Assay Results for Verification Samples Collected by SRK

Borehole ID	SRK Sample ID	Original Sample ID	From (m)	To (m)	Length (m)	Original Ni (%)	SRK Ni (%)
11-RN-242	SRK-01	11-RN-242-213	495.00	496.50	1.50	0.736	0.791
11-RN-242	SRK-02	11-RN-242-217	496.50	498.00	1.50	0.526	0.526
11-RN-242	SRK-03	11-RN-242-218	498.00	499.50	1.50	0.495	0.518
11-RN-242	SRK-04	11-RN-242-219	499.50	501.00	1.50	0.495	0.457
11-RN-242	SRK-05	11-RN-242-220	501.00	502.50	1.50	0.415	0.347
11-RN-242	SRK-06	11-RN-242-221	502.50	504.00	1.50	0.391	0.390
11-RN-242	SRK-07	11-RN-242-222	504.00	505.50	1.50	0.363	0.395
11-RN-242	SRK-08	11-RN-242-223	505.50	507.00	1.50	0.380	0.263
11-RN-242	SRK-09	11-RN-242-227	507.00	508.50	1.50	0.348	0.349
11-RN-242	SRK-10	11-RN-242-228	508.50	510.00	1.50	0.389	0.340
11-RN-242	SRK-11	11-RN-242-229	510.00	511.50	1.50	0.329	0.335
11-RN-242	SRK-12	11-RN-242-230	511.50	513.00	1.50	0.275	0.288
11-RN-242	SRK-13	11-RN-242-231	513.00	514.50	1.50	0.308	0.283
11-RN-242	SRK-14	11-RN-242-232	514.50	516.00	1.50	0.306	0.283
11-RN-242	SRK-15	11-RN-242-233	516.00	517.50	1.50	0.253	0.354
11-RN-242	SRK-16	11-RN-242-234	517.50	519.00	1.50	0.235	0.253
11-RN-242	SRK-17	11-RN-242-235	519.00	520.50	1.50	0.242	0.236
11-RN-242	SRK-18	11-RN-242-236	520.50	522.00	1.50	0.258	0.248
Average						**0.375**	**0.370**

13 MINERAL PROCESSING & METALLURGICAL TESTING

13.1 Introduction

The objective of the feasibility metallurgical study was to quantify the metallurgical response of the Dumont ultramafic nickel mineralization. The program was designed to develop the parameters for process design criteria for ore flow characteristics, comminution, desliming, flotation, and dewatering in the processing plant.

The metallurgical program was conducted by Centre de Technologie Minérale et de Plasturgie Inc (CTMP), Mineral Solutions, SGS Mineral Services (SGS) and RNC metallurgical staff.

The metallurgical program was performed on the following composites and samples:

- metallurgical variability samples;
- mineralization composites (sulphide, alloy and mixed);
- metallurgical domain composite samples;
- outcrop sample; and
- grindability samples.

The samples were selected to represent the spatial distribution, ore grade and mineralization types of the Dumont deposit.

Ninety-two grindability samples were submitted to SGS to complete a suite of grinding characterization tests including Bond ball work index (BWi), Bond rod work index (RWi), SMC test, and abrasion index (Ai). In addition to these 92 samples, 10 additional samples were added from the PQ variability samples to complete crusher work index (CWi) and JK Drop Weight Tests (JK DWT).

Flotation and magnetic separation studies were performed from 2008-2009. This work generated the standard test procedure (STP) which was used to establish metallurgical domains and recovery variability throughout the deposit.

Further optimization work was conducted on the mineralization composites and the PQ variability samples to optimize reagent consumption, flowsheet design and complete locked cycle tests to assess cleaning recovery.

13.2 Previous Testwork

Several rounds of testing have been undertaken at various laboratories prior to the current phase of study. A summary of the programs and results are described below.

13.2.1 Historical Testwork 1971-1972

In 1971 and 1972, Centre de Recherches Minérales, Ste-Foy, Quebec (CRM) conducted a laboratory testwork program on drill core samples from the main zone at the request of Dumont Nickel. The following details have been extracted from a report (Caron, 2004) that used the information contained in the metallurgical section of the historical Caron, DuFour and Seguin (CDS) feasibility study (Caron, 1972).

The CRM metallurgical testwork resulted in the development of a concentration process consisting of grinding, flotation and magnetic separation. Locked-cycle tests were conducted on drill core composite samples.

In the report, Caron considered that the process described in the CDS study would yield a 48% nickel recovery in a concentrate grading approximately 20% nickel.

13.2.2 Preliminary Metallurgical Testwork 2007-2008

Preliminary metallurgical testwork was undertaken in 2007 and early 2008 by RNC. The focus was a conventional wet grind to very fine P_{80} of 53 µm, followed by flotation and magnetic separation. The brucite and chrysotile in the feed caused significant viscosity issues and pasting during grinding. A relatively complex and expensive reagent scheme was developed to attempt to reduce the viscosity and achieve acceptable metallurgy.

In late 2008, the metallurgical program shifted direction, concentrating on pre-treatment of the mineralization by first removing chrysotile in a dry defibring step followed by removal of brucite in a wet desliming stage in an effort to reduce the pulp viscosity and simplify the reagent scheme. The pulp viscosities decreased significantly to improve nickel recoveries and concentrate grades in the magnetic separation and flotation processing that followed. Testwork was completed on ten complete hole composites that were taken from across the deposit and represented the variability observed from the mineralogy. A rigorous standard test procedure was developed. Thirty-two different samples were then evaluated with the STP and used to define both the recovery equations for the three ore types in the orebody.

13.2.2.1 Dry Crushing & Defibring

RNC contracted the Centre de Technologie Minérale et Plasturgie (CTMP), a crown corporation of the Government of Quebec with laboratories in Thetford Mines, QC, to undertake dry crushing, screening, and air classification testwork.

CTMP tested these samples through the standard regime for separating and recovering chrysotile used in the asbestos industry in Quebec. At 841 µm (20 mesh), the separation of chrysotile from the granular serpentine was mostly complete and simple air classification removed a chrysotile product depleted of nickel. The intensity of the air classification determined the weight loss to the chrysotile product and the nickel loss.

13.2.2.2 Wet Grinding & Desliming

Testing of a dry crushed and air classified sample (chrysotile removed) still yielded pulps of high viscosity, which continued to interfere with grinding and flotation. CTMP was successful in stage grinding these products with concurrent desliming in hydrocyclones. The desliming was thought to remove the interstitial brucite which was liberated in grinding and was the principal cause of the high pulp viscosities. CTMP first ground the coarse air classification underflow (U/F) to 80% minus 150 µm (100 mesh) and deslimed in a hydrocyclone. The hydrocyclone overflow (O/F) was discarded. The coarse, free flowing U/F was sent to a low intensity wet magnetic separator for awaruite and magnetite recovery. The non-magnetics ground to 80% minus 74 µm (200 mesh) and the ground pulp again deslimed in hydrocyclones to remove the brucite slime liberated in the second grind. The second stage desliming U/F was again treated on the magnetic separator to scavenge any remaining awaruite and magnetite liberated in the second stage grinding. The non-magnetics were sent to conditioning for flotation. This procedure gave consistently low viscosity pulp for flotation.

Recovery of awaruite and magnetite was excellent in the two rougher magnetic separation stages at grades averaging 1% nickel but as high as 3% nickel and 40% to 50% iron. awaruite recoveries averaged about 80%. Further work on cleaning these awaruite rougher concentrates was deferred until development of the standard test procedure was complete.

It was determined to control the wet desliming to a target mass loss of approximately 5% for each stage resulting in nickel losses of less than 4% per stage. No improvements in pulp viscosity were observed for higher mass losses but nickel losses increased significantly.

13.2.2.3 Flotation Testwork

Flotation testwork was completed on defibred and deslimed composite by Lang and Liu at SGS, Marois at CTMP and Marois, Liang and Lang at CTMP in 2008. The purpose of the testwork was to test if defibring and desliming prior to flotation would allow simplifying of the reagent scheme and reductions in reagent consumption and costs yielding equivalent or better recoveries and concentrate nickel grades.

The standard test procedure was finalized in May 2009. It consisted of the staged grind described in Section 13.2.2.2, with each grind and deslime followed by incremental timed flotation tests.

13.2.2.4 Comminution Testwork

Point load tests on hundreds of samples in the RNC core shack in Amos had reduced the number of primary ore types in term of their breakage and hardness characteristics to four:

- Domain 1 – Samples from the relict olivine zone;
- Domain 2 – Samples from the coalingite zone;
- Domain 3 – Samples from the black competent serpentinite; and
- Domain 4 – Samples from fault zones with strong alteration.

The four domain samples were sent to Hazen Research in Denver, Colorado to have full JK DWT, SMC test, and unconfined compressive strength (UCS) tests performed.

The data in Tables 13-1 to 13-3 are summarized from the Hazen report (Gillespie, 2010).

Table 13-1: JK Drop Weight Tests Summary

	Domain 1	Domain 2	Domain 3	Domain 4
Specific Gravity	2.60	2.43	2.61	2.60
Axb	51.9	74.2	62.2	68.8
t_a	0.54	0.75	0.64	0.88

Table 13-2: SMC Summary

	Domain 1			Domain 4			Domain 3			Domain 4		
Specific Gravity	2.59	2.59	2.62	2.44	2.44	2.42	2.61	2.62	2.61	2.54	2.51	2.52
Axb	35.3	40.8	43.4	63.4	64.9	63.5	52.7	56.6	42.7	46.4	49.9	55.9
t_a	0.35	0.41	0.43	0.66	0.69	0.68	0.52	0.56	0.42	0.47	0.52	0.57

Table 13-3: UCS Summary

Domain	Compressive Strength (psi)
1	9,190
1	16,370
2	7,570
2	4,490
3	10,620
3	11,640
4	14,390
4	7,240

13.2.3 Pre-feasibility Study (PFS)

During the pre-feasibility data collection phase, both laboratory and mini-plant work were completed. The mini-plant was a 20 to 30 kg/h continuous plant that emulated the lab rougher circuit (crushing, defibring, wet grinding, desliming, flotation and magnetic separation). The crushing and defibring were batch processes, while the wet grinding through magnetic separation was continuous and operated at 20 to 30 kg/h depending on the campaign.

The mini-plant was initially commissioned to add additional confidence to the rougher recovery seen in the laboratory STP results. The feed to the mini-plant was from the four PQ holes that were drilled along the length of the deposit to provide greater quantities of material per domain sample than the laboratory scale tests.

The mini-plant also generated higher quantities of rougher concentrate (both flotation and magnetic concentrates) to allow initial cleaning circuit design work.

The laboratory work focused on grindability testing, variability analysis (STP), cleaning recoveries and flowsheet optimization. Much of this work is discussed fully in the feasibility test results section as the feasibility work built on this base. The testwork performed during the PFS modified the flowsheet significantly from a dry crushing- defibring circuit assumed in the 2010 Preliminary Economic Analysis (PEA). The following section provides a summary of the testwork that led to the decision to eliminate the dry circuit and move forward in the PFS with a SAG, Ball Mill and desliming circuit.

13.2.3.1 Elimination of Defibring

In the 2010 PEA, the base case assumption for the flowsheet was a dry crushing circuit with defibring followed by a wet ball mill and desliming. Initially the defibring was introduced to remove the chrysotile fibres which were causing matting and viscosity issues in flotation. However, even with the introduction of defibring, viscosity problems were still evident in the ball mill discharge/flotation feed so a desliming step was introduced. At this point it was still assumed that a very fine grind of 53 µm was required to achieve maximum nickel recovery.

The results of the STP tests show that the majority of the nickel floated after the 150 µm (100 mesh) grind. Further grinding, while increasing mass recovery to the concentrate, did not significantly increase the recoverable nickel in the concentrate. With this coarser grind, it was decided that desliming and no defibring needed to be re-evaluated as part of the PFS metallurgical program. This approach is common practice in Australian ultramafic nickel ore treatment. The dry crushing circuit was extremely complex with a high operating cost, as well as having potential health and safety concerns from dusting in the plant during the dry quaternary crushing stages. There would be numerous advantages to proceeding with a wet grind and deslime circuit only and eliminating the initial dry crushing stage.

Tests were performed on three mineralization composites (sulphide, mixed and alloy) created from the PQ core that was drilled for the mini-plant.

Each composite was tested under the STP flowsheet (defibring, grinding, desliming, flotation and magnetic separation). The results were then compared to a grinding, desliming, flotation and magnetic separation flowsheet (see Table 13-4 for the results). The performance of the desliming without defibring was equal to or better than the STP performance for each case. In addition a test was performed on each with no desliming or defibring, overall the recoveries were similar but the rougher concentrate grade improvement was significant.

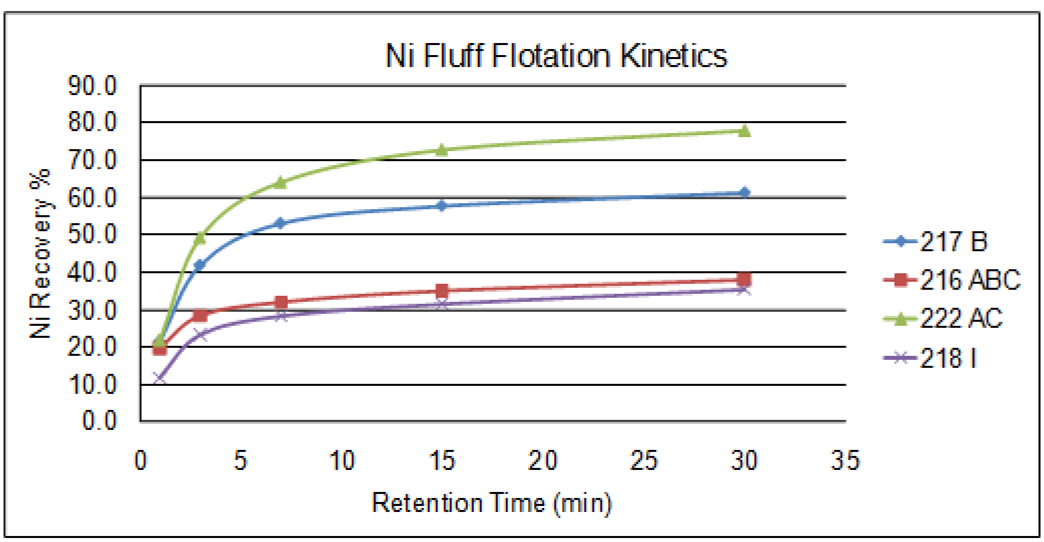
The results from both the laboratory and mini-plant confirm that at the coarser grind, with a P_{80} at or above 150 µm, defibring is not required for successful grinding and nickel flotation for any of the mineralization types. The most effective unit operation for improving flotation performance is an aggressive desliming stage to remove the fine particles that cause viscosity problems in the rougher stage.

13.2.3.2 Recovery from the Fibre Portion of STP samples

The initial 70 STP samples had defibring performed on them. The flotation of this product showed similar nickel recovery to the STP rougher recovery. The P80 of the fibre product is 180 µm which is very similar to the P80 of the rougher grind.

The size distribution of the fluff product is around 180 µm, or similar to P80 of the cyclone underflow after desliming. This material is not expected to report to the slimes fraction, as the size distribution is similar to the overall mill discharge. Tests have been performed on this material which showed similar recovery to the STP rougher recovery. A graph with a range of samples is shown below. For example the STP recovery for 222AC was 72.2%, 217B was 52.9%, 218I was 42.3% and 216ABC was 36.4% which are similar to the recoveries shown in the fluff testwork shown in Figure 13.1.

Figure 13.1: Recovery from Fluff Portion of STP



Source: RNC.

13.3 Feasibility Sample Selection

13.3.1 Comminution Samples

Two different types of samples were selected for the comminution data collection: 92 half NQ core samples and 10 whole PQ core samples, for a total of 102 samples. The larger diameter PQ samples were chosen to complete crusher work index test as well as drop weight tests, both of which require core larger than 63 mm in diameter.

75 of these samples formed the basis for the pre-feasibility study and were previously reported in the June 22, 2012 Dumont Technical Report. 27 samples were added to complete the dataset for the feasibility study. The results from all 102 samples formed the basis for the feasibility study.

13.3.2 Metallurgical Feasibility Samples

The metallurgical testwork program at CTMP and Mineral Solutions was performed on the following composites and samples:

- metallurgical variability samples (STP samples);
- PQ variability samples;
- mineralization and metallurgical domain composites; and
- outcrop sample.

Information on how these samples were selected is available in Section 11.

13.3.2.1 Mineralization Type Composites

Three mineralization composites were generated from the PQ metallurgical variability samples. The following factors were considered when forming the composites:

- spatial relationship (structural domain and depth);
- mineralization type; and
- Ni grade.

In Tables 13-4 to 13-6, each individual component is listed along with the combined weight and grade of the composite. Once the composite was blended and crushed a sample of each was sent for mineralogy and assay.

Table 13-4: Sulphide Composite Composition

Sample	Wt (kg)	From	To	Structural Domain	% Ni
10-RNC-222B	300.4	51.0	73.5	5	0.25
10-RNC-217A	17.5	43.6	45.0	3	0.55
10-RNC-218E	303.0	83.5	88.0	4	0.28
10-RNC-217EG	559.5	225.0	250.0	3	0.25
10-RNC-222DE	367.3	108.0	133.5	5	0.27
10-RNC-218BDF	250.0	44.5	83.5	4	0.64
		88.0	110.5		
10-RNC-222AC	250.0	29.3	51.0	5	0.37
		73.5	108.0		
Total	2,200.7				0.34

Source: RNC.

Table 13-5: Alloy Composite Composition

Sample	Wt (kg)	From	To	Structural Domain	% Ni
10-RNC-216E	1,429.6	153.0	246.0	3	0.26
10-RNC-222H	706.8	199.5	252.0	5	0.21
10-RNC-218I	200.0	151.0	201.0	4	0.23
Total	2,336.4				0.24

Source: RNC.

Table 13-6: Mixed Composite Composition

Sample	Wt (kg)	From	To	Structural Domain	% Ni
10-RNC-216B	252.0	127.5	153.0	3	0.27
10-RNC-217C	481.2	153.0	171.0	3	0.29
10-RNC-222F	205.8	133.0	153.0	5	0.26
10-RNC-218AC	1,103.7	26.5	44.5	4	0.25
		52	74.5		
Total	2,042.7				0.26

Source: RNC.

In 2012, additional domain composites, for each of the metallurgical domains, were created to provide material for flowsheet optimization work, specifically around the desliming circuit. The composition of each sample is listed below in Tables 13-7 to 13-13.

Table 13-14 summarizes the feed assay and mineralogy for each composite.

Table 13-7: Comp 1: High Iron Serpentine – Higher Recoverable Ni

Sample	Wt (kg)	From (m)	To (m)
RNC-216_D	50	127.5	153
RNC-216_E	50	153	246
08-RN-109	26	60.5	90.5
07-RN-48	31	376.5	415.5
08-RN-103	30	261	297
08-RN-105	25	180	214.5
07-RN-47	12	210.5	240
07-RN-16	18	148.5	180
09-RN-170	20	150	180
07-RN-20	21	194.5	225.13
08-RN-60	16	130.5	160.5
07-RN-10	21	338	368
RNC-216ABC	50	Not applicable composite sample	

Table 13-8: Comp 2: High Iron Serpentine - Lower Recoverable Ni

Sample	Wt (kg)	From (m)	To (m)
RNC-217_A	50	43.6	63
RNC-217_B	50	63	153
RNC-217_EG	50	225	250
RNC-217_H	50	204	216
08-RN-83	14	112.5	144
07-RN-14	25	260	293
07-RN-45	7	300	330
07-RN-45	13	178.5	210
08-RN-60	14	48	78
08-RN-101	25	399	429
07-RN-20	13	56.5	87
08-RN-83	22	220.5	252
07-RN-47	12	55	86
08-RN-109	13	213.5	243.5

Table 13-9: Comp 3: Mixed Sulphide

Sample	Wt (kg)	From (m)	To (m)
09-RN-213A	23	53.1	86.5
09-RN-214A	15	238.5	259.8
09-RN-214I	14	489	502.5
09-RN-214K	25	510	575
09-RN-223D	25	130.5	189
09-RN-223F	25	238.5	276
09-RN-224B	19	61.5	81
09-RN-224D	15	94.5	118.5
09-RN-224E	15	118.5	143
09-RN-224F	28	143	189
07-RN-14	13	51	85
07-RN-43	22	68	103.5
08-RN-120	24	402	438
08-RN-129	13	78	107.5
08-RN-37	22	97.5	138
08-RN-79	14	35	63
08-RN-79	14	84	114
09-RN-156	9	216	246
09-RN-156	10	312	342

Table 13-10: Comp 4: Pn Dominant – Higher Recoverable Ni

Sample	Wt (kg)	From (m)	To (m)
09-RN-213B	15	86.5	115.95
09-RN-213C	15	115.95	148.5
09-RN-213D	15	148.5	165
09-RN-213E	15	165	199.5
09-RN-213F	15	207.24	252
09-RN-213G	15	252	274.49
09-RN-213H	15	274.49	314.5
09-RN-214B	15	261	294
09-RN-214C	15	294	328
09-RN-214D	15	328	338
09-RN-214E	15	338	385.5
RNC-217_A	15	43.6	63
07-RN-35	37	129	168
07-RN-39	21	69	99.8
07-RN-48	38	174	210
08-RN-101	19	196.5	226.5
08-RN-130	38	105	114

Table 13-11: Comp 5: Pn Dominant – Lower Recoverable Ni

Sample	Wt (kg)	From (m)	To (m)
09-RN-213I	41	314.5	351
09-RN-224G	43	189	270
07-RN-10	13	123.5	153.5
09-RN-196	30	445.5	480
07-RN-39	28	213.5	249
07-RN-45	17	60	90
08-RN-111	34	251.5	287.5
08-RN-37	36	261	300
08-RN-58	26	216	250.4
08-RN-129	20	193	223
RNC-218_A	29	1	2

Table 13-12: Comp 6: Hz Dominant – Higher Recoverable Ni

Sample	Wt (kg)	From (m)	To (m)
08-RN-123E	25	400.5	426
08-RN-124D	25	462	490.5
08-RN-146G	25	262.5	291
09-RN-161D	25	358.5	366
09-RN-181E	25	376.5	391.5
09-RN-219B	25	102.5	120
09-RN-219E	25	145.5	160.5
10-RN-228B	25	310.5	331.5
11-RN-274G	25	500.5	515.5
11-RN-363F	25	388.5	429
11-RN-366C	25	147	195
Outcrop	25		
222BDE	25		

Table 13-13: Comp 7: Hz Dominant – Lower Recoverable Ni

Sample	Wt (kg)	From (m)	To (m)
08-RN-108C	20	127.5	166.5
08-RN-110C	20	153	187.5
08-RN-123A	20	280.5	319.2
08-RN-124B	20	364.5	412.5
08-RN-146B	20	159	183
09-RN-161C	20	309	358.5
09-RN-219A	20	41	102.5
09-RN-219D	20	133.5	145.5
09-RN-220A	20	66	90
09-RN-220D-G	20	145.5	267
10-RN-228A	20	271.5	310.5
11-RN-274G	20	385	464.5
11-RN-366E	20	216	277.5
11-RN-366F	20	277.5	310.5
11-RN-379A	20	253.5	298.5
08-RN-71	20	9	84

Table 13-14: Feed Assay & Mineralogy for Each Composite

	Ni ppm	% S	Aw (%)	Pn (%)	Hz (%)	Fe Serp (%)
Comp 1	2600	0.05	0.16	0.17	0.02	32.4
Comp 2	3390	0.21	0.09	0.55	0.01	34.3
Comp 3	2830	0.08	0.11	0.06	0.18	8.8
Comp 4	3170	0.2	0.13	0.35	0.08	7.7
Comp 5	2310	0.05	0.19	0.10	0.04	4.8
Comp 6	3020	0.11	0.07	0.01	0.19	7.1
Comp 7	2720	0.05	Pending at time of report			

13.3.2.2 Outcrop Sample

A bulk sample (2 to 3 tonnes) was gathered from a large outcrop in the southeast portion of the Dumont deposit. The area is contained within the southern extent of the pit shell. The sample is primarily a sulphide sample, dominated by heazlewoodite. The grade of the sample is 0.41% Ni and 0.15% S. The 3 tonne sample was collected from previously blasted material, crushed and blended to create individual charges for both mini-plant and laboratory testwork. Representative portions of the sample were split out and sent for QEMSCAN, STP recovery test and assay to characterize the sample.

13.4 Ore Flow Characteristics

A composite sample was sent to Jenike and Johanson (J&J) for flow testing. Eight tests were performed on a composite of -2,380 µm (-8 mesh) material. The composite was composed of samples from the grindability work that represented the various metallurgical domains from the Dumont deposit: 2 kg of GRO-67, 3 kg of GRO-69, 5 kg of GRO-70, 2 kg of GRO-72, 3 kg of

GRO-74, 2 kg of GRO-76, 3 kg of GRO-78, 3 kg of GRO-88, 2 kg of GRO-90, 2 kg of Comp 2, 3 kg of Comp 3, and 3 kg of Comp 4 were combined and blended for the work.

The eight tests are listed below.

- particle density;
- compressibility;
- loose and compacted bulk density;
- flow function;
- wall friction;
- critical chute angle; and
- frozen unconfined strength.

The following discussion is a summary of the J&J report (Hui and Holmes, 2012).

Samples were tested at two different moisture contents, which represented 60 and 80% saturation. Saturation for the Dumont material was determined to be at 17.1% moisture. Testing was completed at 10.2% and 13.5% moisture. This is done so that the testing reflects the conditions under which the ore is expected to be most difficult to handle.

The particle density was calculated as 2.63 g/cm^3 for this sample. The loose bulk density and compacted bulk density were 1,440 and 1,726 kg/m^3, respectively.

The material is somewhat cohesive and had the ability to form a rathole if stored in a funnel-flow bin. It was recommended that the material be stored in a mass flow bin with a minimum recommended outlet diameter of 400mm to prevent cohesive arching. The strength of the fines was not significantly affected by storage time at rest or the change in moisture content.

Wall flow and chute flow tests were conducted to determine maximum wall and chute angles to achieve mass flow. The results varied depending on the liner material tested and time at rest and also demonstrated that the material is slightly sensitive to impact pressure and a low drop height is recommended to minimize the impact of material falling into the chute.

The unconfined yield strength of the frozen ore increases with increasing moisture contents and there is a high risk of arches forming at moistures greater than 3%.

13.5 Comminution Circuit Characterization Testwork

The testing consisted of both grindability testwork to characterize the competency, hardness and abrasion of the Dumont material as well as slurry rheology.

Several shipments of drill cores were sent to SGS, Lakefield site, from January 2011 to March 2012. Ten full PQ and 92 half NQ drill core samples were sent for testing. The ten full PQ samples were submitted for:

- Bond low-energy impact Test (CWi);
- JK Drop Weight Test (JK DWT);
- SMC test (SMC);
- Bond rod mill work index test (RWi);

- Bond ball mill work index test (BWi); and
- Bond abrasion test (Ai).

The 92 half NQ drill core samples were submitted for the same suite of tests with the exception of the Bond low-energy impact test and the JK DWT. The preparation of these drill core samples is shown in Section 11.

The samples submitted for Bond ball mill work index testing were also submitted for the ModBond test to establish the ModBond – BWi correlation parameters.

13.5.1 Grindability Testwork Results

The summary of the results of the grindability tests for the comminution variability samples are shown in Table 13-15. The feasibility design basis was based on the 102 samples, which includes the 75 samples previously reported in the June 22, 2012 43-101 Technical Report and the 27 samples that were added for the feasibility study to fill in spatial gaps in the deposit. The following discussion is a summary of the results from two SGS grindability reports (Verret and Imeson 2011 and Patsius and Imeson, 2013).

Table 13-15: Summary SMC & Work Index Statistics

Statistics	JKTech Parameters				Work Indices					Ni Grade
	Axb smc	**DWI** kWh/m^3	**T_{10} @** 1 kWh/t	**Rel. Density**	**CWi** kWh/t	**RWi** kWh/t	**BWi** kWh/t	**Mod.** kWh/t	**AI** g	%
Results Available	102	102	102	102	10	101	11	102	102	102
Average	**53.8**	**4.91**	**38.3**	**2.57**	**13.5**	**14.9**	**20.1**	**20.9**	**0.009**	**0.29**
Std Dev.	8.6	0.89	4.5	0.06	2.5	1.1	1.6	1.0	0.027	0.06
Rel. S. D. (%)	16	18	12	2	19	8	8	5	313	21
Min	81.1	3.19	61.6	2.44	10.0	11.6	17.1	17.7	0.000	0.18
10th Percentile	63.1	4.08	42.9	2.48	10.7	13.7	18.3	19.5	0.000	0.24
25th Percentile	9.3	4.35	40.9	2.54	11.6	14.2	19.1	20.1	0.000	0.26
Median	**54.6**	**4.71**	**38.5**	**2.58**	**13.1**	**14.7**	**20.3**	**20.8**	**0.002**	**0.28**
75th Percentile	47.6	5.33	35.8	2.61	15.3	15.6	20.9	21.2	0.007	0.32
90th Percentile	43.6	5.91	33.3	2.63	16.0	16.3	22.3	22.0	0.014	0.35
Max	31.0	8.34	26.7	2.73	18.0	18.2	22.4	23.0	0.215	0.52

Note: Min and Max refer to Softest and Hardest for the grindability tests. **Source:** RNC.

Overall, the ore depicted an increase in hardness with finer size, which is typical for many ores. The majority of the test results (percentile 10^{th} to 90^{th}), for the tests performed at coarse size (JK DWT and the SMC test) ranged from moderately soft to medium. At medium size (Bond rod mill test) the majority of the samples fell in the medium to moderately hard range. At fine size (Bond ball mill work index and modified Bond tests), the bulk of the test results fall within the hard to very hard range. The Bond low-energy impact test is the exception; the test uses the coarsest rocks, but the sample tested were categorized as moderately hard to hard. The relative standard deviation of test results within each series ranged from 5% to 19%, which is considered narrow in comparison to other deposits.

The presence of fibrous material was challenging for the dry grindability tests, especially for the completion of the Bond ball mill test. The Bond rod mill test, with a closing screen size of 14 mesh, was not affected by the fibres. This issue is common to a number of other ultramafic Ni deposit. Adjustments are typically made by the engineer in the interpretation of the data for mill selection.

The accumulation of fibres in the plant ball mill circulating loads is not expected to pose the same problems that were observed with the Bond ball mill grindability tests. The plant ball mill circuit will be closed with hydrocyclones, and the fibres will preferably report to the cyclone overflow due to their low density and shape factor.

13.5.2 SMC & JK Drop Weight Tests

The SMC test is an abbreviated version of the standard JK DWT performed on rocks from a single size fraction (-22.4/+19 mm in this case). The SMC test was performed on a total of 102 samples.

The majority of the Axb parameters, corresponding to resistance to impact breakage, ranged from 63.1 (10^{th} percentile) to 43.6 (90^{th} percentile), and covered the moderately soft to medium range with the average (53.8) and median (54.6) values falling in the medium category. The relative density of all the samples averaged 2.57.

The JK DWT was performed on ten samples. The data was interpreted by Contract Support Services (CSS), the North American agent for JKTech. The ten samples submitted for the DWT were also subjected to the SMC test for calibration purposes.

The DWT samples generally fell in the soft to medium range in terms of resistance to impact breakage (Axb) and resistance to abrasion breakage (t_a). Most of the DWT and SMC pairs were similar in terms of resistance to impact breakage (Axb) and relative density, while the t_a presented more variation

13.5.3 Bond Low-energy Impact Test & Bond Rod Mill Grindability Test

The Bond low-energy impact test determines the Bond crusher work index (CWi), which can be used with Bond's Third Theory of comminution to calculate power requirements for crusher sizing. For each of the ten samples tested, twenty rocks in the range of 2 to 3 inches were shipped to Phillips Enterprises LLC for the completion of the Bond low-energy impact test. The average CWi was13.5 kWh/t with a range of 10 to 18 kWh/t.

The Bond rod mill grindability tests were performed at 14 mesh of grind (1,180 µm) on the 102 samples.

Eighty percent of the Bond Rod mill work indices (RWi) ranged from 13.7 kWh/t (10^{th} percentile) to 16.3 kWh/t (90^{th} percentile), covering the medium to moderately hard range of hardness. The median RWi was 14.7 kWh/t, which falls in the medium range of hardness.

13.5.4 Bond Ball Mill Grindability Test

The Bond ball mill grindability test (BWi) was performed with a closing screen of 177 µm (80 mesh) on 11 samples to achieve a P_{80} of approximately 150 µm.

Eighty percent of the BWi ranged from 18.3 kWh/t (10^{th} percentile) to 22.3 kWh/t (90^{th} percentile), covering the hard to very hard range of hardness. The median BWI was 20.3 kWh/t, falling in the hard range of hardness.

13.5.5 ModBond Test

The ModBond test consists of a single batch test, which is calibrated against the standard Bond ball mill grindability test results. The ModBond tests were calibrated at 177 µm (80 mesh). The ModBond tests were performed on all 102 samples.

Eighty percent of the ModBond work index ranged from 19.5 kWh/t (10^{th} percentile) to 22.0 kWh/t (90^{th} percentile), covering the hard to very hard range of hardness. The median ModBond work index was 20.6 kWh/t, which falls in the hard range of hardness.

13.5.6 Bond Abrasion Test

All the 102 samples were submitted for Bond abrasion testing. All the AI values were below 0.090 g, except for sample 08-RN-138-GR061, which yielded an AI of 0.215 g. The median AI was 0.002 g. These values indicate very low abrasion for Dumont ores, typical of other ultramafic orebodies.

13.6 Metallurgical Variability Test Results

Variability samples were selected from the Dumont mineralization and underwent both rheology and recovery characterization testwork (STP). The rheology work was performed by SGS Minerals at their Lakefield site. The recovery variability testing was performed by CTMP in Thetford Mines.

13.6.1 Rheology

All of the 102 grindability samples underwent rheology testing. The samples were pulverized to -125 µm (120 mesh) for testing. All testing was done without reagents or desliming. The summary information reported in this section is from the two SGS reports (Ashbury and Mezei, 2011 and 2013).

13.6.1.1 Rheology Benchmark Samples

Three samples were chosen for benchmark testing (Table 13-16). The samples were picked based primarily on brucite and olivine content. Brucite is known to cause viscosity issues in slurries and olivine is a marker for degree of serpentinization, which is known to impact other metallurgical characteristics of the ore.

- GR018 – 10-RN-218AC Med Brucite – Low Olivine
- GR023 – 10-RN-216E Low Brucite – High Olivine
- GTR027 – 10-RN-222F – High Brucite – Low Olivine.

Table 13-16: Benchmark Sample Summary

Test & Solids			Unsheared Sample		Sheared Sample			Delta (S-U)		Max (S-U)	
Test Code	Solids %	Shear Stress Peak, Pa	$\tau_{\gamma B}$ Pa	η_P mPa.s	$\tau_{\gamma B}$ Pa	η_P mPa.s	Observations	Pa	Pa/%	Pa	Pa/%
Sample 18: 10-RN-218AC01; CSD = 54.2% wt., 45 Pa unsheared and 54 Pa sheared yield stress, respectively.											
T1	61.7	153.0	128	45	189	65	Peak, rheopexy, some plug flow	-61	-0.99	189	3.062
T2	58.8	104.0	86	21	114	8		-28	-0.48	114	1.94
T3	54.9	61.0	49	11	58	7	Peak, transient	-9	-0.17	58	1.056
T4	50.1	30.0	26	5	26	3	Dilatant, possibly some settling	0	-0.01	26	0.519
T5	45.0	11.0	7	5	5	7		2	0.047	7	0.158
T6	40.0	7.0	5	6	4	5		1	0.028	5	0.115
Sample 23: 10-RN-216E01; CSD = 65.4% wt., 73 Pa unsheared yield stress, sheared sample torque overload.											
T7A	68.2	185.0	138	213	--	--	Torque Overload - rheopectic	--	--	--	--
T7	66.5	134.0	94	113	--	--		--	--	--	--
T8	63.5	76.0	53	43	93	8	Rheopectic	-41	-0.64	93	1.468
T9	59.9	40.0	26	21	39	8		-13	-0.21	39	0.648
T10	55.7	22.0	14	13	20	6		-6	-0.1	20	0.356
Sample 27: 10-RN-222F01; CSD=49% wt, 66 Pa unsheared, 82 Pa sheared yield stress, respectively.											
T11	54.3	--	168	197	339	35	Rheopectic	-171	-3.15	339	6.245
T12	51.6	--	112	101	169	54	Thixotropic	-56	-1.09	169	3.267
T13	48.2	--	73	65	95	55		-22	-0.46	95	1.965
T14	43.3	--	36	24	24	2		-12	0.282	36	0.827
T15	35.4	--	13	6	5	8		8	0.223	13	0.359

Source: RNC.

Each of these samples underwent shear stress testing at various slurry densities. The pulverized sample was slurried with water to their critical solids density (CSD). CSD was defined as the solids density value above which a small increase of the solids density causes a significant decrease of the flowability. Once the CSD had been determined for each sample, the solids density was stepped down and the rheological behavior was re-measured. This allowed the rheological behavior to be characterized as a function of their solids density.

The benchmark samples showed a variety of rheological behavior, including some extreme rheopexy at high percent solids on the undeslimed material. Overall, the transition towards less extreme rheopexy occurred with the decrease of the CSD, implying that solids content was a key factor influencing flowability.

13.6.1.2 Rheology Variability Samples

The following summarizes the results from the rheology tests (also includes the three chosen to be the benchmark samples), performed on the 102 grindability samples

Twenty-seven samples displayed flow behavior comparable to Benchmark A, featuring extreme rheopexy, rendering their shear yield stress behavior non-measurable due to torque overload at the CSD. These samples displayed unsheared yield stress values ranging from 7Pa to 83 Pa, average 61 Pa.

Seven samples displayed flow behavior comparable to Benchmark B, which is a transitionary response from moderately rheopectic to slightly thixotropic. These samples displayed unsheared yield stress values ranging from 46 Pa through 139 Pa, averaging 83 Pa. These samples displayed sheared yield stress values were 11 Pa through 390 Pa, average 249 Pa.

Sixty-eight samples displayed flow behavior comparable to Benchmark C, featuring a high but measurable rheopexy response; these samples displayed unsheared yield stress values ranging from 30 Pa to 81 Pa, averaging 61 Pa. The corresponding sheared yield stress values ranged from 133 Pa to 507 Pa.

In general, the overall rheological study substantiated that the main common characteristics of most of the Dumont samples, tested in 2011 and 2012, was their predominantly rheopectic tendency, rendering them rheologically-limited to mineral processing unit operations at typical mineral processing densities (greater than 35% solids in flotation) without desliming.

To manage this issue, desliming is used to remove the slimes and fibres that tend to generate the high viscosity slurries and a low percent solids is used to float in both the slimes and rougher circuits. In addition, dispersant (Calgon) is used in both the slimes and rougher flotation. No sample tested to date at the laboratory scale has shown continued extreme viscosity issues after desliming, dispersant addition and dilution as per the design criteria used for the feasibility study.

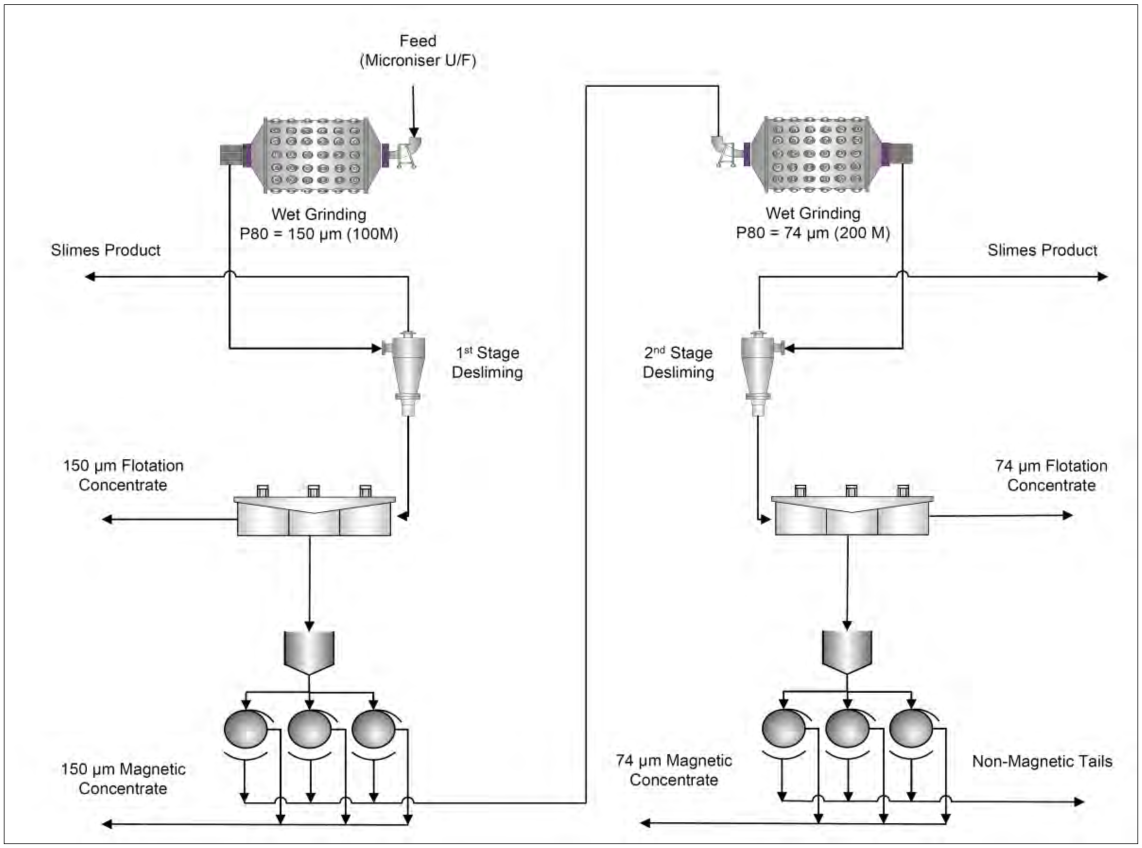
13.6.2 Variability Testing (STP Metallurgical Domain Samples)

The initial standard test procedure (STP) was finalized in May 2009. The composites were prepared from drill core selected from across the deposit. The original STP procedure was applied to the first 83 metallurgical domain samples, and the updated procedure was applied to the additional 22 samples. As per the procedure, a sample of each was sent for quantitative mineralogy and assay. The results are summarized by mineralogy and metallurgical response below.

13.6.2.1 Initial Standard Test Procedure

The original STP flowsheet is shown below in Figure 13.2.

Figure 13.2: Original Standard Test Procedure (STP) Flowsheet



Source: RNC.

The procedures are as follows (applies to all samples other than those listed in Section 13.5.2.2):

- stage crush and stage screen 200 kg of material from core sample size to 100% passing 841 µm (20 mesh) and composite;
- send 1 kg sample of composite to SGS Lakefield for QEMSCAN and electron microprobe (EMP) to confirm nickel deportment mineralogy and liberation;
- air classify 160 kg of crushed and screened material with the objective of removing about 10% weight as fine (light) fraction;
- the coarse (heavy) fraction will be put into bags and then frozen as soon as possible;
- the fluffy portion will be kept frozen;
- one batch of 35 to 40 kg of coarse material from the underflow from the micronizer resulting from air classification (Part 1 work) will be ground in wet media in a ball mill to 80% minus 100 mesh;

- dispersant Calgon at 500 g/t and PAX at 150 g/t will be added into the ball mill prior to wet grinding;
- the sample will be treated with a hydrocyclone to deslime the pulp with approximately 5% weight going to the overflow;
- flotation separation will be conducted on the hydrocyclone U/F;
- magnetic separation will be conducted on the flotation rougher tails;
- the non-magnetic portion will then be wet ground to an 80% minus 200 mesh;
- dispersant Calgon at 500 g/t and PAX at 100 g/t will be added into the ball mill prior to wet grinding;
- second stage of grinding will be followed by a second stage of wet desliming (about 5% weight loss to the overflow);
- a second stage of flotation separation will be conducted on the hydrocyclone U/F;
- the flotation tails will undergo a second stage of magnetic separation;
- weight assessment will be performed on all products.

A complete reagent scheme, conditioning and flotation times are detailed in Table 13-17.

Table 13-17: Standard Conditions for STP Test

Stage	Reagents (g/t)				Time (minutes)		
	PAX	Cytec 65	Calgon	Dep C (2%)	Grind	Cond.	Froth
Grind 1	150		500	500	35		
Deslime							
Rougher 1	150	31.5				5	40
Mag Sep 1							
Grind 2	100	90	500	500	55		
Deslime							
Rougher 2	50	0				1	28
Mag Sep 2							
Total	450	50	1,000	1,000		6	68

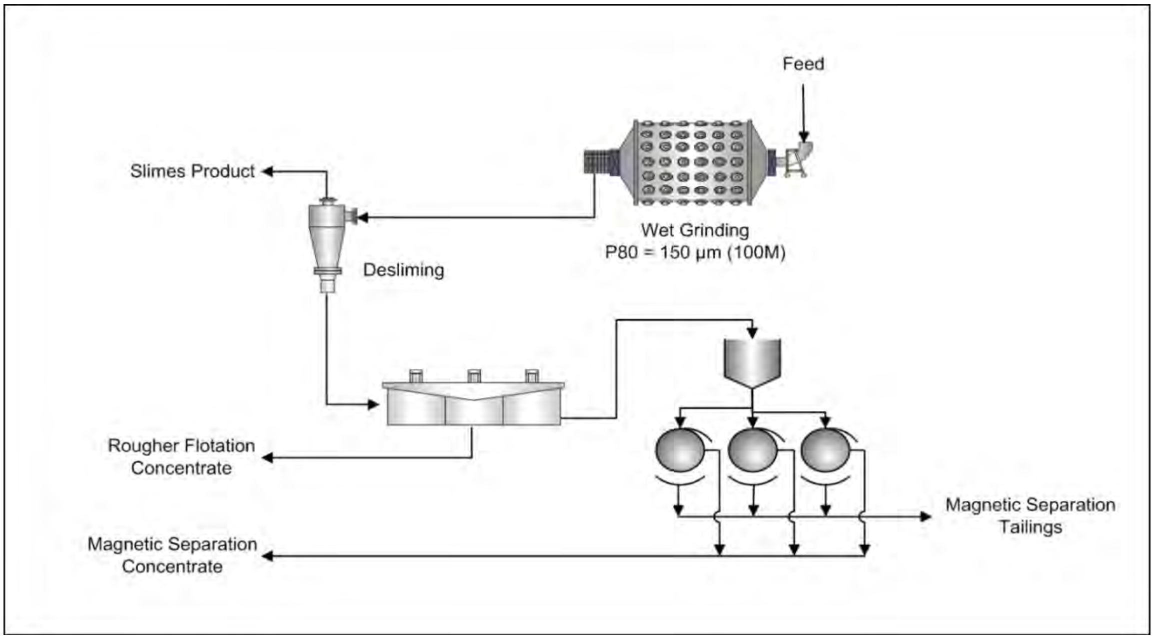
Stage	Flotation Cell	Speed (rpm)
Flotation Cell	Denver D2 60L	1,600

Source: RNC.

13.6.2.2 Updated Standard Test Procedure (applied to samples from holes 108,123, 146, 181, 219, 220,274, 287, 312, 363, 366, 379)

The initial standard test procedure (STP) was modified in 2012 to reflect the updated flowsheet which eliminated the dry defibring (Section 13.2) and a two-stage grind. The elimination of the two-stage grind is discussed in Section 13.7.1. The STP test procedure is shown in the Figure 13.3 and described below. This updated STP procedure was applied to the 22 metallurgical domain samples tested in 2012 and 2013. As per the procedure, a sample of each was sent for quantitative mineralogy and assay. The results are summarized by mineralogy and metallurgical response below.

Figure 13.3: Updated STP Flowsheet



Source: RNC

The procedures are as follows:

- stage crush and stage screen 200 kg of material from core sample size to 100% passing 841 µm (20 mesh) and composite;
- send 1 kg sample of composite to SGS Lakefield for QEMSCAN and electron microprobe (EMP) to confirm nickel deportment mineralogy and liberation;
- one batch of 10kg will be ground in wet media in a ball mill to 80% minus 100 mesh;
- dispersant Calgon at 500 g/t and PAX at 150 g/t will be added into the ball mill prior to wet grinding;
- the sample will be treated with a hydrocyclone to deslime the pulp with approximately 5% weight going to the overflow;
- flotation separation will be conducted on the hydrocyclone underflow;
- magnetic separation will be conducted on the flotation rougher tails; and
- weight assessment will be performed on all products.

A complete reagent scheme, conditioning and flotation times are detailed in Table 13-18.

Table 13-18: Standard Conditions for STP Test

Stage	Reagents (g/t)				Time (minutes)		
	PAX	Cytec 65	Calgon	Dep C (2%)	Grind	Cond.	Froth
Grind 1	150		500	500	35		
Deslime							
Rougher 1	150	31.5				5	60
Mag Sep							
Total	300	31.5	500	500		5	60

Stage	Flotation Cell	Speed (rpm)
Flotation Cell	Denver D2 60L	1,600

Source: RNC.

A representative sample from each of the 102 metallurgical domain samples was sent to SGS Mineral Services (Lakefield) for QEMSCAN quantitative mineralogical analysis.

13.6.3 Variability Testing Results – Rougher Nickel Grade & Recovery

Each sample was processed through either the initial STP or updated STP as described above to assess the variability of the metallurgical response throughout the mineralization. A summary of the results for each sample (listed by drill hole number) is shown below. The results from these samples formed the basis of the rougher recovery equations for the feasibility study. The rougher recovery listed in the following tables is based on the rougher recovery achieved in the STP including the predicted recovery from the fluff portion that was not tested as part of the STP. Testwork has shown that recovery from the fluff portion is similar to the STP rougher recovery (Section 13.2.3.2). This fluff recovery portion has been added to the base rougher recovery.

The average results for each metallurgical domain are shown below in Table 13-19. Additional details of the STP results completed by CTMP/Mineral Solutions and summarized herein are available on RNC's website.

Table 13-19: STP Variability Results Summary

	# of Samples	% Ni	% S	Aw	Hz	Pn	Rougher Ni Recovery
Hz Dom	25	0.31	0.10	0.07	0.27	0.01	56.1
Mixed Sulphide	19	0.30	0.08	0.12	0.23	0.09	55.9
Pn Dom	36	0.34	0.13	0.16	0.11	0.40	58.0
High Fe Serp	25	0.37	0.18	0.13	0.06	0.57	48.4
Total	102	0.33	0.13	0.12	0.16	0.27	54.5

Some of the samples from the STP produced results with low concentrate grades with little nickel upgrading to concentrate. In ultramafic nickel deposits such as Dumont there can be significant levels of nickel contained within the silicates. This nickel is unrecoverable with flotation based techniques.

This is especially true for the alloy (low sulphur) mineralization assemblages as defined in Section 7.3.1.1. Most of these low sulphur samples generated very low concentrate grades, high weight recoveries to concentrate and high tails assays. Table 13-20 compares the average result from the three mineralization assemblages from the 102 STP samples. As the sample moved from sulphide to alloy mineralization the rougher concentrate grade decreases, the tails grade increases and the recovery decreases. It is expected that lower cleaner recoveries will be seen with lower sulphur grades in feed, and that sulphur content is directly related to cleaning recovery.

Table 13-20: STP Summary by Mineralization Type

Sample Name	% Ni Feed	% S Feed	Weight Recovery to Rougher Conc (%)	Rougher Conc Grade (% Ni)	Rougher Tails Grade (%Ni)	Rougher Recovery (%)
Sulphide Average	0.39	0.20	22.6	1.45	0.20	61
Mixed Average	0.29	0.08	22.5	0.95	0.22	50
Alloy Average	0.26	0.03	28.1	0.53	0.24	45

Source: RNC.

The STP test uses staged grinding, long flotation times, low density flotation conditions and very high reagent consumption. These conditions would be extremely expensive to replicate in a full scale plant and optimization work was performed to demonstrate that similar grades and recoveries could be achieved with lower flotation times, higher densities and reduced reagent consumption. The results from this optimization work are presented in Section 13.7 and formed the basis for the FS plant design and operating cost.

13.7 Metallurgical Optimization Results

13.7.1 Grinding Circuit

13.7.1.1 Single-Stage Grind

In the initial STP tests a two-stage grind followed by a deslime, float and magnetic separation was performed. This was done for the first 83 samples. The second grinding stage was eliminated after a review of various samples under a single-stage grind to 150 µm. In Table 13-21, the results of comparative tests are shown. The STP test is shown for each samples compared with various other tests under modified flotation conditions (reagent dosage, desliming operation and % solids of the rougher float).

Table 13-21: Comparative Optimization Tests

Sample	Conditions*	Float Time (mins)	Conc Grade (%Ni)	Ni Recovery (%)
176E	STP	68	1.21	26.7
176E	Single Grind	36.5	0.72	37.4
176E	Single Grind	23.5	1.1	27.6
176G	STP	68	2.44	30.9
	Single Grind	25	1.23	33.6
	Single Grind	34.5	1.50	42.5
	Single Grind	48.5	2.50	32.6
213H	STP	68	0.58	29.7
	Single Grind	56	1.02	32.3
218BDF	STP	68	3.80	57.3
	Single Grind	30	4.62	57.5
	Single Grind	30	5.62	58.8
	Single Grind	30	2.93	66.3
	Single Grind	30	3.92	60.7

* STP = Double Grind (150 µm grind, deslime, float, magnetic separation then 100 µm grind, deslime, float) Single Grind = 150 µm grind, deslime, float, magnetic separation.

Since the results showed for each sample tested that the recovery was equal to or better than the STP recovery, the decision was made to continue to include the STP recovery with the two-stage grind in the rougher recovery equations, but discontinue the two-stage grind for the plant flowsheet. Subsequently the STP was also modified to reflect this decision.

13.7.1.2 Grind Size Selection

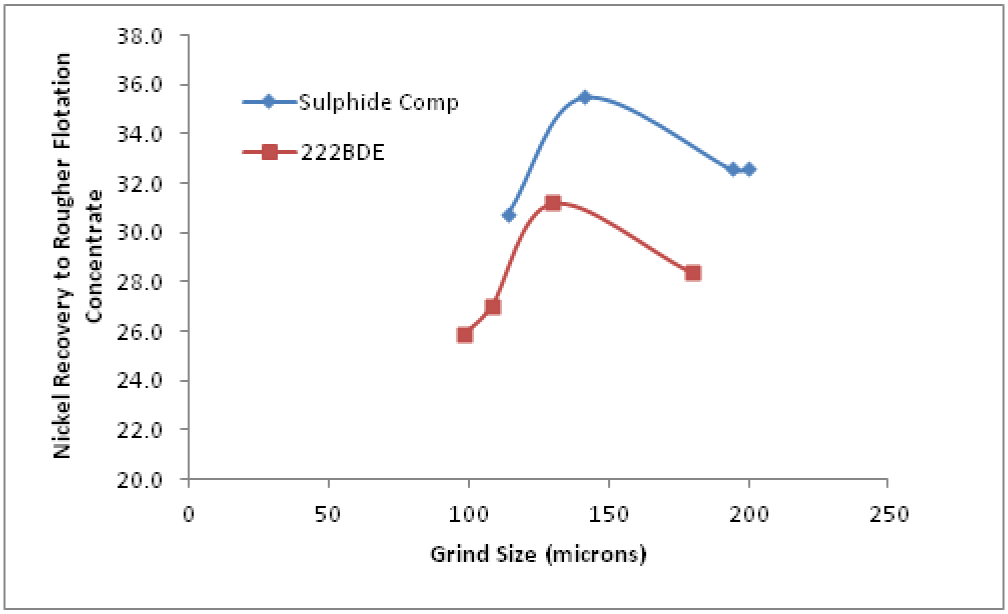
The STP grind size was chosen as 150 µm (100 mesh). Testwork was completed to confirm this is the optimum size to maximize rougher nickel recovery. Tests were performed on two samples, the sulphide composite and 222BDE, a low recovery Hz sample.

The results are shown below in Figure 13.4. This figure plots the flotation rougher recovery vs. the P_{80} of the sample. Both samples show a peak in flotation recovery between 130-160 µm.

Total rougher recovery is comprised of both the flotation and magnetic recovery. Figure 13.5 shows the total rougher recovery vs. the P80 of the sample. In general, the total nickel recovery is higher as the grind size gets coarser (within the size range tested). This may result from increased kinetics due to reduction of slimes generated during grinding. However, as the grind size increases, the rougher concentrate grade generally shows a decreasing trend, which may indicate a reduction of liberation at the coarser grind sizes (Figure 13.6).

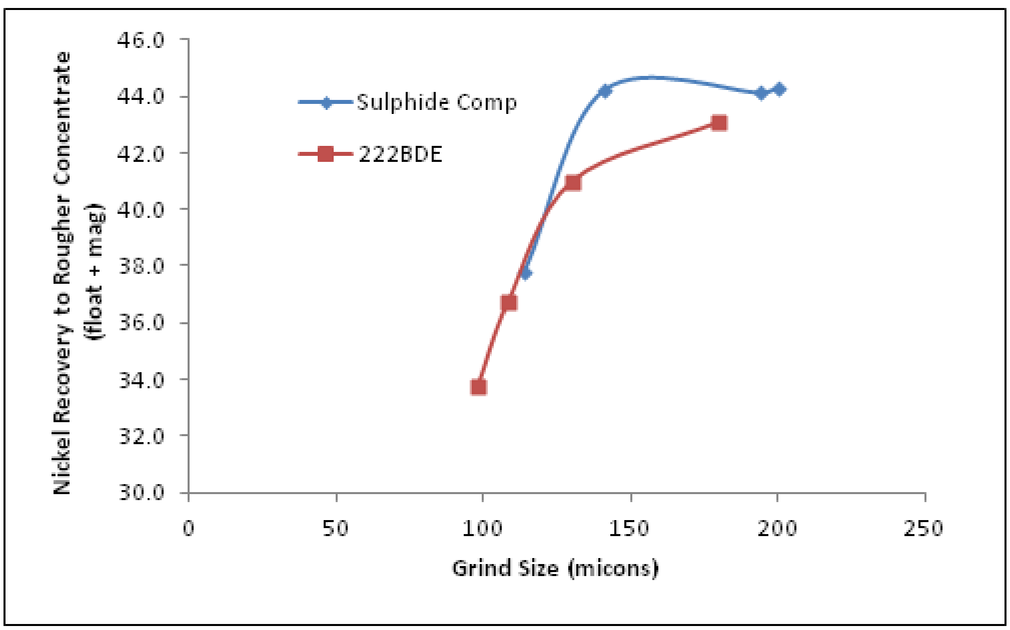
From this work, the grind size of 150 µm was chosen for the feasibility design basis.

Figure 13.4: Flotation Recovery as a Function of Grind Size



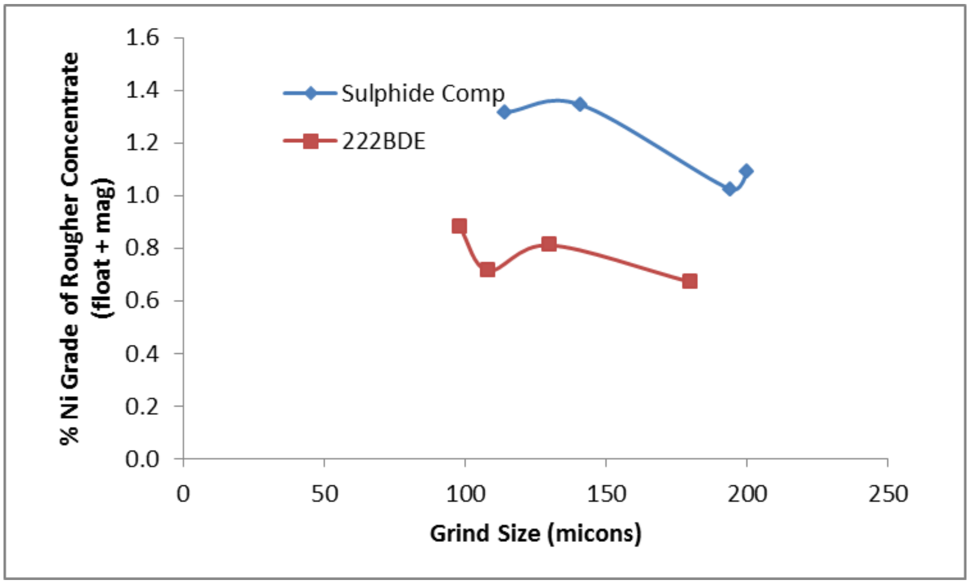
Source: RNC.

Figure 13.5: Rougher Recovery as a Function of Grind Size



Source: RNC.

Figure 13.6: Rougher Concentrate Grade as a Function of Grind Size



Source: RNC.

13.7.2 Desliming & Rougher Flotation Optimization (including residence time)

Desliming is a critical process step to maximize the rougher flotation performance of the Dumont mineralization. Without desliming the rougher is very viscous, nickel flotation kinetics are slow and the rougher concentrate grades are very low.

The STP had an average of 7% mass report to the slimes fraction. This material was not floated as part of the STP.

Benchmarks from other ultramafic desliming operations indicate a greater percentage of material will report to the slimes fraction from the closed grinding circuit. Benchmarking has indicated that approximately 10-20% of the nickel will report to the slimes fraction in a full-scale plant. Twenty percent weight recovery to the overflow (O/F) has formed the basis for the feasibility design.

To understand the differences in reagent consumption and flotation performance, tests on several samples were conducted at various weight recovery to the slimes product.

13.7.2.1 Outcrop Sample

To understand the reagent consumption and performance difference between the 10% and 20% mass recovery to the O/F were performed on the Outcrop sample. As part of this test program the underflow (U/.F) was also tested. Both the overflow and underflow were floated in each test with different reagent dosages. Each flotation test was a kinetic test floated for 30 min, with incremental concentrates being removed at 1, 4, 10, 20, and 30 minutes for both the U/F and the O/F. This was to determine whether the reagents could be modified to increase the kinetics of the float as well as to determine the optimum flotation time to match the STP flotation recovery. Table 13-22 and 13-23 is a summary of the conditions and results for the O/F and U/F testing respectively at 10% Wt Recovery to O/F.

Table 13-22: Overflow Reagent & Kinetic Testing (10% Wt to O/F)

Test Number	PAX (g/t)	Calgon (g/t)	% Ni Grade 10 min	N*i* Recy[1] 10 min	% Ni Grade 20 min	N*i* Recy[1] 20 min
O/F Kin-T1	50	100	0.71	5.1	0.67	6.2
O/F Kin-T2	150	100	0.60	3.9	0.57	5.4
O/F Kin-T3	300	100	0.63	4.4	0.64	6.3
O/F Kin-T4	50	250	0.43	3.8	0.42	4.6
O/F Kin-T5	150	250	0.68	4.9	0.68	6.9
O/F Kin-T6	300	250	0.56	4.7	0.58	5.8
O/F Kin-T7	50	400	0.71	6.0	0.68	6.7
O/F Kin-T8	150	400	0.78	4.6	0.71	5.4
O/F Kin-T9	300	400	0.72	4.7	0.68	5.5

[1]Nickel recovery is stated as % of total feed.

Table 13-23: Overflow Reagent & Kinetic Testing (20% Wt to O/F)

Test Number	PAX (g/t)	Calgon (g/t)	% Ni Grade 10 min	N*i* Recy[1] 10 min	% Ni Grade 20 min	N*i* Recy[1] 20 min
O/F Kin-T10	50	100	0.88	14.3	0.8	15.3
O/F Kin-T11	150	100	0.78	14.8	0.67	16.7
O/F Kin-T12	300	100	0.78	14.7	0.68	16.7
O/F Kin-T13	50	250	0.76	15.5	0.68	16.9
O/F Kin-T14	150	250	0.68	15.9	0.64	17.3
O/F Kin-T15	300	250	0.66	18.3	0.65	18.5
O/F Kin-T16	50	400	0.90	13.6	0.72	15.9
O/F Kin-T17	150	400	0.78	14.7	0.71	16.0
O/F Kin-T18	300	400	0.74	14.5	0.69	16.1

[1] Nickel recovery is stated as % of total feed.

Table 13-24 and 13-25 is a summary of the conditions and results for the O/F and U/F testing respectively at 20% Wt Recovery to O/F. All reagents are shown as g/t O/F or U/F feed to the stage.

In general the O/F tests produced relatively low-grade concentrates with little upgrading and high mass pull to concentrate, irrespective of either PAX or Calgon addition. The recovery and grade performance in the slimes flotation were better in the 20% weight pull to the O/F due to the presence of more recoverable nickel compared with the 10% stream. The overflow tests were floated at 10% solids.

Reviewing the results, although there was variation between tests, there was little to no trends with regard to either xanthate or calgon addition. Increased xanthate did appear to lower the concentrate grade slightly, resulting from a less selective float. However increased calgon did not appear to increase the recovery from the slimes for this sample. Based on the results from this testwork, a reagent dosage of 50 g/t PAX and 100 g/t Calgon were used for the slimes flotation. Those reagent dosages are references in grams per tonne of feed to the slimes circuit.

Table 13-24: Underflow Reagent & Kinetic Testing (10% Wt to O/F)

Test Number	PAX (g/t)	Calgon (g/t)	% Ni Grade 20 min	N*i* Recy[1] 20 min	% Ni Grade 30 min	N*i* Recy[1] 30 min
U/F Kin-T10	50	100	2.37	44.6	1.83	49.0
U/F Kin-T11	150	100	1.63	49.1	1.45	53.7
U/F Kin-T12	300	100	1.33	47.6	1.26	52.3
U/F Kin-T1	50	275	2.47	49.0	2.07	51.7
U/F Kin-T2	150	275	2.34	46.5	1.94	51.7
U/F Kin-T3	300	275	1.62	49.1	1.43	53.6
U/F Kin-T13	50	400	1.83	49.9	1.55	54.1
U/F Kin-T14	150	400	2.53	43.7	1.89	48.5
U/F Kin-T15	300	400	1.741	48.5	1.48	53.4

[1] Nickel recovery is stated as % of total feed

Table 13-25: Underflow Reagent & Kinetic Testing (20% Wt to O/F)

Test Number	PAX (g/t)	Calgon (g/t)	% Ni Grade 20 min	N*i* Recy[1] 20 min	% Ni Grade 30 min	N*i* Recy[1] 30 min
U/F Kin-T16	50	100	3.20	41.4	2.07	51.7
U/F Kin-T17	150	100	3.64	41.4	3.00	43.7
U/F Kin-T18	300	100	2.06	43.0	1.91	45.5
U/F Kin-T19	50	250	4.02	41.7	3.08	43.8
U/F Kin-T20	150	250	2.58	42.0	2.22	44.4
U/F Kin-T21	300	250	1.90	40.6	1.63	44.6
U/F Kin-T22	50	400	4.04	43.7	3.44	45.3
U/F Kin-T23	150	400	3.20	44.6	2.42	47.2
U/F Kin-T24	300	400	2.49	39.3	1.98	42.5

[1] Nickel recovery is stated as % of total feed

Reviewing the U/F results in both sets of tests, increased xanthate did lower the concentrate grade, resulting from a less selective float. Increased calgon did appear to increase the recovery from the underflow for this sample at lower xanthate additions, but the results are not clear with some lower calgon additions giving the same recovery as the higher calgon additions. Based on the results from this testwork, a reagent dosage of 50 g/t PAX and 200 g/t Calgon were used for the rougher (U/F) flotation design basis. Dosages are given in g/t of U/F.

To establish the optimum residence time split between the O/F and U/F, the kinetics of each were reviewed. Additional flotation time in the slimes float had much less of an impact to the overall grade recovery curve, compared with additional flotation time in the U/F float. Less than 1.5% Ni recovery was added by extending the flotation time in the O/F from 10 to 20 minutes. This is compared to the U/F, where extending the flotation time from 20 to 30 min increased the recovery by over 3%. All numbers quoted above are from the 20% mass split to the O/F tests.

The decision was made for the FS design to use a 10 min lab residence time for the O/F and 30 minutes' lab residence for the U/F.

Comparing this suite of testwork to the STP (shown in Table 13-26) that was performed for 60 minutes under very different flotation conditions (1000 g/t calgon and 1200g/t CMC) gave very similar results to the average conditions seen in the kinetic tests, illustrating that the large reagent dosage used in the STP and lengthy residence times could be reduced.

Table 13-26: Summary of O/F & U/F Flotation kinetic tests

Conditions	Rghr Float Time	OF Float Time	Conc Grade[1]	% Ni Recovery[1]
10% Wt O/F	30	10	1.1	68.9
20% Wt O/F	30	10	1.2	70.7
STP (updated)	60	0	1.3	68.6[2]
STP (original)	68	0	1.3	66.3[2]

1. Includes both flotation and magnetic recoveries. 2. Includes estimated recovery from slimes portion for comparison purposes only.

13.7.2.2 Domain Composite Samples – Residence Time

Over flow and underflow samples of the seven domain composites were tested as part of feasibility study data collection phase. Three different levels of weight recovery to O/F were tested for each composite approximately 10%, 15% and 20%.

This work was performed to confirm the ability to reduce the laboratory residence time in the rougher from the 60 min STP to 30 min, with reduced reagents.

Table 13-27 shows the kinetic results with the shorter flotation time and reduced reagent suite compared with the STP for each sample.

Table 13-27: Summary of Kinetic Results

Composite	Test	Rghr Concentrate (%Ni)	% Ni in Rougher Tails
Comp 1	Kinetic	0.72	0.23
	STP	0.55	0.24
Comp 2	Kinetic	2.60	0.21
	STP	1.23	0.22
Comp 3	Kinetic	0.78	0.21
	STP	0.50	0.21
Comp 4	Kinetic	1.07	0.16
	STP	0.86	0.18
Comp 5	Kinetic	0.54	0.22
	STP	0.47	0.20
Comp 6	Kinetic	1.01	0.20
	STP	0.70	0.21
Comp 7	Kinetic	0.61	0.19
	STP	0.54	0.19

Other than Composite 5, all the rougher tails had equivalent or lower assays than the STP test, at a 20-minute flotation time. The kinetic tests also in general had a higher concentrate grade. Based on these tests and the Outcrop sample discussed in the previous section a laboratory flotation time of 30 minutes for the rougher flotation circuit was used for the feasibility study.

13.7.3 Reagents

The reagents used in the STP tests are shown below (Table 13-28). This reagent scheme was very expensive; specifically Calgon (sodium hexametaphosphate) and carboxy-methyl cellulose (CMC) dosages are very high. Large dosages of these dispersants are known to slow flotation kinetics, which may be potentially combated by increased PAX dosage. This would explain the extremely slow kinetics seen in the STP tests. Further optimization testing was performed to identify an alternate reagent scheme could result in the same performance.

Table 13-28: Reagent Consumption from STP Testwork

		MIBC (g/t)	Cytec 65 (g/t)	KAX (g/t)	Calgon (g/t)	CMC (g/t)	H_2SO_4 (g/t)
Initial STP		**0**	**86**	**250**	**1000**	**1200**	**0**
Updated STP		**0**	**90**	**225**	**500**	**500**	**0**

Source: RNC.

Tests performed for flowsheet optimization and the locked cycle tests have used much lower amounts of reagents with higher flotation densities compared to the STP.

13.7.3.1 Rougher & O/F Reagents

Optimization of the rougher reagents was evaluated using the Outcrop sample and the results were presented in Section 13.7.2.1. The results from this testwork indicated increased PAX caused a less selective flotation without a final increase in nickel recovery. Increased calgon did show some increase in recovery at lower PAX dosages. From this testwork the following reagents were selected as the basis for the feasibility study.

Table 13-29: Reagent Consumption for Rougher & O/F

		MIBC (g/t)	Cytec 65 (g/t)	KAX (g/t)	Calgon (g/t)	CMC (g/t)	H_2SO_4 (g/t)
Rougher		**32**	**0**	**42**	**196**	**0**	**0**
O/F		**50**	**0**	**10**	**20**	**0**	**0**

Source: RNC.

Cytec 65 is more expensive than MIBC and during the locked cycle testing led to a persistent froth in the cleaner circuit that was hard to control. Therefore a decision was made to reduce the use of Cytec 65 significantly and replace it with MIBC. The tests performed to optimize the rougher and O/F reagent scheme only used MIBC.

13.7.3.2 Cleaner / Scavenger & Aw Circuit Reagents

The cleaner reagent consumption is taken from the average of 21 locked cycle tests. This includes both the sulphide cleaners and the Aw rougher float and associated cleaners.

Table 13-30: Reagent Consumption for Cleaner / Scavenger & Aw Circuit

	MIBC (g/t)	Cytec 65 (g/t)	KAX (g/t)	Calgon (g/t)	CMC (g/t)	H_2SO_4 (g/t)
Remainder of Circuit	**7**	**2**	**28**	**38**	**6**	**3888**

Source: RNC.

13.7.3.3 Xanthate

The xanthate used in the majority of the tests was KAX51, potassium is-amyl xanthate, considered one of the strongest and least selective collectors. Tests were conducted to determine whether a lower strength collector could provide higher-grade rougher concentrates without a recovery loss. Tests were performed on the Outcrop sample.

A test using KAX20, potassium ethyl xanthate, which is a weaker collector, was done. The results are shown below on both the U/F (rougher) and O/F (slimes). These tests were performed with 20% of the material reporting to the O/F.

Table 13-31: Effect of Xanthate Strength on Rougher Flotation

Conditions	PAX	% Ni Grade	% Ni Recovery
High Xanthate Dosage	KAX51	2.0	55.5
	KAX20	2.7	53.5
Low Xanthate Dosage	KAX51	3.4	57.8
	KAX20	2.8	55.8

No improvement on selectivity or recovery was seen on the rougher (U/F) flotation and results indicate a potentially recovery loss with the weaker collector.

Table 13-32: Effect of Xanthate Strength on Slimes Flotation

Conditions	PAX	% Ni Grade	% Ni Recovery
High Xanthate Dosage	KAX51	0.76	70.4
	KAX20	0.97	59.0
Low Xanthate Dosage	KAX51	0.66	78.8
	KAX20	1.00	53.9

The slimes flotation did show an improvement in selectivity with the weaker collector. However, the improvement was not considered worth the addition and complication of adding a second collector for just this stream.

KAX51 was chosen for the design basis of the feasibility study.

13.7.3.4 Reagent Summary

The complete reagent summary used as the basis of the feasibility study is shown in Table 13-33.

Table 13-33: Reagent Consumption for Overall Circuit

	MIBC (g/t)	Cytec 65 (g/t)	KAX (g/t)	Calgon (g/t)	CMC (g/t)	H_2SO_4 (g/t)
Total	**89**	**2**	**80**	**254**	**6**	**3888**

Source: RNC.

13.7.4 Regrind Size Selection

The rougher magnetic concentrate and first sulphide cleaner tails report to a regrind mill to liberate any locked particles prior to sulphide scavenging and Aw flotation.

The average grind size in the locked cycle tests was 56 µm with a range of 38 to 73 µm.

The design basis for the feasibility study was 46 µm to account for the requirement for a finer size with some samples.

13.7.5 Tailings Dewatering

A sample of each of the seven metallurgical domain composites was sent to Outotec for thickener testing. This represents the range of variability throughout the Dumont deposit. The tests were conducted in a bench scale 100 mm diameter thickener. Flocculent screening was performed and Magnafloc 333 was chosen for the tests.

Table 13-34 is a summary of the results extracted from the Outotec report (Barnes, A., 2012).

Table 13-34: Thickener Testing Results

Sample	Solids Loading Rate (t/m^{2}h)	Rise Rate (m/h)	Floc Dosage	Achievable Underflow Density (%w/w solids)	Achievable Overflow Clarity (ppm TSS)	Max Unsheared U/F Yield Stress (Pa)
Comp 1	0.3-0.8	4-11	14-115	26-50	84-5324	91
Comp 2	0.3-0.6	5-10	28-34	36-50	83-558	118
Comp 3	0.5-0.8	5-9	10-20	20-52	95-234	429
Comp 4	0.3-0.6	6-8	6-24	29-49	27-158	71
Comp 5	0.3-0.6	4-8	6-26	42-49	58-283	61
Comp 6	0.3-0.6	4-8	10-31	31-48	45-131	51
Comp 7	0.3-0.6	4-8	10-30	22-48	72-115	63

Source: Barnes, A. 2012

The ultramafic Dumont mineralization does not settle quickly and the serpentine and brucite content prevent traditional underflow densities of 60-65% from being achieved.

From this testwork, 40% w/w solids was used in the feasibility study as the density achieved in the underflow from a high rate tailings thickener.

13.8 Recovery Equations

In the 2010 PEA, the rougher recovery equations were defined from 32 samples from five drill holes that were available at the time of the evaluation. The samples were grouped by mineralization type (sulphide, alloy and mixed) and by structural domain. For the 2011 PFS, an additional 38 samples, for a total of 70, were added to the STP suite to update the recovery equations. The 2012 revised PFS had an additional 13 samples processed, for a total of 83 samples. To support the feasibility study an additional 22 samples were added for a total 105 STP tests.

13.8.1 Rougher Recovery Equations

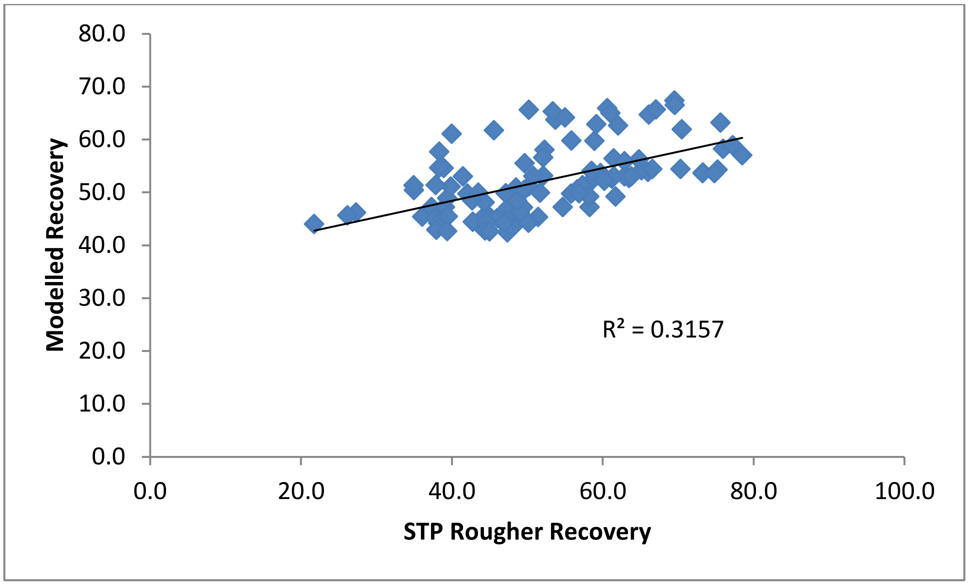
The 105 STP tests formed the basis for the rougher recovery equations. The complete assay and mineralogical data (QEMSCAN) were available for each of the 105 samples.

This information was entered into Minitab statistical software program to perform multiple linear regression analysis on the results. Rougher recovery was used as the response. The predictor variables were limited to the assay data set. It was decided that the mineralogy would not be used in the recovery equations for the FS due to the higher confidence in the deposit assay model compared with the deposit mineralogical model.

At first the regression was applied to the entire STP dataset without domaining, however the R^2 was low (shown in Figure 13.7). The resulting regression equation is shown below, as is the plot of actual vs. modelled recovery using this equation:

*Rougher Ni Recovery = 37.68+25*S/Ni + 0.0018*Ni ppm*

Figure 13.7: Regression Results without Domaining STP Samples

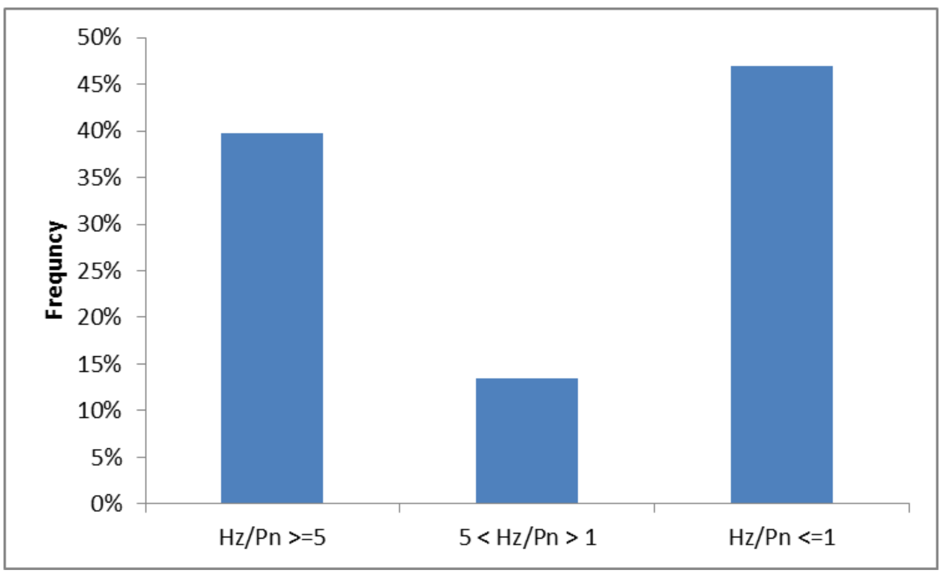


Source: RNC.

To improve the regression, domaining the deposit was performed based on mineralogy.

A review of the larger EXPLOMIN™ data set (1,420 QEMSCAN mineralogy samples) showed that there were distinct populations of samples, either Pn-dominant or Hz-dominant with a small amount that fell in a mixed category between the two extremes (Figure 13.8).

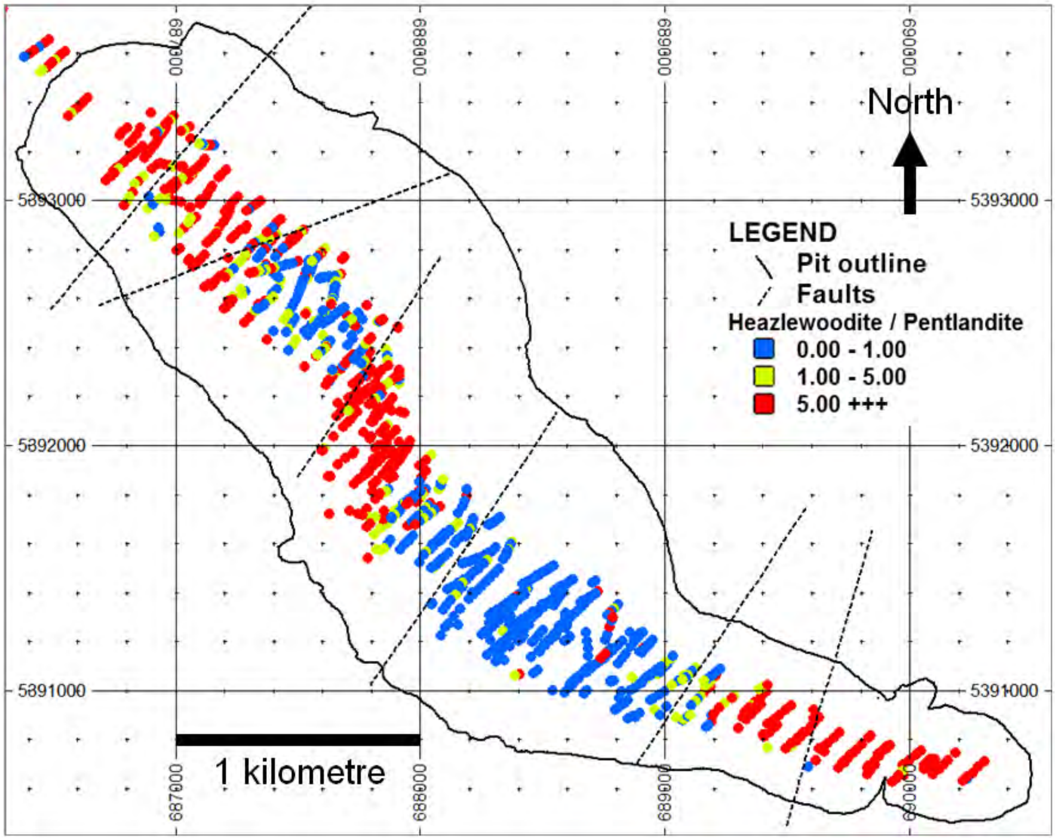
Figure 13.8: Distribution of Hz/Pn Ratio in EXPLOMIN™ Results



Source: RNC

The Pn and Hz dominant samples are in distinct spatial locations relative to each other. In Figure 13.9, the distribution of the sulphide mineralization, with red being Hz Dominant and blue being Pn Dominant, is shown overlaid by the feasibility pit shell. The mixed sulphide can be seen in yellow and is generally a transition zone between the Hz dominant areas and Pn dominant areas.

Figure 13.9: Sulphide Distribution



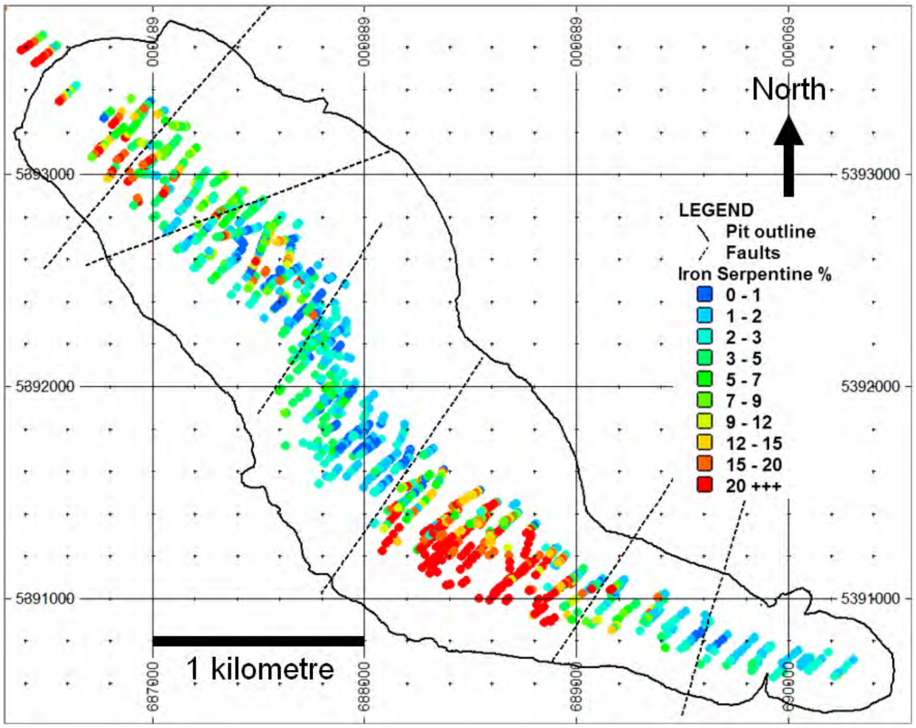
Source: RNC.

A review of the locations of the Pn dominant samples showed two district populations: one in the southern end of the mineralization (structural domain 2, 3 and 4) and the other in the northern end (structural domain 5 and 6).

There were other differences within these two pentlandite populations including the degree of serpentinization (as evidenced by increased amounts of iron serpentine) and the nickel tenor of the pentlandite. In Figure 13.10, the higher iron serpentine samples from the EXPLOMIN™ database are shown in orange and red. The majority of the high iron serpentine occurrences are located in structural domain 3, with an additional smaller population in structural domain 5 and 6 to the northwest.

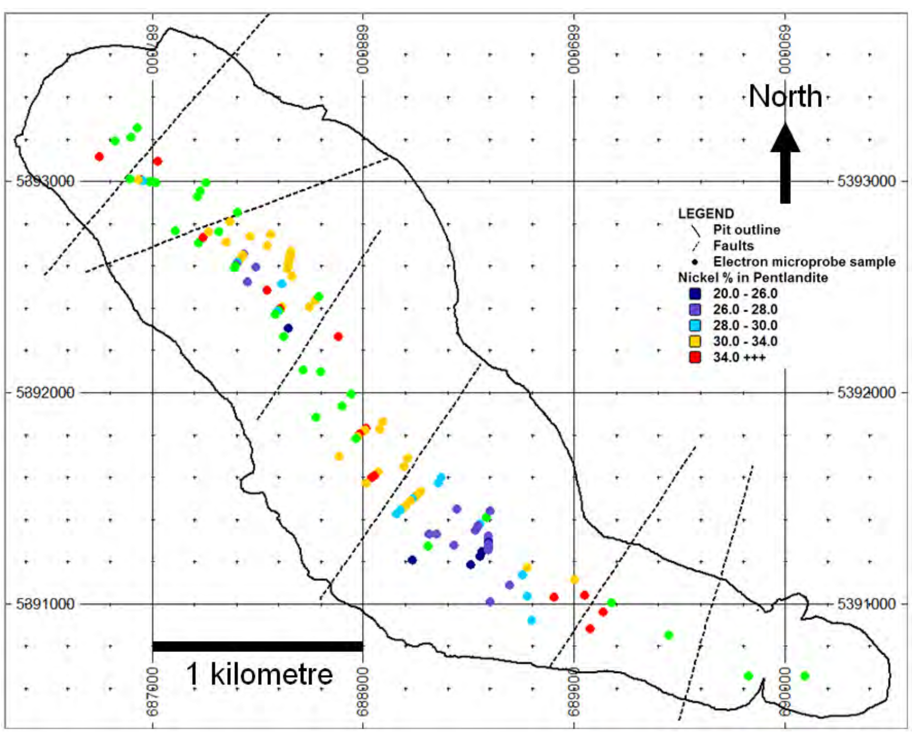
The lower tenor pentlandite (shown in dark blue and black in Figure 13.11) is highly correlated with the areas of Fe Serpentine in both structural Domain 3 and 5. The pentlandite outside these areas has an average Ni tenor of 34%.

Figure 13.10: Distribution of Fe Serpentine within the FS Pit Shell



Source: RNC.

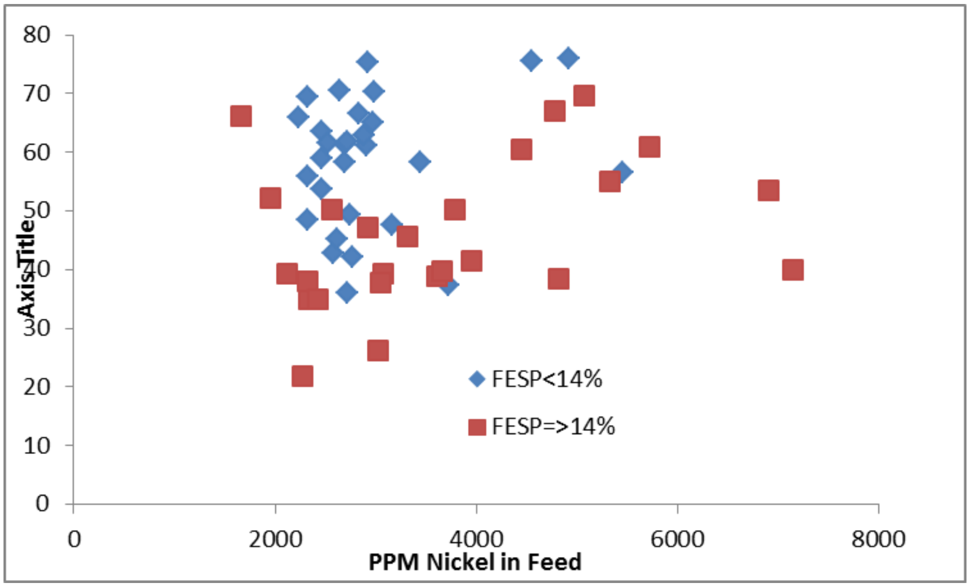
Figure 13.11: Nickel Tenor in Pentlandite



Source: RNC.

The mineralogical model abundances and electron microprobe results show that they have different mineralogical characteristics and this may lead to different metallurgical performance. This was confirmed by a review of the STP test results as shown below in Figure 13.12. Figure 13.12 shows a trend of lower recovery for the same head grade in the higher iron serpentine samples.

Figure 13.12: STP Recovery for High & Low FESP Samples



Source: RNC.

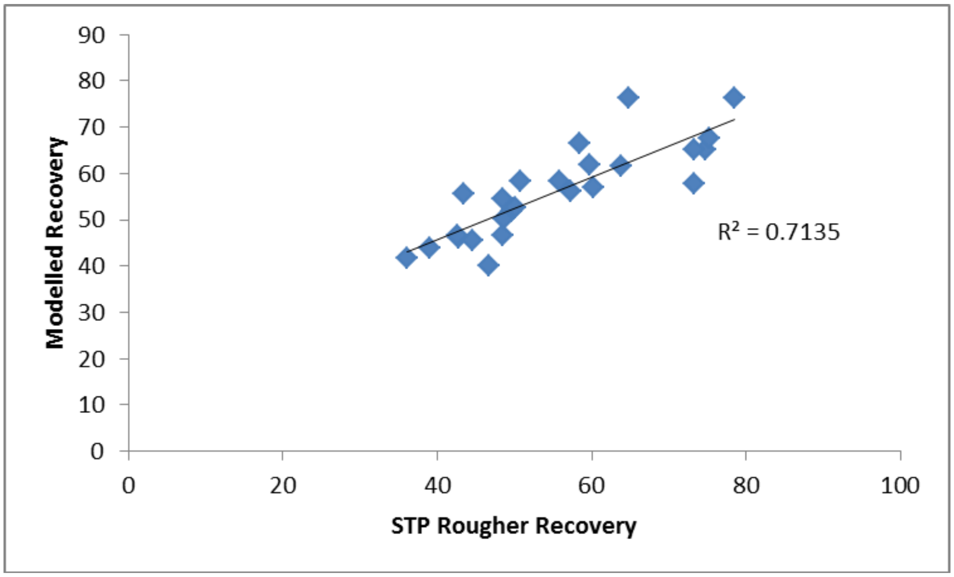
The samples containing over 14% iron serpentine by weight were split out and the regression was run separately.

The differences in mineralogy discussed above support the generation four metallurgical domains for the feasibility recovery equations: (1) Hz Dominant, (2) Mixed Sulphide, (3) Pn Dominant, and (4) High Iron Serpentine.

13.8.1.1 Results for Hz Dominant: FESP < 14, Hz/Pn ≥ 5

Rougher Ni Recovery = 18.11 + 0.0211*S + 0.00039 * Fe

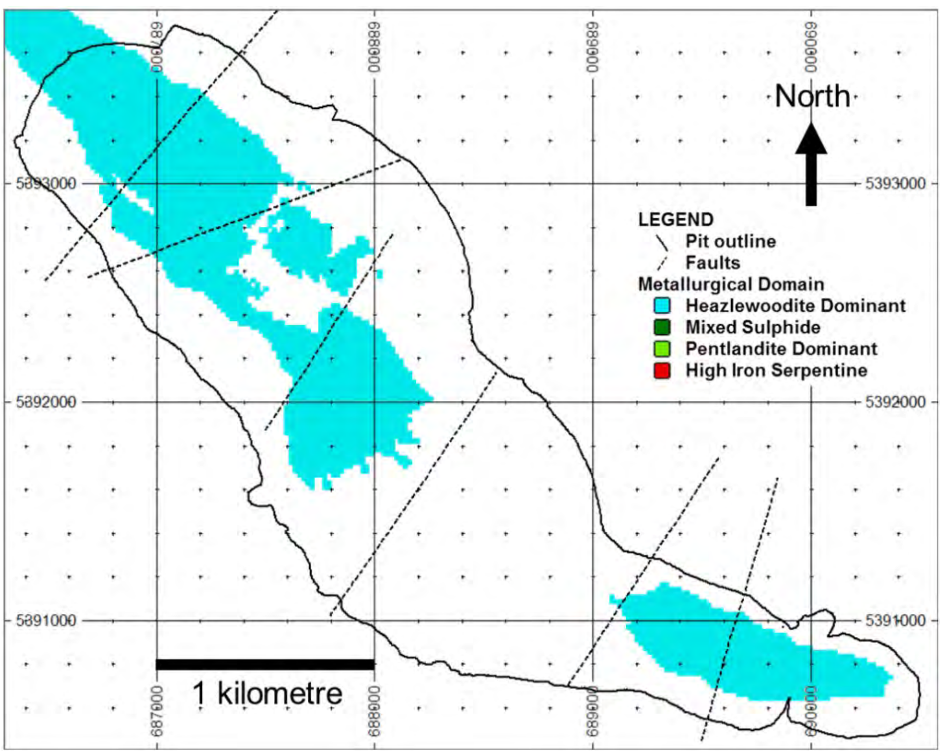
Figure 13.13: Recovery Regression Model for Hz Dominant Samples



Source: RNC.

The distribution of the Hz Dominant zones is shown in Figure 13.14.

Figure 13.14: Distribution of Hz Rich Metallurgical Domain

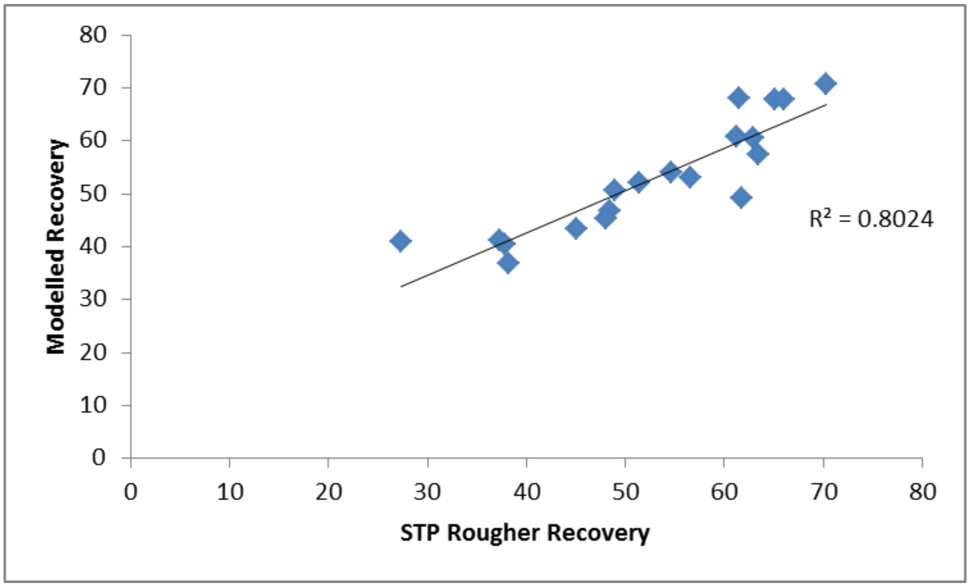


Source: RNC.

13.8.1.2 Results for Mixed Sulphide: FESP < 14, 1 < Hz/Pn < 5

Rougher Ni Recovery = 9.73 + 0.222*S + 0.0111*Ca

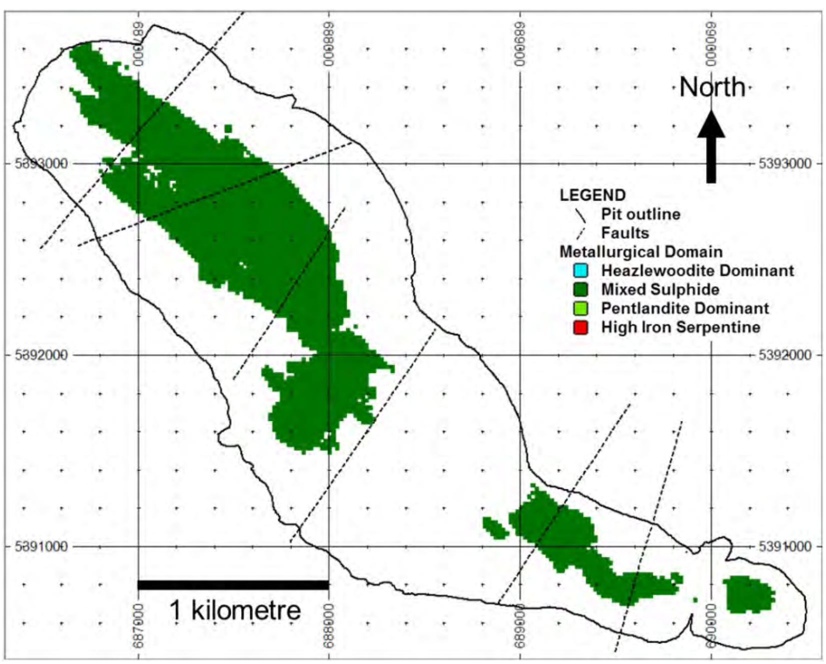
Figure 13.15: Recovery Regression Model for Mixed Sulphide Samples



Source: RNC.

The distribution of the Mixed Sulphide domain can be seen in Figure 13.16.

Figure 13.16: Distribution of Mixed Sulphide Metallurgical Domain

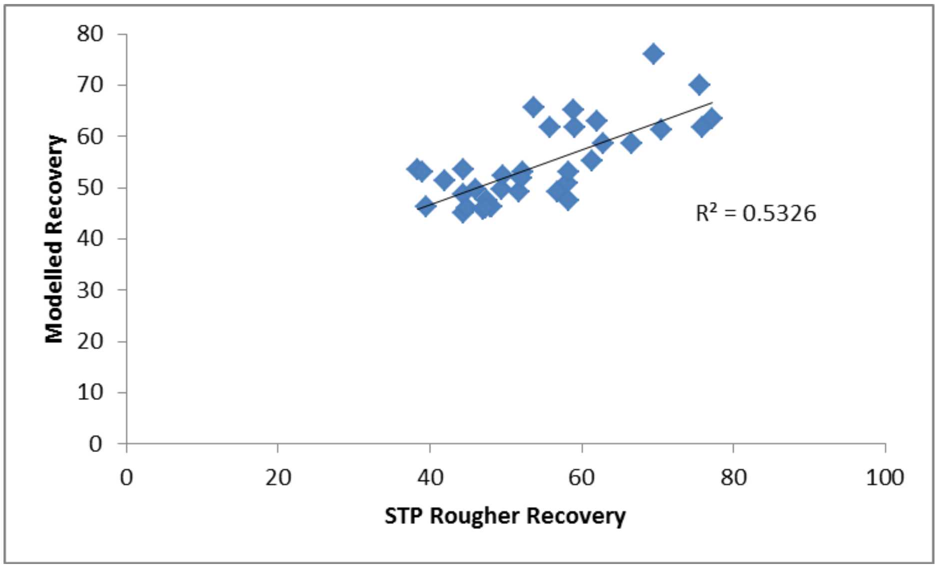


Source: RNC.

13.8.1.3 Results for Pn Dominant: FESP<14%, Pn/Hz <= 1

Rougher Ni Recovery = 43.6 + 00055* S + 0.0111* Ca

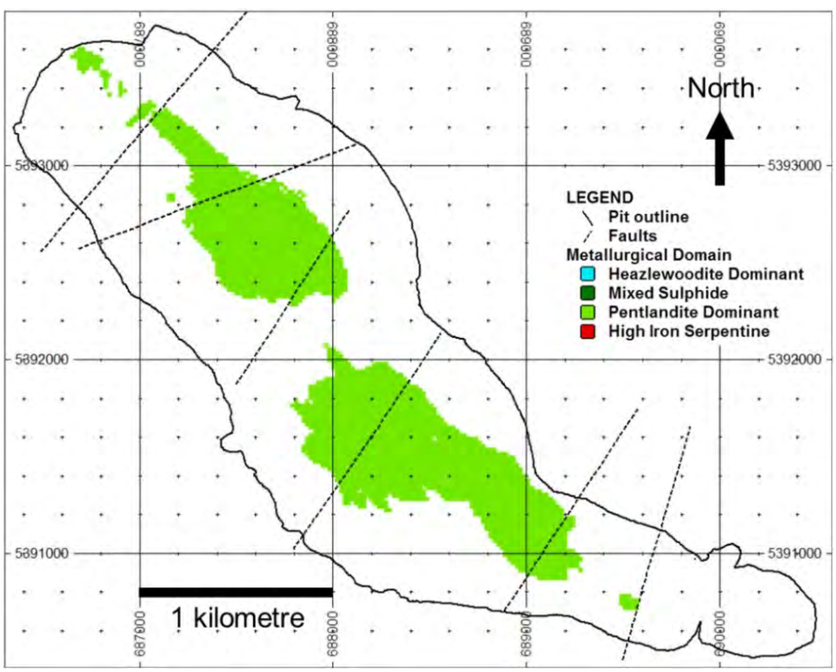
Figure 13.17: Recovery Regression Model for Pn Dominant



Source: RNC.

The distribution of the Pn Dominant domain is shown in Figure 13.18.

Figure 13.18: Distribution of the Pn Dominant Domain

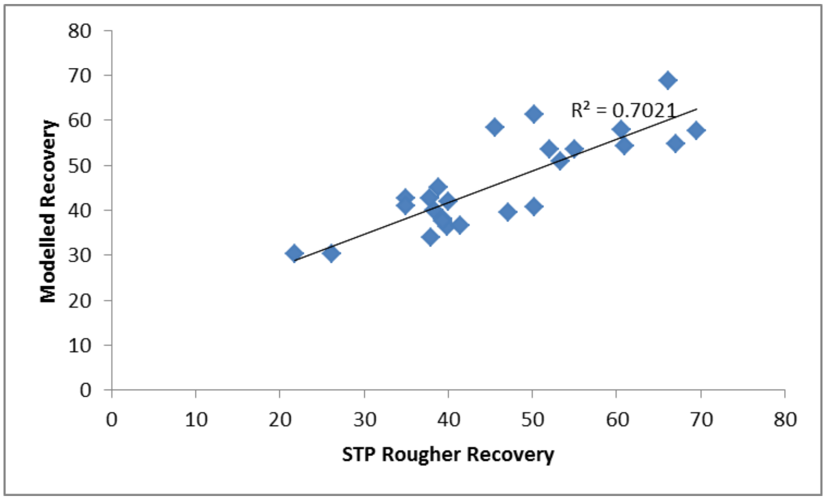


Source: RNC.

13.8.1.4 Results for High Iron Serpentine: FESP >= 14%

Rougher Ni Recovery = 14.83 + 38.9*S/Ni + 0.0143* Cr

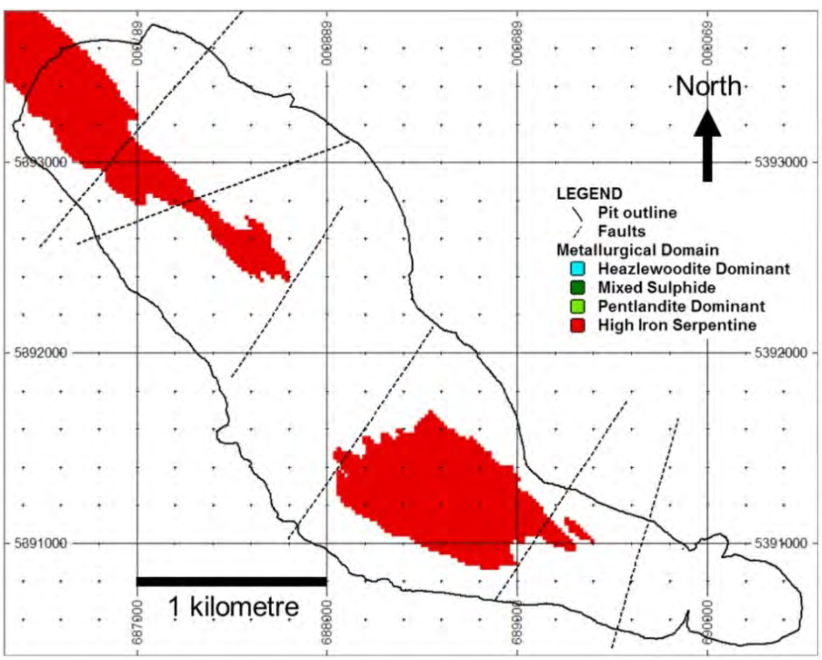
Figure 13.19: Recovery Regression Model for High Iron Serpentine



Source: RNC.

The distribution of the High Iron Serpentine domain is shown in Figure 13.20. It is almost exclusively located in structural domain 3, with limited amounts found at depth in structural domain 5 in the north.

Figure 13.20: Distribution of High Iron Serpentine Domain



Source: RNC.

13.8.1.5 Rougher Ni Recovery Equations Summary

Based on this analysis the final recovery equations used in the FS were as follows:

- Hz Dominant Domain: (FESP < 14%, Hz/Pn ≥ 5):

  Rougher NI Recovery = 18.11 + 0.0211*S + 0.00039 * Fe
- Mixed Sulphide Domain (FESP < 14%, 1 < Hz/Pn < 5):

  Rougher Ni Recovery = 9.73 + 0.222*S + 0.0111*Ca
- Pn Dominant Domain (FESP < 14%, Hz/Pn <=1)

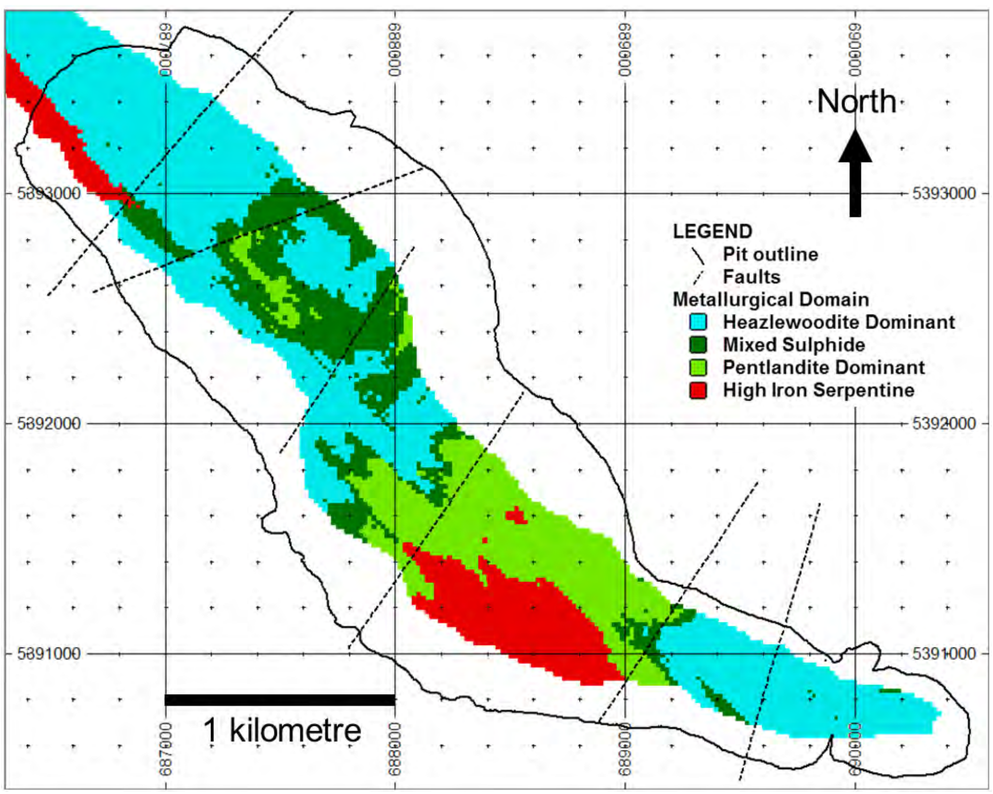
  Rougher Ni Recovery = 43.6 + 0.0055* S + 0.0111* Ca
- High Iron Serpentine Domain (FESP >=14%)

  Rougher Ni Recovery = 14.83 + 38.9*S/Ni + 0.0143* Cr

Each equation was applied to the entire modelled resource for Structural Domains 1 to 7 on a block-by-block basis.

Overall the distribution of the metallurgical domains within the FS pit shell is shown in Figure 13.21.

Figure 13.21: Metallurgical Domains within the FS Pit Shell



Source: RNC.

13.8.2 Cleaning Recovery

Several locked cycle tests were completed on different samples to assess the cleaner performance across a variety of feed characteristics. A summary is provided in Table 13-35.

The cleaner recoveries in Table 13-35 for LCT Test # 4-8 do not include the contribution from the slimes stream. The results from LCT Test 9-17 include the contribution from the slimes stream.

Cleaner recovery is highly correlated to sulphur in the feed sample, because of this the Hz Dominant samples, which have lower sulphur in feed for the same amount of recoverable minerals were separated from the other three metallurgical domains.

The locked cycle tests of the Hz domain samples showed high cleaner recovery irrespective of sulphur grade in feed. The average from the four locked cycle tests for the Hz Dominant domain was 92%. A cleaner recovery for 90% was assumed for all Hz Dominant blocks.

The Mixed Sulphide, Pn Dominant and Iron Serpentine Domain showed more variability in cleaning recovery, with lower cleaning recovery seen at low sulphur in feed. This is illustrated in Figure 13.22 (overleaf).

$$\textit{Cleaner Ni Recovery} = 0.1215 \ln(\%S) + 1.0959$$

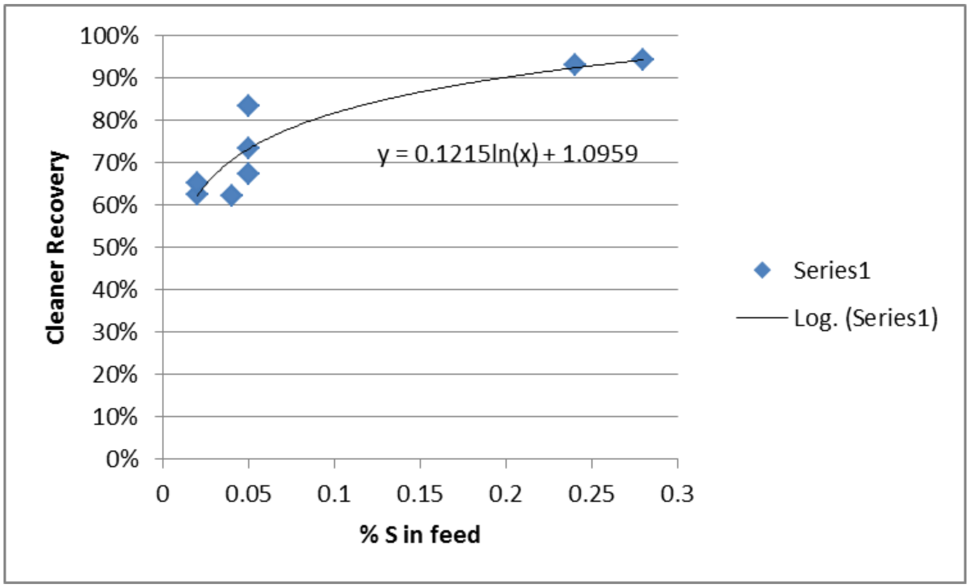
This equation was applied on a block-by-block basis to the Mixed Sulphide, Pn Dominant and Iron Serpentine domains within the resource.

Table 13-35: Locked Cycle Cleaning Test Summary

					LCT Rougher Recovery		LCT Overall Recovery		
LCT Test #	Sample	Met Domain	%S	Hz+Pn	%Ni	Recovery	%Ni	Dist.	LCT Clnr Recovery
4	222AC	Hz Dom	0.15	0.66	2.5	65.4	32.7	61.7	94%
5	218I	Pn Dom	0.06	0.09	0.5	33.5	22.8	20.9	62%
6	218G	Pn Dom	0.05	0.17	0.6	40.9	25.9	30.0	73%
8	214C	Mixed S	0.24	1.05	2.6	68.5	30.1	63.7	93%
9	Outcrop	Hz Dom	0.14	0.38	2.0	62.2	22.9	53.1	85%
10	223C	Pn Dom	0.02	0.16	0.7	38.2	43.8	27.0	71%
12	222AC	Hz Dom	0.15	0.66	1.9	67.4	31.2	65.7	98%
13	222BDE	Hz Dom	0.09	0.17	1.2	53.8	31.8	47.7	89%
14	217B	Fe Serp	0.36	1.35	5.9	49.8	20.6	46.9	94%
15	222H	Pn Dom	0.03	0.03	0.47	38.3	18.4	23.8	62%
16	218G	Pn Dom	0.05	0.17	0.7	31.6	26.0	21.2	67%
17	216ABC	Fe Serp	0.1	0.41	1.5	27.4	19.0	22.8	83%

Source: RNC.

Figure 13.22: Relationship between Cleaner Recovery & %S in Feed



Source: RNC.

13.8.3 Slimes Recovery

Approximately 20% of the nickel in the feed reports to the slimes flotation circuit. Recovery from the slimes stream was not assessed in the STP. Work was conducted on several samples to assess recovery from the slimes and ability to upgrade to a saleable concentrate.

The results were very variable depending on the feed material. Samples that were high in sulphide had better slimes recovery; samples that were higher in Awaruite had lower slimes recovery. Addition of a magnetic recovery stage on the slimes was evaluated, but not found to increase recovery.

Cleaning of the slimes was tested as part of the locked cycle tests. The results are shown in Table 13-36. Additionally more samples were tested as part of the locked cycle cleaning tests.

Table 13-36: Slimes Nickel Recovery to Cleaner Concentrate

LCT Test #	Sample	Met Domain	Ni Dis. To Slimes	Ni Recovery to Final Conc*
9	Outcrop	Hz Dom	23.5	8.1
10	223C	Pn Dom	25.6	0.3
12	222AC	Hz Dom	4.9	2.6
13	222BDE	Hz Dom	13.4	0.7
14	217B	Pn Dom	13.0	0.5
15	222H	Pn Dom	13.6	0.1
16	218G	Pn Dom	14.0	0.3
17	216ABC	Fe Serp	10.7	1.0
			Average	1.7

Note: * after cleaning

For the purposes of the feasibility study 1.7% was added to the rougher recovery * cleaner recovery for each block.

13.8.4 Overall Recovery Formula

The overall recovery formula is as follows:

*(Rougher Recovery * Cleaner Recovery) + Slimes Recovery = Total Recovery*

To prevent over or underestimation from the linear rougher regression equations, capping was applied to rougher recovery on the block by block assay inputs. Any block which had higher assay values than maximum and minimum of the STP dataset for that domain were capped at the STP dataset limits.

This reduced the average rougher recovery from 51.6% to 49.5%, a reduction of 2.1% rougher recovery. After these input limits were applied, rougher recovery was limited to 80%, to reflect the maximum recovery seen in the STP tests.

The input sulphur assay for the cleaner recovery equation for the Pn Dominant, Mixed Sulphide and Iron Serpentine domain was capped to the limits of the STP data and cleaner recovery was limited to 95%. Neither of these caps significantly reduced the cleaner recovery.

Finally, if the calculated overall recovery per block was greater than the theoretical recovery per block, the recovery was limited to the theoretical recovery to attempt to minimize extrapolation errors. The theoretical recovery was calculated using the modal percentages of pentlandite, awaruite and heazlewoodite multiplied by the respective nickel tenors of each mineral (sourced from the electron microprobe data), divided by the block's nickel assay. This reduced the final recovery from 43.3 to 42.9%. This cap was applied to 64M tonnes or 5% of the Dumont reserve.

In aggregate the various rougher and cleaner capping reduced the deposit recovery from 45% to 43%.

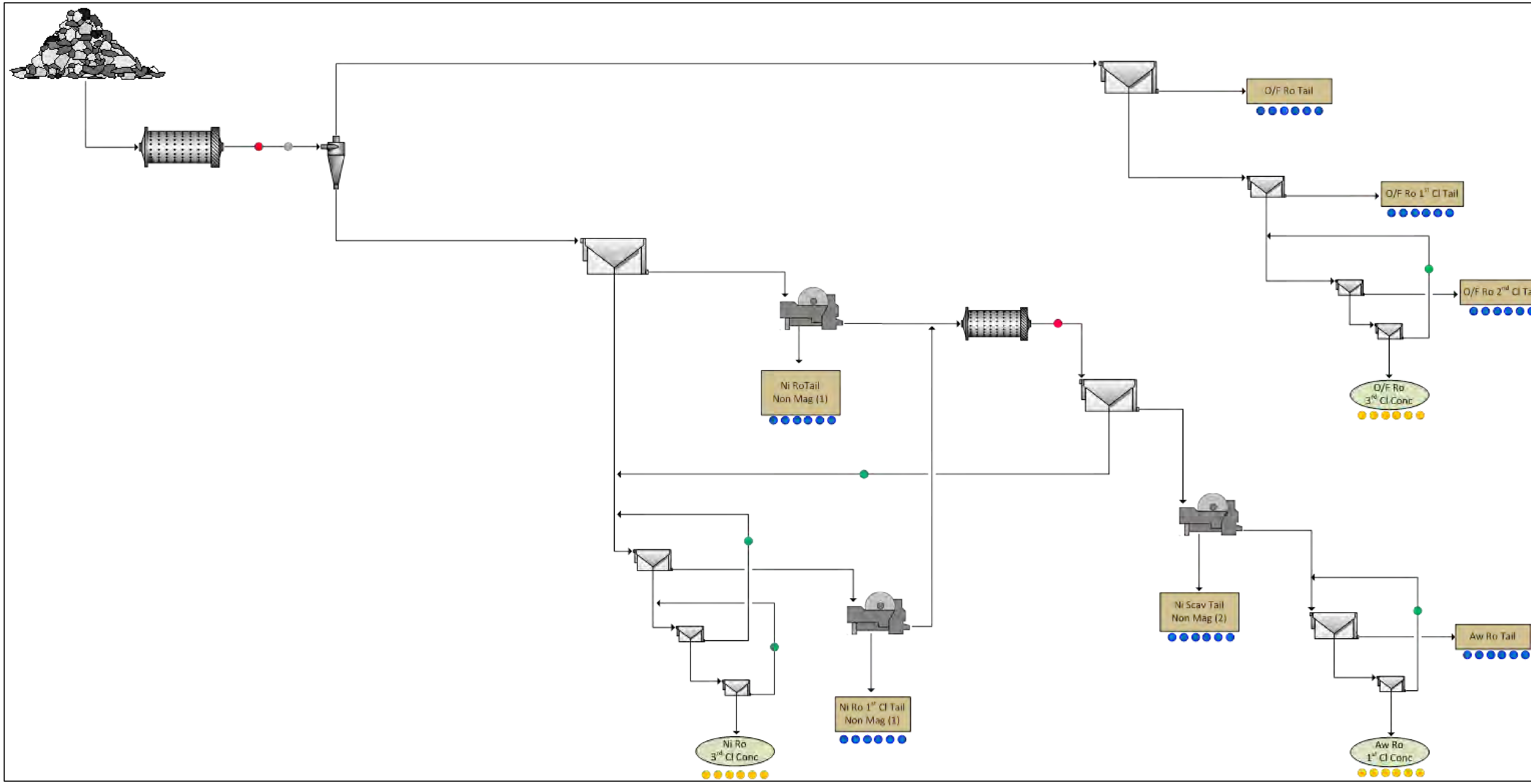
13.8.5 Confirmation of Flowsheet

Locked cycle tests of samples from different domains were completed to confirm the feasibility plant design basis and the recovery equations. The locked cycle tests were performed at CTMP.

Tests were performed on several samples, representing the four metallurgical domains as well as a range of recovery. Two datasets are presented. The first data set is from 2013 testing of the feasibility flowsheet, which includes 20% weight distribution to the slimes, separate slimes cleaning and combined rougher and combined scavenger cleaning with reagents and residence times as per the feasibility design basis. The second dataset is comprised of selected tests from 2011 and 2012 locked cycle tests that had separate slimes cleaning circuits, similar floatation times, and 20% weight recovery to the slimes portion.

The flowsheet for the 2013 locked cycle testing is shown in Figure 13.23.

Figure 13.23: 2013 Locked Cycle Confirmation Testing



The starting points for the study compared with the actual average used in the 2013 LCT testing (Table 13-37). The average reagent consumption for the 2013 locked cycle tests were less than the feasibility design basis, potentially indicating upside potential to the mill operating cost.

Table 13-37: Reagent Consumption for the 2013 Locked Cycle Tests

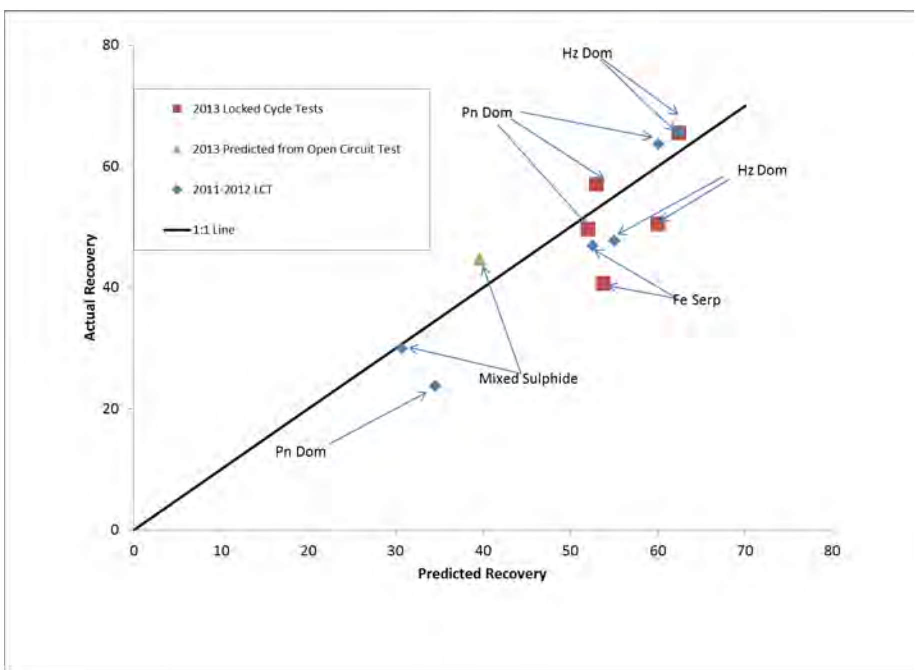
	PAX (g/t)	MIBC (g/t)	Cytec 65 (g/t)	Calgon (g/t)	CMC (g/t)	H_2SO_4 (g/t)	Cost ($/t)
FS Opex	80	89	2	254	6	3888	1.41
Actual (average of all tests)	89	77	0	135	16	5100	1.25

Six samples were tested in the 2013 flowsheet confirmation locked cycle testing. Testing focused on higher recovery samples that would be more representative of ore processed in the first five to six years. Three out of the four metallurgical domains (Hz Dominant, Pn Dominant and High Iron Serpentine) were tested, representing 90% of the material feeding the mill in the first five years.

Samples from selected 2011 and 2012 locked cycle tests were selected to compare to the newer 2013 tests and add confidence in the robustness of the testwork. These older tests used a similar flowsheet but slightly longer residence times and higher reagent dosages. However, with the previous work performed to optimize the reagents and residence time, it is expected that the results would be similar.

The overall recovery from the locked cycle tests is shown in Figure 13.24 compared to the recovery model used in the feasibility study. Variation around the model is shown; however, overall the model is adequately predicting the recovery seen in the locked cycle tests.

Figure 13.24: Locked Cycle Test Recovery Performance vs. Model



Source: RNC

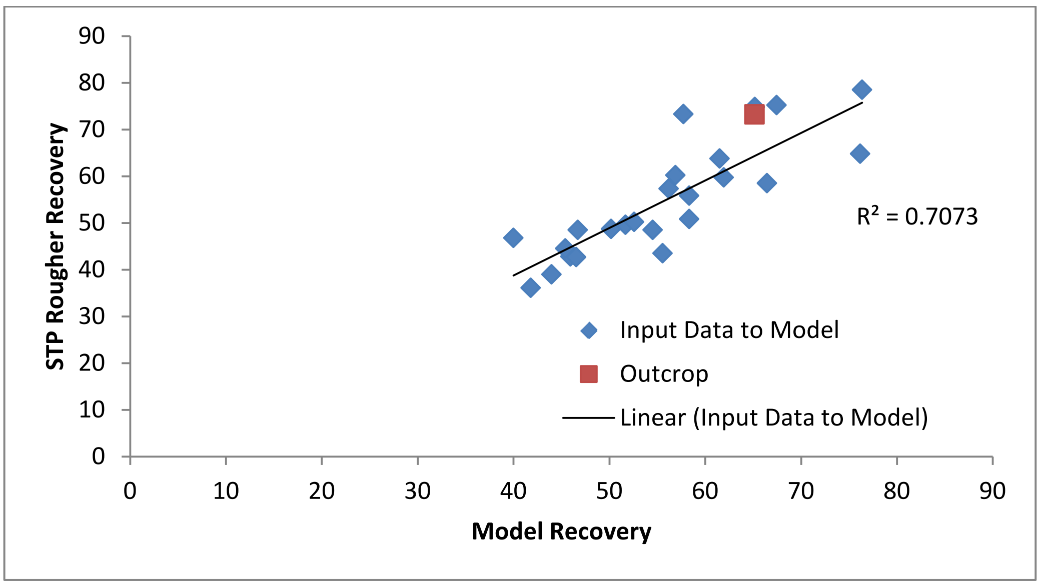
13.8.6 Effect of Stockpile Aging

Over the life of the mine, 606 Mt of ore will be stockpiled and subsequently processed by the mill at a later time, with an average duration on the stockpile of 12 years. It is not expected that significant aging will take place due to the low-nickel grade, low sulphur, highly disseminated nature of the mineralization.

An initial evaluation of material that was blasted in 1970 and left in a test pit on surface was tested under the STP (called the Outcrop Initial sample). The material behaved in a manner similar to freshly drilled core material (see Figure 13.25 for results). The predicted recovery from the rougher recovery equations (generated from fresh material) matches the laboratory test recovery on the aged material.

The recovery of the sample in the STP test exceeded model predictions.

Figure 13.25: Results from 1970s Test Pit Sample



Source: RNC

13.8.7 Byproduct Recovery

13.8.7.1 Cobalt

Within the Dumont deposit, cobalt is associated with the recoverable nickel minerals in the deposit; and similarly to nickel, it is also found in significant amounts in both serpentine and olivine. Consequently, the cobalt recovery is estimated to be tied to Ni recovery.

Electron microprobe analyses were performed to quantify the variability of cobalt content (tenor) in key minerals of interest for samples from locations throughout the Dumont deposit. Table 13-38 is a summary of the electron microprobe work that shows the low amount of cobalt in the serpentine, which makes up 92% of the mineralization. The cobalt in Serpentine is approximately 40 ppm on average. The overall cobalt assay in the resource is approximately 107 ppm. Therefore the cobalt contained in the serpentine represents 30-40% of the total cobalt in the deposit, which is similar to the nickel deportment. It also shows that the Cobalt is tied to the Pn and Aw minerals. Based on the deportment of cobalt in the recoverable minerals, Co recovery is assumed to equal nickel recovery. The average cobalt recovery for the life of the project is 42%.

Table 13-38: Cobalt Deportment by Mineral

Mineral	Minimum Value (% Co)	Maximum Value (% Co)	Average (% Co)	Number of Points	St.Dev
Pentlandite	0.34	35.94	3.72	840	4.64
Awaruite	0.02	5.05	0.85	534	0.90
Heazlewoodite	0.00	1.48	0.03	419	0.10
Serpentine	0.00	0.05	0.00	672	0.01

Source: RNC.

13.8.7.2 Platinum Group Elements

Concentrate produced from the initial locked cycle tests was combined into two composite samples to generate enough material for PGE measurement (labelled CA02195 and CA02469 in the table below). The remainder of the concentrates from the various locked cycle tests were sent individually to understand the variability of PGM recovery from different metallurgical domains as well as different S, Pt, and Pd assays in the feed. The concentrate samples were sent to SGS Mineral Services (Lakefield) for assay. The concentrate assay and recovery are shown in Table 13-39.

The calculated recoveries shown in Table 13-40 have some degree of error associated with them due to the low feed assays and inability to assay the tail samples (under the detection limit). The Pn Dominant, Mixed Sulphide, and Fe Serp domains generally show higher Pt and Pd recoveries and higher concentrations in concentrate than the Hz Dominant domain.

Table 13-39: PGE Concentration in Dumont Concentrate

Mineral	Pt (g/t)	Pd (g/t)	Met Domain	Pt Recy*	Pd Recy*
CA02195-APR11	2.4	4.7			
CA02469-MAY11	2.1	3.2			
RNC-214C	0.86	1.69	Pn Dom	44%	54%
SE_Outcrop1	0.67	1.22	Hz Dom	92%	75%
RNC-222AC	0.83	1.74	Hz Dom	46%	51%
RNC-222BDE	1.46	1.83	Hz Dom	61%	43%
RNC-217B	3.23	13.2	Fe Serp	159%	283%
RNC-222H	5.31	5.44	Pn Dom	85%	101%
RNC-218G	4.91	11.8	Pn Dom	127%	126%
RNC-216ABC	5.39	8.94	Fe Serp	99%	109%
Comp 1	2.12	2.53	Fe Serp	45%	36%
Comp 2	1.56	3.21	Fe Serp	68%	43%
Comp 3	1.47	2.71	Mixed Sulphide	128%	69%
Comp 4	1.47	2.43	Pn Dom	83%	66%
Comp 5	2.13	4.02	Pn Dom	115%	101%
Comp 6	2.05	3.58	Hz Dom	107%	54%
Comp 7	0.92	1.11	Hz Dom	51%	36%
Average	2.3	4.3		87%	83%

***Calculated based on units in concentrate / units in feed. Source:** RNC.

Table 13-40: Average Pt & Pd in Concentrates by Metallurgical Domain

Met Domains	Pt (g/t)	Pd (g/t)	Pt Recovery	Pd Recovery
Hz Dominant	1.9	2.5	72%	52%
Pn Dom, Mixed Sulphide, Fe Serp	2.6	5.6	95%	99%

In the block model for Hz Dominant blocks, an estimate of 50% Pt recovery and 36% Pd recovery were used. In the block model for Pn Dominant, Mixed Sulphide and High Iron Serpentine blocks, an estimate of 67% Pt recovery and 69% Pd recovery were used. These values reflect 70% of the lab recovery. The calculated Pt + Pd g/t in concentrate from these recoveries over the life of the project is 4.3 g/t, which is less than seen in the average of the locked cycle test concentrates.

13.8.8 Concentrate Quality

The concentrate from both the open circuit cleaning optimization tests and the locked cycle tests was composited and sent for assay to SGS Mineral Services (Lakefield) in several batches to analyse for impurity and PGE concentrations. Table 13-41 summarizes the results.

Table 13-41: Concentrate Assays

Sample	% Ni	%Cu	%Co	%Fe	%S	%MgO	%Cr	Pt (g/t)	Pd (g/t)
CA02195-APR11	34.5	0.6	0.5	25.7	23.5	4.0	0.03	2.4	4.7
CA02469-MAY11	39.2	0.6	0.6	27.5	23.1	3.1	0.04	2.1	3.2
CA02404-JUL11	32.8	N/A	N/A	18.5	11.8	13.3	0.04	N/A	N/A
CA02499-OCT11	34.9	N/A	N/A	21.1	16.5	8.7	0.13	N/A	N/A

Note: *N/A = no analysis was performed

The concentrate grades from the additional locked cycle tests were also reviewed. Additional electron microprobe data showed that the Pn in the High Fe SP area had an average Ni tenor of 26%.

For the FS the following concentrate grades were assumed for each metallurgical domain, based on the microprobe analysis summarized in Section 7 and the locked cycle tests.

- Hz Dominant Domain: (FESP < 14%, Hz/Pn ≥ 5): 35% Ni
- Mixed Sulphide Domain (FESP < 14%, 1 < Hz/Pn < 5): 35% Ni
- Pn Dominant Domain (FESP < 14%, Hz/Pn <=1): 30% Ni
- High Fe Serpentine Domain (FESP >=14%): 20% Ni

Based on these results the average life of project concentrate grade is 29% Ni, with a range of 22% to 33%.

Other impurities—such as As, Pb, Cl, and P—were all near or below detection limits in the measured samples. Zn was less than 0.05%, with the exception of CA02499-OCT11, which assayed 0.23% zinc.

13.8.9 Concentrate Transportation Criteria

Two samples of Dumont nickel concentrate were submitted for self-heating tests. To generate enough sample for testing the concentrate used for testing was a composite formed from the various locked cycle tests. One concentrate was Hz dominant and the other was Pn dominant. The following is a summary of the results from the Nesset report (Nesset, J.E., and Rosenblum, F., 2012).

The Dumont Ni concentrate samples do not exhibit any self-heating behaviour having Stage A and Stage B SCH values of 0.0 J/g. These results are not typical for a nickel concentrate. However they are expected due to the lack of pyrrhotite or pyrite contained in the Dumont concentrates.

13.8.10 Concentrate Chrysotile Content

No mineralogical analysis has quantified the amount of chrysotile in the concentrate. Although the goal of the Dumont nickel recovery process is to reject waste gangue (primarily serpentine) to the tailings stream, there is a still a portion of the concentrate that is made up of serpentine.

The range of serpentine in concentrate is expected to be approximately 20-25% by weight. Based on quantitative testing of the core, on average less than 2% of the serpentine in the ore is chrysotile.

Therefore is can be expected that the chrysotile content of the concentrate will be less than 1% and likely in the range of 0.4-0.5%.

Testing of concentrates from the locked cycle tests is recommended to confirm this value. Concentrate will be shipped from the site as a wet filter cake in closed containers; there is no risk of concentrate or chrysotile dispersion to the atmosphere during normal road or rail transport.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

SRK was retained by RNC to update the mineral resource estimate for the Dumont nickel project located near Amos, Québec. The Dumont nickel project is an undeveloped, large low-grade nickel deposit amenable to open pit mining. The nickel mineralization occurs in a complex assemblage of magmatic sulphides hosted in the dunite subzone of the Archean Dumont layered mafic intrusion.

In February 2011, RNC commissioned SRK to prepare a Mineral Resource Statement to support a preliminary feasibility study prepared by Ausenco Solutions Canada Inc. (Ausenco, 2011). An updated preliminary feasibility study was subsequently prepared considering drilling information available to February 1, 2012 and the recoverable magnetite data as of 8 May 2012 (Ausenco, 2012). The updated preliminary feasibility study was published by Ausenco on 22 June 2012.

This section summarizes an updated mineral resource model prepared by SRK to include new drilling information available to December 31, 2012. This revised model was used to support the feasibility study. The mineral resource evaluation work discussed herein represents the fourth Mineral Resource Statement prepared for this project, the third by SRK. The Mineral Resource Statement includes the second disclosure of palladium and platinum grade and magnetite concentrations.

The mineral resources reported herein consider drilling information available to 31 December 2012, and were evaluated using a geostatistical block modelling approach constrained by seven sulphide mineralization wireframes. The mineral resources have been estimated in conformity with CIM Mineral Resource and Mineral Reserves Estimation Best Practices Guidelines and are classified according to CIM Standard Definition for Mineral Resources and Mineral Reserves (November 2010) guidelines. The Mineral Resource Statement is reported in accordance with Canadian Securities Administrators' National Instrument 43-101.

The construction of the mineral resource model was a collaborative effort between RNC and SRK personnel. The construction of the three-dimensional resource domains was completed by RNC personnel and reviewed by SRK. Most of the resource evaluation work was completed by Mr. Sébastien Bernier, P.Geo (OGQ#1034, APGO#1847). Dr. Oy Leuangthong, P.Eng (APEGA#82746, PEO#90563867), assisted Mr. Bernier with the geostatistical analysis, variography, and the selection of resource estimation parameters. The open pit optimization work to test the "reasonable prospects for economic extraction" requirement for a mineral resource was completed by RNC personnel and by Mr. Anton Von Wiellingh, P.Eng, a mining engineer independent of RNC and SRK. The mineral resources are reported relative to an updated conceptual pit shell. Finally, this assignment benefited from the senior review of Mr. Glen Cole, P.Geo (APGO#1416), and Dr. Jean-Francois Couture, P.Geo (OGQ#1106, APGO#0197).

By virtue of their education, relevant project experiences, and affiliation to a recognized professional association, Mr. Bernier and Dr. Leuangthong are Qualified Persons independent of RNC for the purposes of National Instrument 43-101.

The block model was classified using criteria similar to that used for the preparation of the May 2012 Mineral Resource Statement (based on nickel cut-off grade, consideration of borehole spacing, the CAE Mining Studio 3 Mineable Reserve Optimizer application, and a final manual smoothing to ensure the continuity of similar class blocks). The final classification for nickel was applied to cobalt, palladium, and platinum. The mineral resource classification applied to magnetite follows the same sampling spacing approach as for nickel, in which three nearby boreholes are required within a radius of 120 m and 240 m for Indicated and Inferred categories, respectively. In the case of nickel, a radius of 60 m was required for Measured classification.

To ensure that only relevant platinum and palladium values are reported any values below or equal twice the assay detection limit for palladium and platinum were set to zero for the mineral resource estimation.

The Mineral Resource Statement for the Dumont project presented in Table 14-1 is reported at a cut-off grade of 0.15% nickel assuming a nickel price of US$9.00 per pound and an average recovery of 40%. The statement includes all classified blocks above the cut-off grade inside the conceptual open pit shells.

Table 14-1: Dumont Nickel Project, Quebec, SRK Consulting (Canada) Inc., April 30, 2013*

Resource Category	Quantity (kt)	Grade Ni (%)	Grade Co (ppm)	Contained Nickel (kt)	Contained Nickel (Mlbs)	Contained Cobalt (kt)	Contained Cobalt (Mlbs)
Measured	372,100	0.28	112	1050	2,310	40	92
Indicated	1,293,500	0.26	106	3,380	7,441	140	302
Measured + Indicated	**1,665,600**	**0.27**	**107**	**4,430**	**9,750**	**180**	**394**
Inferred	499,800	0.26	101	1,300	2,862	50	112
Resource Category	**Quantity (kt)**	**Grade Pd (g/t)**	**Grade Pt (g/t)**	**Contained Palladium (koz)**		**Contained Platinum (koz)**	
Measured	372,100	0.024	0.011	288		126	
Indicated	1,293,500	0.017	0.008	720		335	
Measured + Indicated	**1,665,600**	**0.020**	**0.009**	**1,008**		**461**	
Inferred	499,800	0.014	0.006	220		92	
Resource Category	**Quantity (kt)**	**Grade Magnetite (%)**		**Contained Magnetite (kt)**	**Contained Magnetite (Mlbs)**		
Measured	-	-		-	-		
Indicated	1,114,300	4.27		47,580	104,905		
Measured + Indicated	**1,114,300**	**4.27**		**47,580**	**104,905**		
Inferred	832,000	4.02		33,430	73,702		

Note: *Reported at a cut-off grade of 0.15% nickel inside conceptual pit shells optimized using nickel price of US$9.00 per pound, average metallurgical and process recovery of 40%, processing and G&A costs of US$6.30 per tonne milled, exchange rate of C$1.00 equal US$0.90, overall pit slope of 42° to 50° depending on the sector, and a production rate of 105 kt/d. Values of cobalt, palladium, platinum and magnetite are not considered in the cut-off grade calculation as they are byproducts of recovered nickel. All figures are rounded to reflect the relative accuracy of the estimates. Mineral resources are not mineral reserves and do not have demonstrated economic viability. The Measured and Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce Mineral Reserves.

Mineral resources are not mineral reserves and do not have a demonstrated economic viability. There is no certainty that all or any part of the mineral resources will be converted into mineral reserves. SRK is unaware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant issues that may materially affect the mineral resources.

The following sections summarize the data, methodology, parameters, and validation considered by SRK in estimating the mineral resources for the Dumont nickel project. Two models were constructed an elements model for 8 elements (calcium, cobalt, chromium, iron, nickel, palladium, platinum, and sulphur) and specific gravity and a minerals model for the distribution of ten minerals (awaruite, brucite, coalingite, high iron serpentine, heazlewoodite, serpentine, low-iron serpentine, magnetite, olivine and pentlandite). The mineral model was constructed to support ongoing metallurgical studies.

Full details of the data, methodology, parameters, assumptions, and validation considered and performed by SRK and summarized herein are included in Bernier and Leuangthong (2013), which is available on RNC's website.

14.2 Estimation Methodology

14.2.1 Resource Database, Preparation & Compositing

Exploration data available to evaluate the mineral resources include surface NQ core drilling information collected by RNC since 2007. The database includes 440 core boreholes (161,703 metres), and 90,967 assay samples. A total of 35 main elements are available for consideration. After discussions with RNC, SRK focussed on modelling the spatial distribution of eight main elements: calcium, cobalt, chromium, iron, nickel, palladium, platinum, and sulphur; and specific gravity.

For the minerals model, RNC provided a total of 1,561 EXPLOMIN™ data for the ten minerals (awaruite, brucite, coalingite, heazlewoodite, serpentine, low-iron serpentine, iron-rich serpentine, magnetite, olivine, and pentlandite), with approximately 74% of these data located within Domains 3, 4, and 5. 1,420 EXPLOMIN™ data points occurred within the mineralized domains and were used to inform the mineral model.

This section describes the resource domains used to constrain the estimation model, the available assays for analysis, compositing methodology, and the treatment of outliers for subsequent modelling. In addition, specific gravity data and its consideration in this resource estimate are also discussed.

14.2.1.1 Mineralized Domains & Geological Modelling

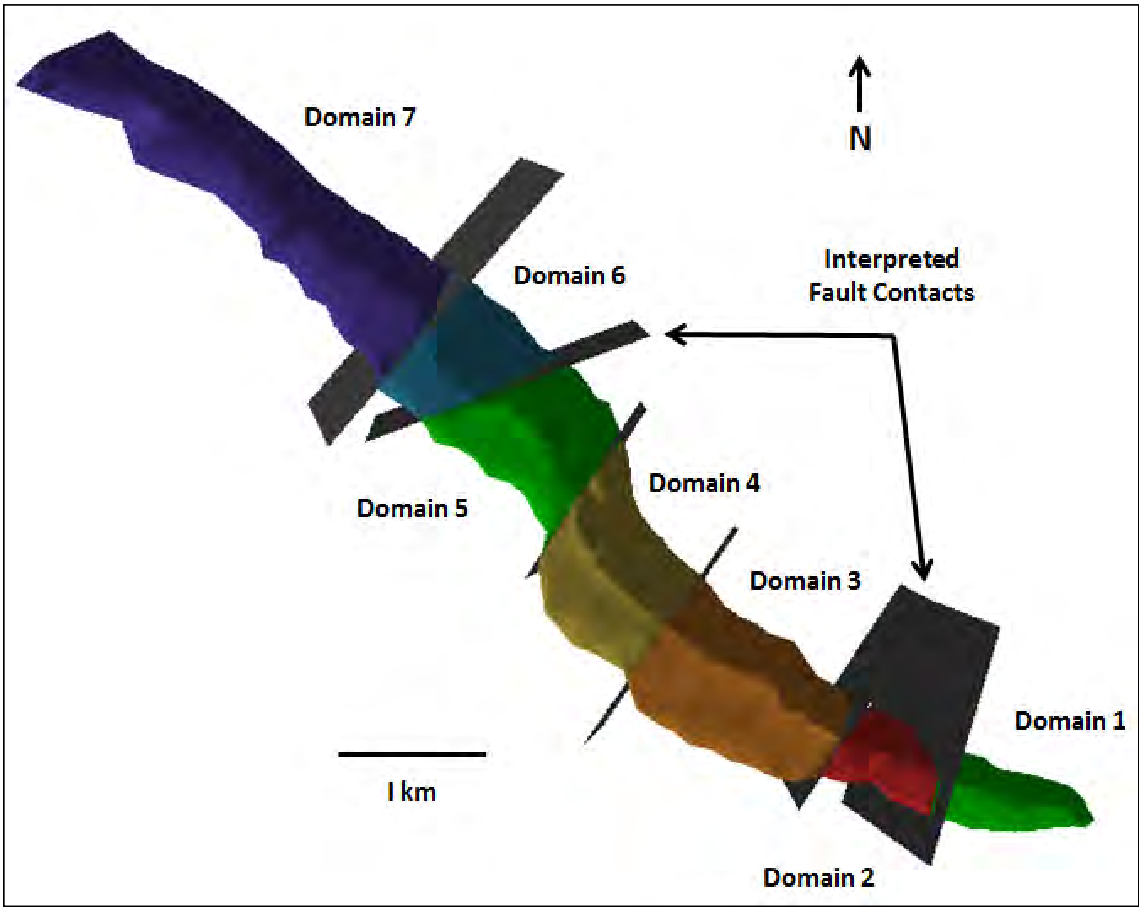
The geological interpretation and modelling of the deposit was performed by RNC staff, and delivered to SRK in the form of mineralization wireframes for use in constraining the resource estimation. SRK understands that RNC used a structural (fault) model, developed by Itasca Consulting in 2010 (Fedorowich, 2010) and updated in 2012 (Fedorowich, 2012), in conjunction with the definition of geological contacts and grade distribution defined by drilling, to construct several mineralized envelopes corresponding to structural domains. The fault model was reviewed and updated by SRK in 2011 (SRK, 2011), and the mineralized wireframes were updated accordingly. The structural model was revised at the end of 2012 by Itasca to reflect the drilling information acquired in 2012. Relative to those used in the previous resource model constructed by SRK in 2012, these mineralized envelopes are almost identical with minor differences at the boundary between domains and the dunite contact. Table 14-2 shows a comparison of wireframe volumes by domain. Most of the differences are due to modelling refinement at depth.

These envelopes were used to constrain the resource block model. Seven separate solids were generated (see Figure 14.1). The seven contiguous solids do not overlap spatially and are broadly constrained by a 0.20% nickel cut-off grade. SRK reviewed and confirmed that the majority of assays within the dunite subzone contain a minimum of 0.20% nickel.

Table 14-2: Volume Comparison of the 2012 & 2013 Geological Domains

Domain	Wireframe Volume (m^3) 2012	Wireframe Volume (m^3) 2013	Difference Volume (m^3)	Difference Percentage
1	55,160,936	53,228,618	(1,932,318)	-3.63%
2	45,331,916	39,188,549	(6,143,367)	-15.68%
3	205,411,411	222,659,537	17,248,126	7.75%
4	150,643,417	155,174,799	4,531,382	2.92%
5	221,945,776	187,045,497	(34,900,279)	-18.66%
6	103,610,361	102,331,944	(1,278,417)	-1.25%
7	272,332,991	257,641,602	(14,691,389)	-5.70%
Total	**1,054,436,808**	**1,017,270,546**	**(37,166,262)**	**-3.65%**

Figure 14.1: Distribution of the Seven Mineralized Envelopes Used as Resource Domains to Constrain Resource Estimation



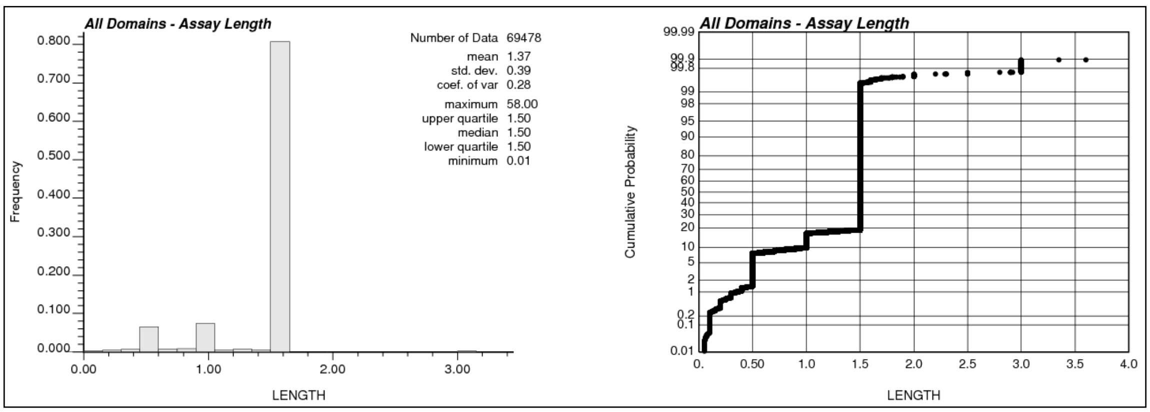
Source: SRK.

14.2.1.2 Exploratory Data Analysis & Compositing

The original assay data within the seven domains were extracted for statistical analysis, providing a total of 90,967 assay intervals for consideration, 69,478 of which intersect the resource domains. More than 99% of all samples were collected at intervals of 1.5 metres or less (Figure 14.2).

Given the large extent of this deposit and the anticipated block model vertical dimension of 15 metres (see Section 14.2.3), assay intervals were composited to a modal 7.5 metres downhole. Although unsampled assay intervals are rare in the data set, SRK assigned a detection limit value (Table 14-3) prior to compositing. Lost core intervals through fault zones were assigned an absent value.

Figure 14.2: Histogram & Probability Plot Showing the Distribution of Sample Length Intervals



Source: SRK.

Table 14-3: Detection Limit Values

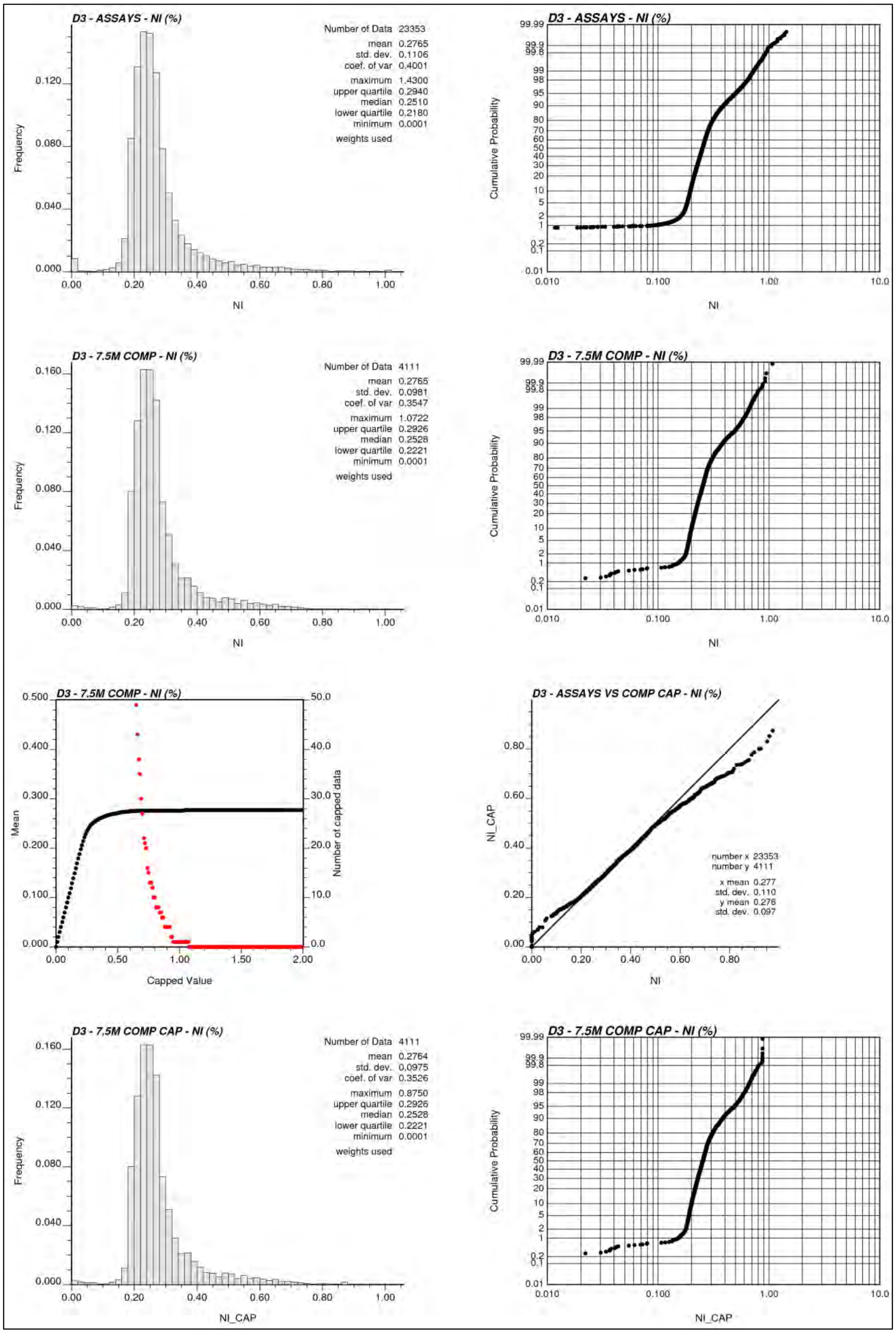
Element	Value	Unit	Element	Value	Unit
Ca	0.01	%	Ni	1	ppm
Co	1	ppm	Pd	0.001	ppm
Cr	1	ppm	Pt	0.005	ppm
Fe	0.01	%	S	0.01	%

Assay and composite statistics were calculated and analysed for each of the eight variables considering all domains together and each domain separately. Summary plots were generated to facilitate this analysis; see Figure 14.3 for an example plot for nickel in Domain 3.

SRK performed capping analysis by examining histograms, probability plots and assessing the sensitivity of the mean grade to prospective cap values. This was performed on a by domain basis.

Figure 14.3 shows an example of the plots used to assess capping values for percent nickel in Domain 3. Bernier and Leuangthong (2013), which is available on RNC's website, shows similar capping sensitivity plots for all other elements within each domain. The chosen capping values are given in Table 14-4.

Figure 14.3: Basic Statistics for Nickel in Domain 3



Source: SRK.

Table 14-4: Capping Values for Each Domain

Domain		Ca (%)	Co (ppm)	Cr (ppm)	Fe (%)	Ni (%)	Pd (ppm)	Pt (ppm)	S (%)
1	Cap Value	0.8	135	4100	7	0.34	0.055	0.028	0.21
	No. Capped	16	1	12	3	9	15	21	6
	% Equiv.	97%	>99%	96%	99%	98%	97%	96%	98%
2	Cap Value	0.5	125	3750	6.8	0.35	0.045	0.022	0.21
	No. Capped	37	10	7	3	7	30	38	18
	% Equiv.	92%	98%	98%	>99%	98%	95%	92%	96%
3	Cap Value	0.36	200	3200	8.75	0.875	0.7	0.34	0.85
	No. Capped	33	7	22	13	6	9	6	9
	% Equiv.	>99%	>99%	>99%	>99%	>99%	>99%	>99%	>99%
4	Cap Value	0.6	200	3750	9.75	0.75	0.285	0.14	0.75
	No. Capped	18	6	7	13	10	13	15	6
	% Equiv.	>99%	>99%	>99%	>99%	>99%	>99%	>99%	>99%
5	Cap Value	0.6	180	3300	9.25	0.625	0.14	0.065	0.45
	No. Capped	38	4	6	15	11	13	24	19
	% Equiv.	98%	>99%	>99%	>99%	>99%	>99%	99%	99%
6	Cap Value	1.4	140	2600	6.75	0.55	0.08	0.075	0.33
	No. Capped	25	7	6	8	3	26	15	6
	% Equiv.	97%	>99%	>99%	>99%	>99%	96%	98%	99%
7	Cap Value	0.9	140	2300	7.2	0.38	0.11	0.038	0.105
	No. Capped	25	3	10	4	9	9	26	32
	% Equiv.	98%	>99%	99%	>99%	99%	99%	98%	98%

A comparison of the assay and composite summary statistics was also compiled for each variable and within each domain. The full set of tables and plots is provided in Bernier and Leuangthong (2013), which is available on RNC's website; a table comparing the summary statistics for percent nickel is shown in Table 14-5.

Table 14-5: Summary Assay, Composite & Capped Composite Nickel (%) Statistics by Domain

	Original 1.5 m Assays				7.5 m Composites			7.5 m Capped Composites		
Domain	Count	Missing	Mean	Std Dev	Count	Mean	Std Dev	Count	Mean	Std Dev
ALL	69,478	0	0.270	0.098	12,705	0.270	0.087	12,705	0.270	0.085
1	2,528	0	0.250	0.056	515	0.246	0.049	515	0.244	0.045
2	2,530	0	0.250	0.050	481	0.250	0.041	481	0.249	0.038
3	23,353	0	0.277	0.111	4,111	0.277	0.098	4,111	0.276	0.098
4	18,100	0	0.279	0.108	3,263	0.279	0.095	3,263	0.278	0.093
5	12,324	0	0.276	0.095	2,293	0.276	0.084	2,293	0.275	0.083
6	4,891	0	0.254	0.076	940	0.254	0.069	940	0.254	0.067
7	5,752	0	0.247	0.048	1,102	0.247	0.041	1,102	0.246	0.038

14.2.1.3 Specific Gravity

Specific gravity measurements were made at the ALS Chemex Laboratory (ALS) in Vancouver (Canada) using a pycnometer on the pulp material as part of the routine assaying procedures. The specific gravity database contains 51,934 measurements. Missing intervals were assigned the average specific gravity value of that particular domain; as summarized in Table 14-6.

Table 14-6: Summary of the Specific Gravity Database

Domain	Available No. of Data	Missing	Percentage of Missing	Applied Average Value
1	2,315	213	8%	2.605
2	2,326	204	8%	2.605
3	17,582	5,771	25%	2.556
4	9,948	8,152	45%	2.575
5	10,648	1,676	14%	2.583
6	3,920	971	20%	2.586
7	5,195	557	10%	2.608

Given the dense sampling of specific gravity available for the Dumont deposit, SRK decided to populate the block model with specific gravity values using geostatistical estimation. As a result, SRK estimated nine variables (eight main elements and specific gravity) for each of the seven domains.

The next sections describe the spatial analysis and estimation parameters used to construct the three-dimensional block model for these nine variables.

14.2.2 Variography

SRK evaluated the spatial distribution of nine elements using a variogram and a correlogram for each element and its normal score transform. A total of four spatial metrics were considered to infer the correlation structure of each element for use in the grade estimation. Continuity directions were assessed based on the orientation of the wireframes, composites and the spatial distribution of the element. Further, variogram calculation considered sensitivities on orientation angles prior to finalizing the correlation orientation. All variogram analysis and modelling was performed using the Geostatistical Software Library (GSLib; Deutsch and Journal, 1998).

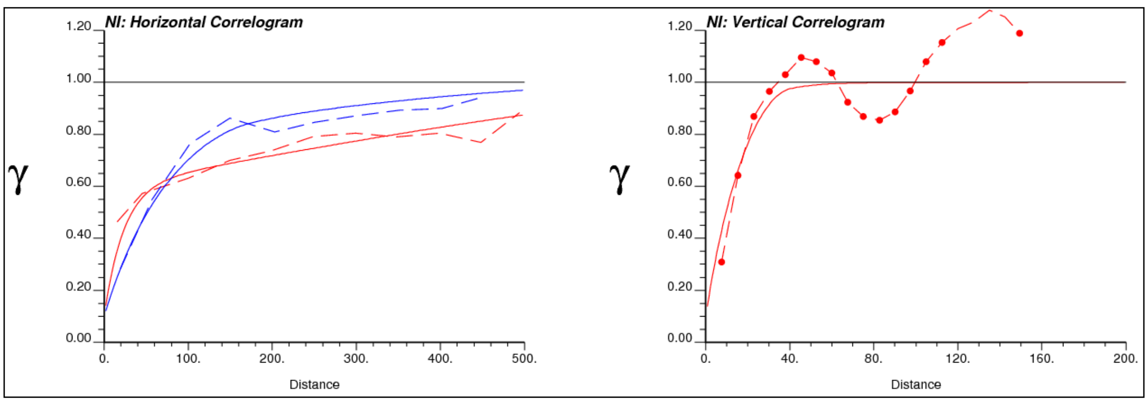
Variogram modelling was based on the combination of the four metrics, and in almost all cases, the correlogram of the main element yielded reasonably clear continuity structures that are amenable to variogram fitting.

For Domains 3, 4 and 5, variograms were calculated and modelled on a domain basis, using all capped composites within that particular domain. Relative to these three domains, the remaining four domains have considerably fewer composites leading to unreliability in the spatial correlation model. For these four domains (1, 2, 6 and 7), the variograms for each domain were considered separately and generally resulted in poor variogram inference.

SRK also assessed variograms for the combined Domains 1 and 2 since they are adjacent to each other; this yielded reasonable variograms to reliably infer a spatial model. A similar strategy was applied for Domains 6 and 7.

The modelled variograms used for the estimation of all elements (including specific gravity) for each domain are presented and illustrated in Bernier and Leuangthong (2013), which is available on RNC's website. Figure 14.4 shows an example of the correlogram calculated and modelled for nickel in Domain 3.

Figure 14.4: Correlogram of Percent Nickel in Domain 3 That Forms the Basis for Variogram Fitting



Note: The correlogram is inverted for the purposes of variogram modelling. The solid lines correspond to the fitted model while the dashed lines correspond to the experimental variogram in those same directions. **Source:** SRK.

14.2.3 Block Model & Grade Estimation

A block model was generated using CAE Mining Studio 3 software. In collaboration with RNC, SRK chose a block size of 20 by 20 by 15 metres after considering the borehole spacing, the extents of the modelled mineralization envelopes, and the anticipated open pit mining methods. The model was not rotated. SRK does not expect that adding a local grid would improve the accuracy or the precision of the mineral resource model. Subcells were used to honour the geometry of the modelled mineralization, but were subsequently recombined into the parent cell dimensions for open pit optimization. Subcells were assigned the same grade as the parent cell.

The block model coordinates are based on the local UTM coordinate grid (NAD83 datum, Zone 17). The definition of the Dumont block model is presented in Table 14-7.

Table 14-7: Dumont Block Model Characteristics

	Block Size (m)	Origin* (m)	Extent (m)	Number of Blocks
X	20	684,000	692,000	400
Y	20	5,390,000	5,395,500	275
Z	15	-700	425	75

Note: *UTM coordinates (NAD83 datum, Zone 17)

14.2.3.1 Estimation Strategy for Main Elements

Table 14-8 summarizes the general parameters used for the grade estimation. In all cases, grade estimation was based on ordinary kriging using three passes, with the first pass as the most restrictive in terms of search radii and number of boreholes required. Successive passes usually populated areas with less dense drilling, thus the corresponding parameters were relaxed with generally larger search radii and more relaxed data requirements.

Table 14-8: Estimation Strategy Applied to All Seven Resource Domains

Axis	1st Pass	2nd Pass	3rd Pass
Search Increment	Variogram range, up to Domain dimension	Twice the 1st pass range	Ten times the 1st pass range
Interpolation Method	Ordinary Kriging	Ordinary Kriging	Ordinary Kriging
Octant Search	Yes	No	No
Min Number of Octants	3	N/A	N/A
Min Number of Composites per Octant	2	N/A	N/A
Max Number of Composites per Octant	5	N/A	N/A
Minimum Number of Composites	9	5	3
Maximum Number of Composites	12	15	15
Maximum Number of Composites per Borehole	4	4	4

SRK assessed the sensitivity of the nickel block estimates to estimation parameters such as minimum and maximum number of data. Results from these studies showed that the model is relatively insensitive to increases in the maximum number of composites informing a block. For the first estimation pass, composites from at least three boreholes were necessary to estimate a block. This pass also used the octant search option. For subsequent passes the criteria were relaxed. In all cases, the search radii were chosen to reflect variogram continuity structure, ranges, and orientation. Bernier and Leuangthong (2013), which is available on RNC's website provides a complete listing of the specific search ranges per variable by domain and by estimation pass.

Table 14-9 provides statistics on the percentage of the block model filled by estimation pass on the basis of the nickel block model.

Table 14-9: Tonnage Estimated per Passes for All Seven Resource Domains

Domain	Estimation Pass	Tonnage Estimation	Percent Estimated
1	1	28,239,570	20.3%
	2	72,331,054	52.1%
	3	38,388,241	27.6%
2	1	24,541,984	24.0%
	2	69,031,919	67.6%
	3	8,592,607	8.4%
3	1	187,535,593	32.7%
	2	386,504,908	67.3%
4	1	133,756,287	33.4%
	2	266,323,667	66.6%
5	1	219,698,886	45.2%
	2	266,034,360	54.8%
6	1	132,323,946	50.1%
	2	131,613,253	49.9%
7	1	416,611,085	61.5%
	2	257,531,785	38.0%
	3	3,075,205	0.5%

14.2.3.2 Estimation of Mineral Abundances

To facilitate RNC's ongoing evaluation of metallurgical recovery, SRK also constructed estimation models of mineral abundances. Specifically, SRK modelled the abundance distribution of awaruite, brucite, coalingite, heazlewoodite, serpentine, low-iron serpentine, iron-rich serpentine, magnetite, olivine, and pentlandite. Mineral abundances may affect the metallurgical recovery, and thus may have a direct impact on project economics.

For this mineral model, a total of 1,420 EXPLOMIN™ samples occur within the mineralized envelope for the nine minerals, with approximately 74% of these data located within Domains 3, 4, and 5. In light of the estimation algorithms sensitivity study completed for the last resource model in 2012, SRK applied ordinary kriging to model all mineral abundances. SRK checked the distributions of the block models with the declustered, change-of-support corrected distributions of the EXPLOMIN™ data to verify the reasonableness of the statistics of the final block model.

Bernier and Leuangthong (2013), which is available on RNC's website, provides details related to the variography of the mineral abundance model, the estimation parameters, preparation of the final model delivered to RNC, and the quantitative comparisons performed by SRK.

14.2.4 Resource Model Validation

To validate the block estimates, SRK constructed parallel estimation models for nickel using an inverse distance (power of two) estimator as an alternate estimation method. SRK visually compared the results against the ordinary kriging model and found similar trends in both models. SRK also checked that the global quantities and average percent nickel grade from each method are reasonably comparable. The global estimated values were also checked against the declustered mean grade (Bernier and Leuangthong, 2013), which is available on RNC's website.

For the 2011 resource model (Ausenco, 2011), SRK also constructed geostatistical simulation models for nickel, iron, sulphur, calcium, cobalt, chromium, and specific gravity. At that time, these simulation models were used to check against the estimation model results. Specifically, the global grade-tonnage curves from each of the 100 simulation models were calculated (unconstrained by pit shells) and within domains, and compared against those obtained from the estimation model. The comparison showed that the 2011 estimation model did a reasonable job of honouring the scale differences between the informing composites and the resultant block model.

Geostatistical simulation models were not reconstructed using this latest database; however, SRK deems that the model parameters and general input information for the 2012 resource model did not change appreciably. As such, SRK expects that this latest estimation model should compare just as well against a simulation model.

The mineral abundance models were also validated by constructing a series of parallel estimation models using an inverse distance (power of two) estimator as an alternate estimation method. SRK visually compared the results against the ordinary kriging model and found similar trends in both models. SRK also checked that the global quantities and average estimated value from each method are reasonably comparable. The global estimated values were also checked against the declustered mean (Bernier and Leuangthong, 2013), which is available on RNC's website.

14.2.5 Mineral Resource Classification

In early 2011, SRK completed a study on the optimum borehole spacing to be considered for the resource classification. The study considered the classification of resources in the presence of grade uncertainty, which was assessed via geostatistical simulation. The scope of the study was limited to Domains 3, 4, and 5 and was performed for nickel only, with a borehole database and wireframes that were current up to December 6, 2010. The study was based solely on grade uncertainty and did not consider uncertainties related to the quantity and quality of the exploration database, sample collection procedures, or the confidence in the geological interpretation. The results of the study showed that, depending on the domain, a borehole spacing of 40 to 60 metres may be reasonable to classify Measured mineral resources, and a borehole spacing of 110 to 140 metres may be reasonable for Indicated mineral resources. Most of the drilling completed since this 2011 study consists of infill drilling on 50 metres or 100 metres sections. SRK considers the results of this study still valid and appropriate for resource classification.

Using the results of borehole spacing study, SRK developed a four-step approach to classification:

1. Identify blocks that satisfy specified borehole spacing criteria, requiring a minimum of two boreholes to be within:
   - 60 x 60 m borehole spacing for Measured
   - 120 x 120 m borehole spacing for Indicated
   - 240 x 240 m borehole spacing for Inferred.

2. Use CAE Mining Studio 3's Mineable Reserve Optimizer (MRO) module to ensure practical continuity of blocks assigned a given category, particularly for those classified in Step 1 as Measured. The following MRO parameters were specified:
   - Five percent maximum amount of material of a different class allowed for an envelope to be created
   - Search is constrained to only existing blocks
   - Minimum envelop size of 100 x 100 x 90 m, which approximates a nominal mass of 2.5 Mt
   - Minimum increment of the envelope by one block of 20 x 20 x 15 m.
3. Visualize MRO envelopes to ensure continuity of Measured blocks and tagging of blocks based on MRO results.
4. Manual smoothing of block classification to avoid isolation of individual cells in areas of predominantly different class. Isolated blocks are reclassified to the classification of surrounding blocks.

The methodology described above was used to classify nickel and was applied to cobalt, palladium, and platinum. The same approach was used for magnetite but the borehole spacing requirements were adjusted to map blocks coded with a minimum of three boreholes as described in Step 1 above. SRK ran some sensitivities related to increasing the number of boreholes found within the distance criterion, and found that specifying three boreholes yielded reasonable regions for classification purposes. This more restrictive criterion accounts for the uncertainty associated with the sparser database used to estimate magnetite, compared to that available for the estimation of nickel, cobalt, palladium, and platinum. Magnetite was only reported where nickel, cobalt, palladium, and platinum were reported.

14.3 Preparation of Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (November 2010) defines a mineral resource as:

> *"[A] concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge".*

The "reasonable prospects for economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade that takes into account extraction scenarios and processing recoveries. SRK considers that the nickel mineralization at the Dumont project is amenable to open pit extraction. In order to satisfy the "reasonable prospects for economic extraction," SRK is comfortable with reporting as mineral resource those classified blocks that are above the cut-off grade and fall within the extents of conceptual pit envelopes.

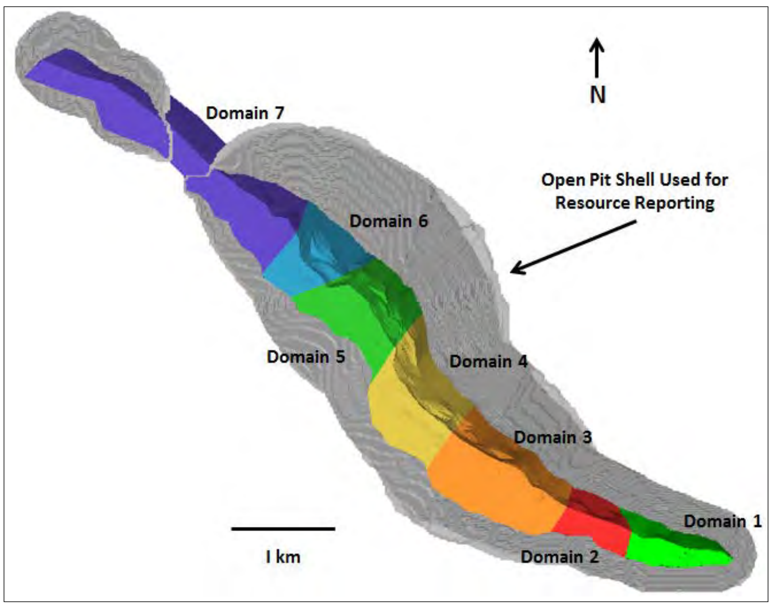
RNC conceptual pit shells (von Wielligh, 2013) were provided by Mr. Anton Von Wielligh, a mining engineer independent of RNC and SRK. The Mineral Resource Statement reported herein was prepared using a pit envelope developed with the 2012 mineral resource model. The optimization parameters used by Mr. Anton Von Wielligh are provided in Table 14-10. The reader is cautioned that the results from the pit optimization are used solely for testing the "reasonable prospects for economic extraction" by an open pit and do not represent an economic study as is required to evaluate mineral reserves.

Table 14-10: Conceptual Pit Optimization Assumptions for Open Pit Resource Reporting

Parameter	Assumption
Pit Slopes (per geotechnical sector)	42° to 50°
Process and G&A costs	US$6.30/t feed
Process recovery	40.0%
Assumed production rate	105 kt/d
Nickel price	US$9.00/lb

SRK considers blocks located within a conceptual pit shell to be amenable for open pit extraction (Figure 14.5) and can be reported as an open pit mineral resource.

Figure 14.5: Dumont Nickel Project Modelled Domains in Relation to Conceptual Pit Shell



Source: SRK.

14.4 Mineral Resource Statement

Mineral resources were classified according to CIM *Standard Definition for Mineral Resources and Mineral Reserves* (November 2010) guidelines by Mr. Sébastien Bernier, P.Geo (OGQ#1034, APGO#1847), an appropriate independent Qualified Person for the purpose of National Instrument 43-101. The mineral resources for the Dumont nickel project are reported at a cut-off grade of 0.15% nickel. The Mineral Resource Statement for the Dumont nickel project is summarized in Table 14-11.

The mineral resources are sensitive to the selection of reporting cut-off grade. To illustrate this sensitivity, the block model quantities and grade estimates are shown at various cut-off grades in Table 14-12 for Measured, Indicated and Inferred mineral resources. The reader is cautioned that these figures should not be misconstrued as a Mineral Resource Statement. The reported quantities and grades are only presented to illustrate the sensitivity of the resource model to the selection of a cut-off grade. The grade-tonnage curve is shown in Figure 14.6.

Table 14-11: Mineral Resource Statement, Dumont Nickel Project, Quebec, SRK Consulting (Canada) Inc., April 30, 2013 *

Resource Category	Quantity	Grade		Contained Nickel		Contained Cobalt	
	(kt)	Ni (%)	Co (ppm)	(kt)	(Mlbs)	(kt)	(Mlbs)
Measured	372,100	0.28	112	1050	2,310	40	92
Indicated	1,293,500	0.26	106	3,380	7,441	140	302
Measured + Indicated	**1,665,600**	**0.27**	**107**	**4,430**	**9,750**	**180**	**394**
Inferred	499,800	0.26	101	1,300	2,862	50	112
Resource Category	**Quantity**	**Grade**		**Contained Palladium**		**Contained Platinum**	
	(kt)	**Pd (g/t)**	**Pt (g/t)**	**(koz)**		**(koz)**	
Measured	372,100	0.024	0.011	288		126	
Indicated	1,293,500	0.017	0.008	720		335	
Measured + Indicated	**1,665,600**	**0.020**	**0.009**	**1,008**		**461**	
Inferred	499,800	0.014	0.006	220		92	
Resource Category	**Quantity**	**Grade**		**Contained Magnetite**			
	(kt)	**Magnetite (%)**		**(kt)**	**(Mlbs)**		
Measured	-	-		-	-		
Indicated	1,114,300	4.27		47,580	104,905		
Measured + Indicated	**1,114,300**	**4.27**		**47,580**	**104,905**		
Inferred	832,000	4.02		33,430	73,702		

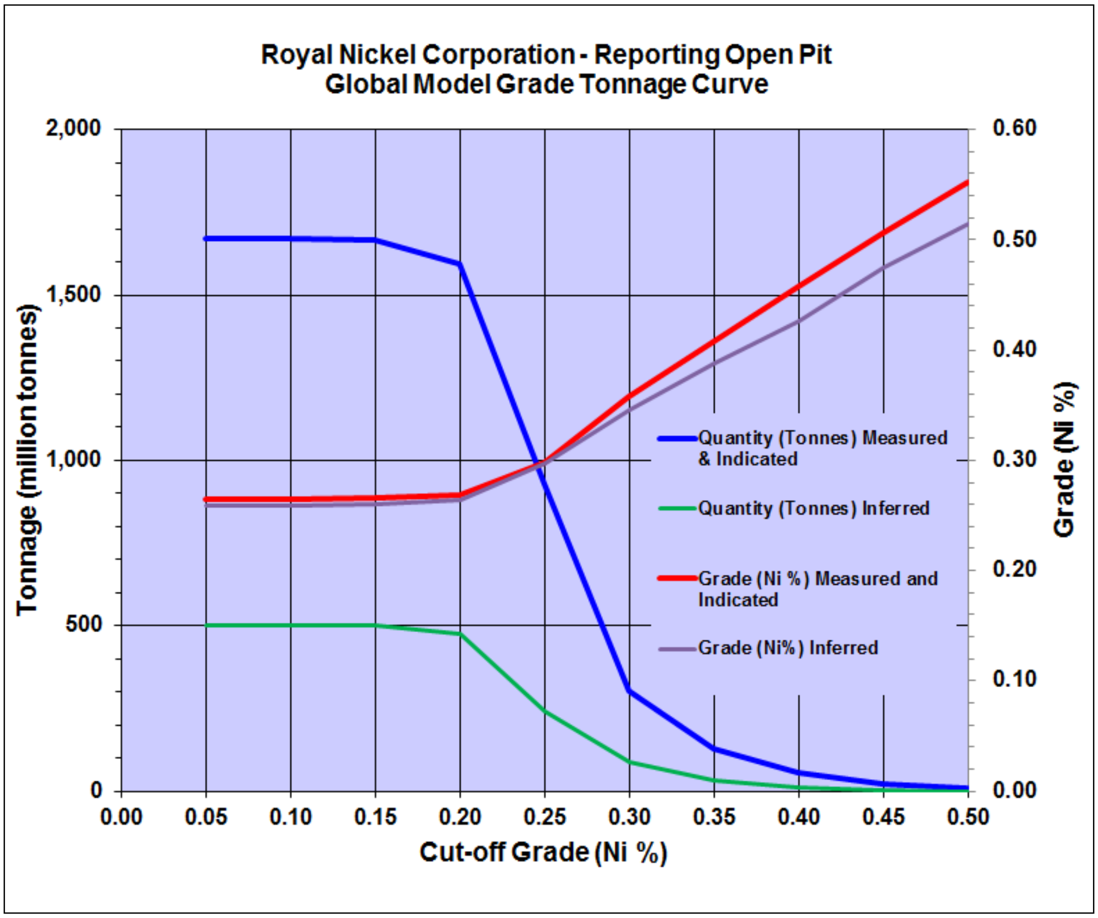
Note: *Reported at a cut-off grade of 0.15% nickel inside conceptual pit shells optimized using nickel price of US$9.00 per pound, average metallurgical and process recovery of 40%, processing and G&A costs of US$6.30 per tonne milled, exchange rate of C$1.00 equal US$0.90, overall pit slope of 42° to 50° depending on the sector, and a production rate of 105 kt/d. Values of cobalt, palladium, platinum and magnetite are not considered in the cut-off grade calculation as they are byproducts of recovered nickel. All figures are rounded to reflect the relative accuracy of the estimates. Mineral resources are not mineral reserves and do not have demonstrated economic viability. The Measured and Indicated Mineral Resources are inclusive of those Mineral Resources modified to produce Mineral Reserves.

Table 14-12: In-Pit Block Model Measured & Indicated Quantity & Grades* Estimates at Various Cut-offs

Cut-off Grade	Volume (km^3)	Tonnage (kt)	Ni (%)	Volume (km^3)	Tonnage (kt)	Ni (%)
Ni (%)	Measured & Indicated			Inferred		
0.05	704,148	1,669,361	0.27	223,950	501,219	0.26
0.10	703,920	1,668,817	0.27	223,896	501,078	0.26
0.15	**702,156**	**1,665,599**	**0.27**	**223,266**	**499,769**	**0.26**
0.20	664,008	1,596,031	0.27	209,832	474,311	0.26
0.25	369,708	928,925	0.30	96,402	239,325	0.30
0.30	119,502	302,261	0.36	34,800	87,630	0.35
0.35	49,956	126,176	0.41	12,564	31,688	0.39
0.40	21,240	53,629	0.46	3,900	9,836	0.43
0.45	8,922	22,387	0.51	630	1,613	0.48
0.50	3,972	9,854	0.55	78	200	0.52

Note: *The reader is cautioned that the figures presented in this table should not be misconstrued as a mineral resource statement. The reported quantities and grades are only presented as a sensitivity of the deposit model to the selection of cut-off grade.

Figure 14.6: RNC Dumont Project Grade-Tonnage Curve



Source: SRK.

15 MINERAL RESERVE ESTIMATES

15.1 Summary

The Dumont mineral reserves are summarized in Table 15-1.

Table 15-1: Mineral Reserves Statement* (Snowden, 17 June 2013)[1]

		Grades				Contained Metal			
Category	(kt)	Ni (%)	Co (ppm)	Pt (g/t)	Pd (g/t)	Ni (Mlb)	Co (Mlb)	Pt (koz)	Pd (koz)
Proven	179,600	0.32	114	0.013	0.029	1,274	45	77	166
Probable	999,000	0.26	106	0.008	0.017	5,667	233	250	550
Total	**1,178,600**	**0.27**	**107**	**0.009**	**0.019**	**6,942**	**278**	**328**	**716**

1. *Reported at a cut-off grade of 0.15% nickel inside an engineered pit design based on a Lerchs-Grossmann (LG) optimized pit shell using a nickel price of US$5.58 per pound (62% of the long-term forecast of US$9.00 per pound), average metallurgical recovery of 43%, marginal processing and G&A costs of US$6.30 per tonne milled, long-term exchange rate of C$1.00 equal US$0.90, overall pit slope of 42° to 50° depending on the sector, and a production rate of 105 kt/d. Mineral Reserves include mining losses of 0.28% and dilution of 0.49% that will be incurred at the bedrock overburden interface (which corresponds to mining losses of 1 metre and 2 metres of dilution along this contact). The Proven Reserves are based on Measured Resources included within run-of-mine (ROM) mill feed. Probable Reserves are based on Measured Resources included within stockpile mill feed plus Indicated Resources included in both ROM and stockpile mill feed. All figures are rounded to reflect the relative accuracy of the estimates.

Reserves were prepared under the direction of David A. Warren, Eng., Principle Consultant - Mining with Snowden Mining Industry Consultants, based on the mineral resource block model described in the previous Section. Reserves are estimated within an engineered pit design which is based upon a Lerchs-Grossmann (LG) optimized pit shell generated using a nickel price of US$5.58/lb, which is 62% of the long-term forecast of US$9.00/lb and include mining losses of 0.28% and dilution of 0.49%.

The proven reserves are based on measured resources included within run of mine (ROM) mill feed. Probable Reserves are based on Measured Resources included within stockpile mill feed plus Indicated Resources included in both ROM and stockpile mill feed. All figures are rounded to reflect the relative accuracy of the estimates.

In addition to Ni, Co, Pt and Pd, Dumont reserves contain 39.9 Mt of potentially economic magnetite.

15.2 Reserve Estimation Process

15.2.1 Summary

The reserve estimate uses the mineral resource block model that has been described in Section 14. The process of estimating reserves can be summarized as follows:

- The feasibility study (FS) mine design was initiated in Q2 2012, using the resource block model produced for the revised PFS that was published in Q2 2012. This model included the estimated content of the economic metals nickel, cobalt, platinum and palladium. The resource block model also included the estimated recovery of each economic metal to concentrate, and associated grade of Ni concentrate that would be produced, on a block-by-block basis.
- The net smelter return (NSR) for each block was calculated from the estimated content and recovery of economic metals and RNC's forecast of commercial terms (including long-term metal prices, exchange rate, percentage payables, and treatment and refining charges).
- The LG algorithm was used to define the theoretical ultimate pit shell for a nickel price of US$9.00/lb, as well as nested shells produced using lower nickel prices. The various nested shells were evaluated using a techno-economic model. The selection of a shell to use as the basis for the FS mine plan (and thus, reserve statement) was based on several factors, which included the incremental profit generated. The selection also took account of the tonnage of total reserves and associated operating footprint, as there are currently some limitations to the space available for infrastructural items such as waste dumps and the TSF. Finally, the selection took into account that the underlying resource model was from the PFS and subject to revision during the FS. The selected shell was thus conservatively based on a nickel price of US$4.86/lb or only 54% of the long-term forecast of US$9.00/lb. The resulting shell was smaller (and higher margin) than the shell that generated the maximum post-tax $NPV_{8\%}$, with the selected shell containing approximately 13% less ore and Ni.
- An engineered pit design was produced for the selected ultimate LG shell. This design used inter-ramp angles as recommended by the geotechnical consultants and ramps of sufficient width for the 230 t class trucks planned for use. The detailed engineering process resulted in a design that was superior to that produced by the LG algorithm, with the engineered design containing approximately 99% of the LG design nickel units in 100% of the LG ore, but with a reduction in stripping ratio of 0.15 tonnes of waste per tonne ore (for a total reduction in waste mined of 165 Mt or 11.5%).
- In parallel with the mine design process, the resource block model and recovery equations were updated based on additional data collected during the FS. The LG algorithm was re-run using the final FS resource model and recovery equations. The pit shell corresponding to the engineered design was generated with a Ni price of US$5.58/lb or 62% of the long-term forecast.
- Unplanned dilution and mining losses were added to the reserve estimate to reflect additional losses and dilution that would occur when mining at the upper contact between mineralization and overlying overburden. The gradational nature of mineralization near the hanging and footwall contacts, coupled with the methodology for interpolating grades into 20 m x 20 m x 15 m blocks was considered to have accounted for any possible dilution at these contacts and no extra unplanned dilution was added.
- A theoretical calculation of the cut-off grade was supported by an iterative investigation, which confirmed the highest project $NPV_{8\%}$ was achieved with the selected NSR cut-off of $7/t.
- Measured resources that would be treated as ROM ore were classified as proven reserves. Measured resources that would be initially stockpiled and all indicated resources were classified as probable reserves.

15.2.2 Revised PFS Resource Block Model

Generating an optimal mine design, including the selection of ultimate pit limits and schedule for mining the contained material, is an iterative process. Accordingly, to provide engineers sufficient time to evaluate the many potential scopes of design, the process was initiated in Q2 2012 using the resource block model that was produced for the revised PFS. In addition to geologic information (tonnage and grade of mineralization), the model also contained metallurgical information, including the expected recovery of metals to concentrate and the associated concentrate grade. The assumptions used to generate these estimates have been described previously, in the technical study for the revised PFS dated 22 June 2012.

As there has been limited resource drilling since completion of the PFS study, the mineral content of the revised PFS is materially similar to that of the current FS resource model and salient differences between the two models are limited to:

- Recovery equations developed for the revised PFS model were based on 83 Standard Test Procedure (STP) metallurgical tests, while those used in the current FS model are based on an additional 22 tests for 105 in total. (A discussion of the recovery equations is given in Section 13.7.) The earlier equations forecast an overall recovery of Ni for the selected ultimate pit shell of approximately 46% compared to 43% for the same shell using the current equations. Consequently, the revised PFS resource block model overstates the average value per tonne of mineralization contained within the selected pit shell by approximately 7.0% compared to the current FS model.
- Partially offsetting the overstatement of Ni recovery, the revised PFS model also classified the bulk of platinum and palladium (PGE) mineralization as inferred resources and therefore excluded from the design and analysis. PGE mineralization is now classified as measured or indicated resources due to locked cycle testwork conducted during the FS confirming that PGE mineralization can be recovered in economic concentrations to concentrate (see Section 13.7.5.2 for a discussion). PGE mineralization is now included in the design and analysis. The exclusion of PGE mineralization from the earlier model resulted in an understatement for average values contained within the selected pit shell of 2.0% compared to the current model.

The net impact is that the revised PFS resource model overstates the average value of resource blocks by approximately 5.0% compared to the current FS resource block model.

15.2.3 NSR Model

Each block of mineralization within the resource block model has a unique estimate of grade, metallurgical recovery and concentrate grade. These were then used to calculate a value of NSR per tonne using the parameters given in Tables 15-2 and 15-3.

Key assumptions used in the NSR calculations include the following:

- Table 15-2 reports the average concentrate grade over the entire life-of-project. The scheduled concentrate grade ranges from a low of 22.1% to a high of 33.2%, with associated TC/RCs ranging from US$1.23/lb to US$1.40/lb.
- Transport charges in Table 15-2 assume that 50% of total production will be treated in Sudbury, with the remainder evenly divided between Finland and China.

Table 15-2: Dumont NSR Calculation for Nickel

Item	Units	Value
Long-term Ni Price	US$/lb	US$9.00
Long-term C$ F/X	C$1.00 =	US$0.90
Long-term Ni Price	$/lb	$10.00
Concentrate Grade	% Ni	29.2%
Concentrate Transport	$/t	$71.12
Concentrate Treatment	US$/t	$175.00
Refining - Base Charge	US$/lb	$0.70
Price Participation Basis	US$/lb	$8.00
Refining Price Participation	% of incremental	10%
Total Transport + TC/RC	US$/lb	US$1.27
Total Transport + TC/RC	$/lb	$1.41
NSR	$/lb	$8.59
Payables	% of contained	93%

Table 15-3: Dumont NSR Calculation for Byproducts

Item	Units	Cobalt	Platinum	Palladium	Total[1]
Long-term Price[4]	US$/lb or oz	US$1,200	US$1,500	US$750	
Long-term C$ F/X	C$1.00 =	US$0.90	US$0.90	US$0.90	
Long-term Price	**$/lb**	**$13.33**	**$1,667**	**$833**	
Transportation & Treatment[2]	US$/t conc	$0	$0	$0	
Refining Charge	US$/lb or oz	$3.00	$50.00	$50.00	
Total Transport + TC/RC	**$/lb or oz**	**$3.33**	**$56**	**$56**	**$1.72**
NSR	**$/lb or oz**	**$10.00**	**$1,611**	**$777**	**$8.28**
Payables[3]	**% of contained**	**50%**	**76%**	**76%**	

Note: 1. Total expressed per lb payable NiEq in byproducts. 2. No incremental costs above what is paid for Ni concentrate. 3. PGE payables based on 1.0 g/t deduction, LOM average PGE concentrate grade of 4.3 g/t results in 76%. 4. Note that byproduct prices used in project evaluation were subsequently updated based on latest consensus forecasts (See Section 24).

15.2.4 LG Pit Shells – Initial Pass

The LG algorithm is the industry standard tool used to define the limits of an open pit. The design process was initiated by calculating the net value of each block in the model by subtracting estimated costs for mining, processing and administration from the NSR of each block (waste blocks with no NSR value have a negative net value).

Costs assumptions used in the LG evaluation were based on the full mill production rate of 105 kt /d and are as follows:

- Fixed costs of $2.90/t for mining clay and $2.35/t for mining granular overburden (which is mainly composed of sand and gravel, and thus referred to as sand and gravel). The depth of material was not considered as these lithologies overlie the mineralization and are thus shallow. The key cost driver is rather the horizontal distance to the ultimate impoundment. The costs used were based on the average distance traveled.
- Rock mining costs comprised of:
  - a base cost of $1.20/t (for blocks lying at or above the elevation of pit exits); and
  - a cost increment of $0.045/t for every 15 m bench below the exit.

- Marginal processing + G&A costs of $7.00/t ore, comprised of:
  - processing costs of $4.80/t;
  - G&A of $0.55/t;
  - TSF construction of $0.30/t;
  - plant sustaining capital cost of $0.35/t; and
  - rehandle (for low-grade material only) of $1.00/t.

Note the distinction of 'marginal' costs; these are costs that would be incurred by low-grade material initially stockpiled then re-handled. Higher grade material would not be subjected to the re-handle cost.

Overall slope angles were assigned to the various sectors based on the recommendations provided in the following section. These slopes ranged from 42° to 50°.

The LG algorithm then selected a 'cone' of ore and associated waste stripping that maximises NPV. By varying the metal price, it was possible to generate higher value nested cones that can be used to identify the optimal development sequence. With the revised PFS resource model that was used to generate shells initially, the lowest Ni price able to generate a cone was US$1.98/lb or 22% of the long-term price of US$9.00/lb. Shells were generated for each subsequent 1% increment in the Ni price then aggregated into 10 potential stages of mine development as summarized in Table 15-4.

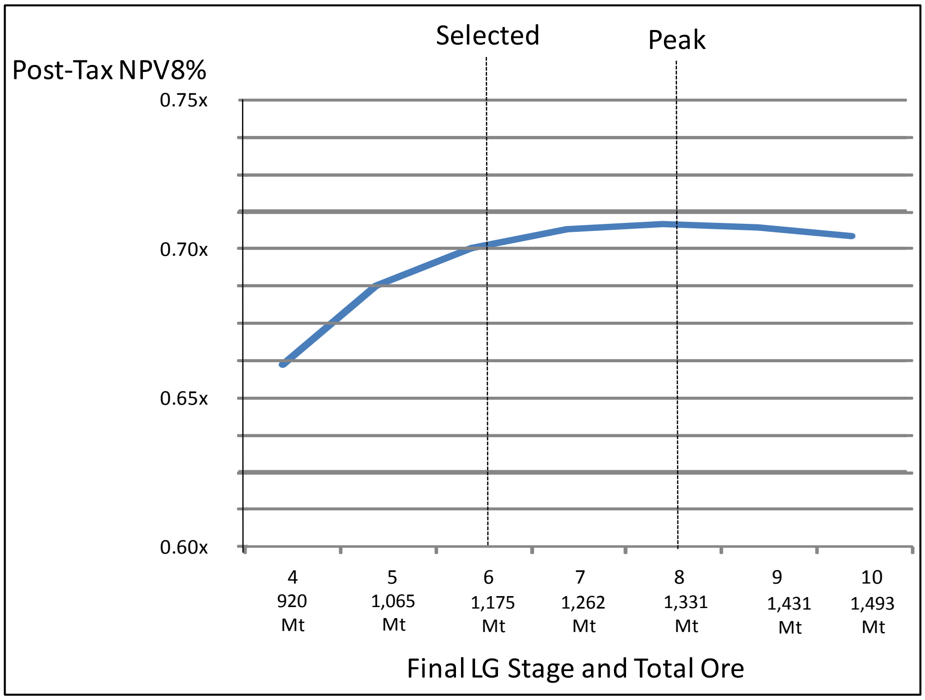
Table 15-4: First Pass LG - Nested Pit Shells

Stage	Revenue Factor	Nested Shell	Mill Feed (Mt)	Waste Rock (Mt)	O/B (Mt)	Stripping Ratio	Grade % Ni	Avg. Recovery	NSR $/t
1	33%	Shell 12	147	42	54	0.65	0.30	51.6%	$29.02
2	39%	Shell 18	196	156	34	0.97	0.28	47.0%	$24.28
3	44%	Shell 23	340	383	51	1.28	0.27	45.4%	$22.76
4	48%	Shell 27	237	331	30	1.52	0.26	45.2%	$22.24
5	51%	Shell 30	144	198	14	1.47	0.26	43.6%	$20.80
6	54%	Shell 33	111	202	9	1.91	0.26	45.0%	$21.65
7	58%	Shell 37	87	170	8	2.06	0.26	44.2%	$21.26
8	62%	Shell 41	69	155	6	2.33	0.26	42.6%	$20.65
9	70%	Shell 49	101	263	9	2.71	0.26	44.2%	$21.16
10	77%	Shell 56	62	172	6	2.87	0.26	41.0%	$19.59
Total to Stage 6			1,175	1,310	192	1.28	0.27	46.3%	$23.35
Total to Stage 8			1,331	1,635	206	1.38	0.27	45.9%	$23.07
Total to Stage 10			1,493	2,070	221	1.53	0.27	45.6%	$22.80
Selected as Ultimate Pit Shell									

The ten stages were evaluated using a spreadsheet techno-economic model. The evaluation revealed that NPV increased fairly rapidly until Stage 6 (1,175 Mt ore total), then moderated but continued to increase through Stage 8 (1,331 Mt ore total) then declined moderately through Stage 10 (1,493 Mt). It should be noted that the NPV for Stage 10 remained higher than that for

Stage 6 (see Figure 15.1). The impact of discount rate should also be noted – the incremental material in Stages 7 to 10 would require an extra 8.3 years of project life at a mill rate of 105 kt/d, and the trajectory in NPV in part reflects the impact of an 8% discount rate being applied to the longer life.

Figure 15.1: First Pass LG – Post-Tax NPV8% as a Function of Final Stage



Source: RNC.

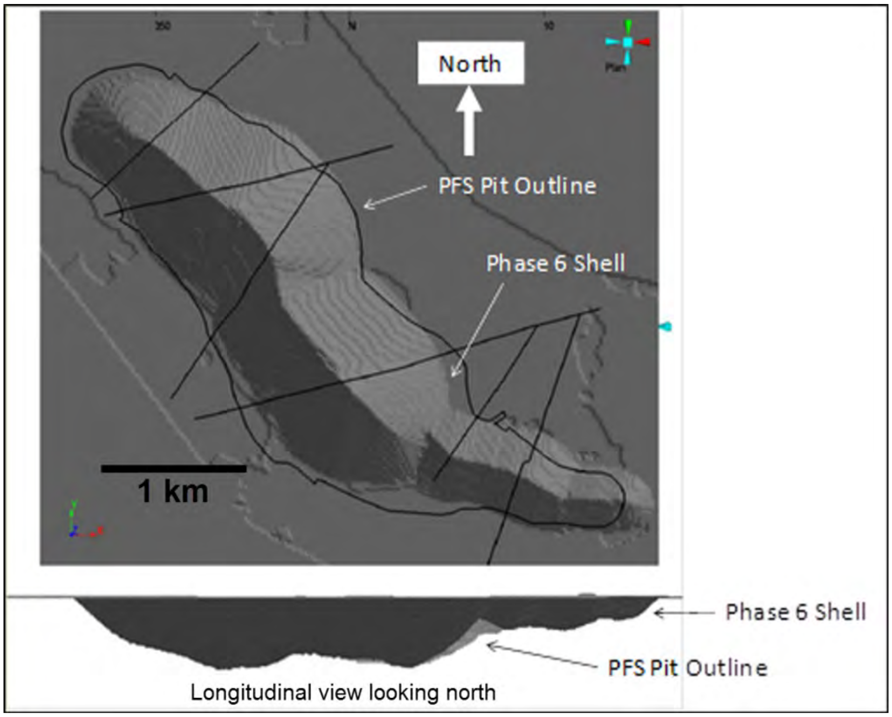
Nonetheless, Stage 6 was chosen as the basis for the FS mine design and associated reserve statement as:

- The rate of NPV accretion for subsequent stages slows.
- The resulting mine plan of approximately 1,175 Mt ore and 1,500 Mt waste could be accommodated using the footprint and concepts established during the PFS. Notwithstanding changes to the location of impoundments (the location of TSF and waste dumps effectively reversed), the FS design is materially similar to the PFS design. A significant increase to this tonnage (such as would occur with selection of Stage 8 as the ultimate limit) would be feasible, but could require new concepts for the impoundment of incremental material.
- Given the resource model on which the design was based was of PFS level accuracy, it was felt prudent to select a shell smaller than that which generated the maximum value.

Plan and sectional views of the LG Stage 6 design are given in Figure 15.2. It can be seen that the surface footprint and ultimate depth of pit are very similar to the earlier PFS design, though this shell contains approximately 10% more ore and total tonnes.

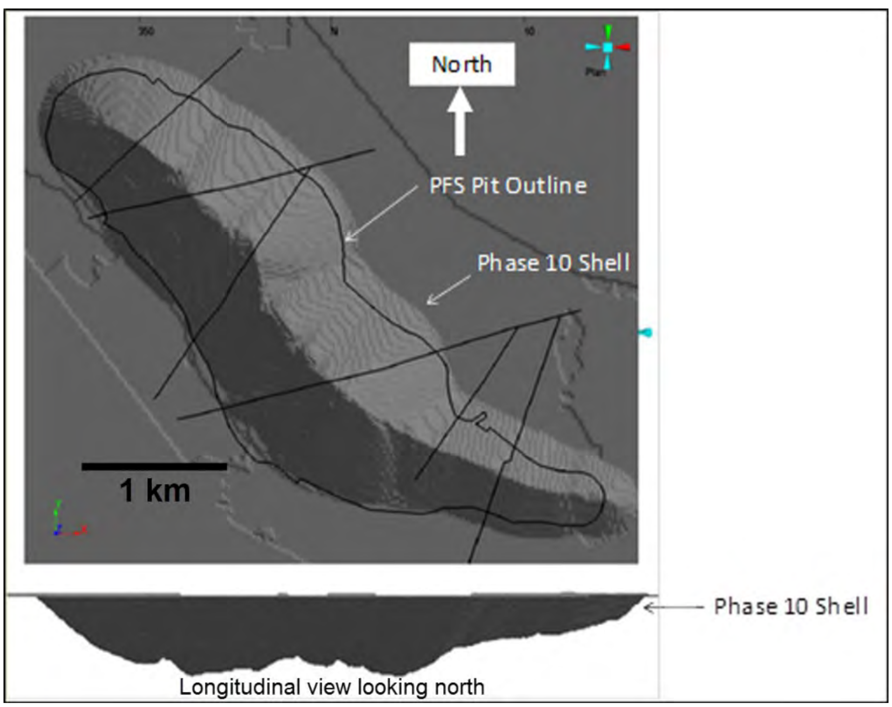
Figure 15.3 provides a similar view of the LG Stage 10 design. While the total ore and waste is 43% higher than Stage 6 (27% more ore, 58% more waste), the surface footprint for this design is not materially larger.

Figure 15.2: First Pass LG – Stage Shell Used as Basis of Pit Design (54% Revenue Factor)



Source: RNC.

Figure 15.3: First Pass LG – Stage 10 LG Pit Shell (77% Revenue Factor)



Source: RNC.

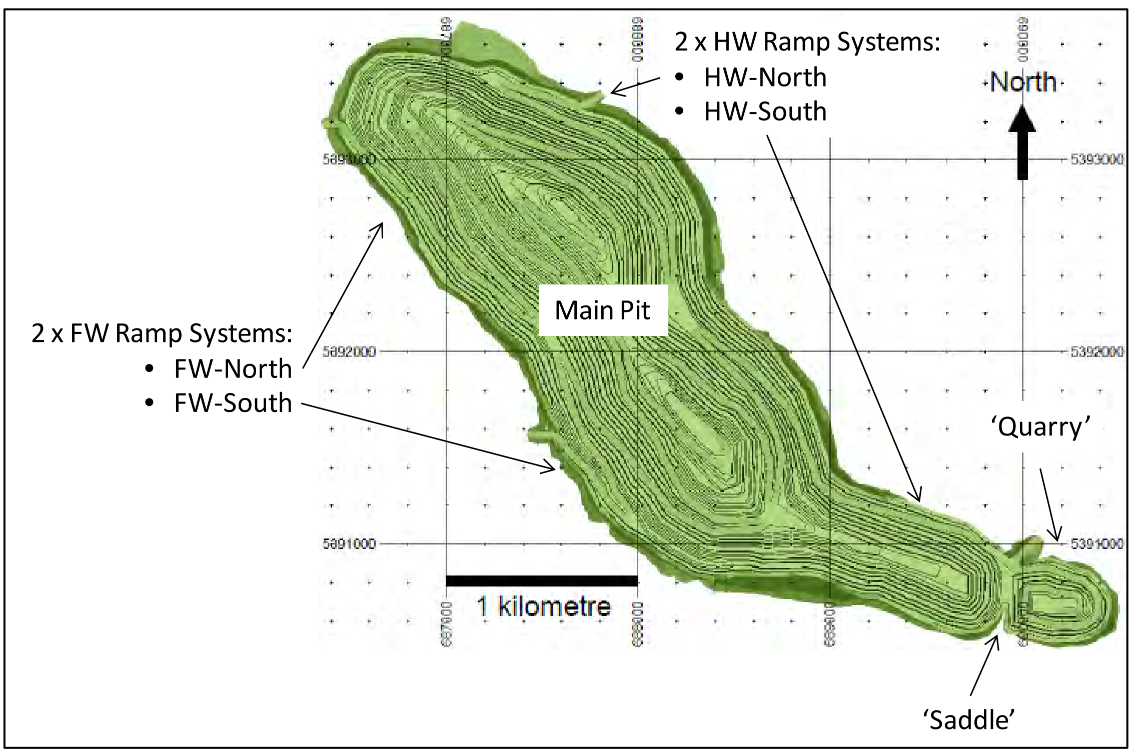
15.2.5 Engineered Pit Design

The LG shell pictured in Figure 15.2 represents the theoretical design, and while the final walls honour the imposed overall slope constraints, it cannot be considered a practical design as no provision is made for ramps. The engineered design includes 35 m wide ramps (approximately 4.5x width of a 230 t truck) and inter-ramp angles rather than the overall angles used as the basis for LG wall designs.

As can be seen in Figure 15.4, the engineered design provides dual-redundant ramp systems for both the hanging and footwalls. This not only reduces geotechnical risk (instability in one sector has a lesser impact when vehicles can choose from multiple exits), but also allows for split shell pushbacks that reduce the instantaneous mining rate. However, the dual redundant ramps result in marginally flatter final walls than used in the LG design. The engineered wall design has also been flattened by the inclusion of geotechnical berms of 20 m width, every 120 m vertically.

It can also be seen that a ‘saddle’ has been left between the southeast pit (which will be the site of the starter pit) and main pit. This is to allow use of the southeast pit as a water reservoir during mill operation.

Figure 15.4: Engineered Final Pit Design



Source: RNC.

The impact of slightly flatter slopes and leaving the saddle was largely offset by a rigorous and iterative design process that resulted in the optimization of material selected for inclusion within the final pit shell. As a result, the engineered shell is considered to be an improvement on the LG design, as shown in Table 15-5. It contains only 0.8% less Ni (60 Mlbs) in 0.1% more ore than the LG, reflecting a decrease in average grade of 1.0%. More significantly, the stripping ratio has been reduced by 11%, for a reduction of 165 Mt total waste rock and overburden.

Table 15-5: Comparison of Engineered & First Pass LG Designs

	Mill Feed (Mt)[1]	Grade % Ni	Contained Ni (Mlbs)	Waste Rock (Mt)	O/B (Mt)	Stripping Ratio
LG	1,175	0.271	7,024	1,310	192	1.28
Engineered	1,178	0.268	6,964	1,159	178	1.13
Variance	**0.1%**	**(1.0%)**	**(0.8%)**	**(11.5%)**	**(7.3%)**	**(11.0%)**

15.2.6 FS Resource Block Model & Second Pass LG Pit Shells

In parallel with the mine design process, the PFS resource block model and recovery equations were updated based on additional data collected during the FS. As noted previously, the inclusion of additional metallurgical testwork resulted in a reduction in average recovery for the selected pit shell from approximately 46% to 43% (see Section 13 for a discussion of recovery equations). This was partially offset by the reclassification of PGE mineralization as Measured or Indicated Resources due to locked cycle testwork conducted during the FS confirming that PGE mineralization can be recovered in economic concentrations to concentrate, allowing this material to be included in the evaluation. The net impact was a 5% reduction in the average value per tonne of mineralization.

The LG algorithm was re-run to confirm that the pit shell selected as the basis for the engineered pit design was in fact economic given changes to the resource block model. Additionally, as the decision had been taken to leave a pillar between the southeast pit and main pit to create a water reservoir (see Figure 15.4 previously), this material was sterilized for purposes of the evaluation. All other evaluation parameters were unchanged from those listed in Sections 15.2.3 and 15.2.4 of this report.

The various nested pit shells generated with the second pass LG run were similar to those from the first pass (after accounting for the saddle that was sterilized). The shell best corresponding to the engineered design was generated using a Revenue Factor of 62% (US$5.58/lb Ni price).

Table 15-6 on the following page compares key parameters for the Engineered Design to output of the Second Pass LG. For simplicity, LG output has been grouped in stages of 10 nested 1-percent increment shells. Additional granularity is provided for the range between a 60% and 70% Revenue Factor.

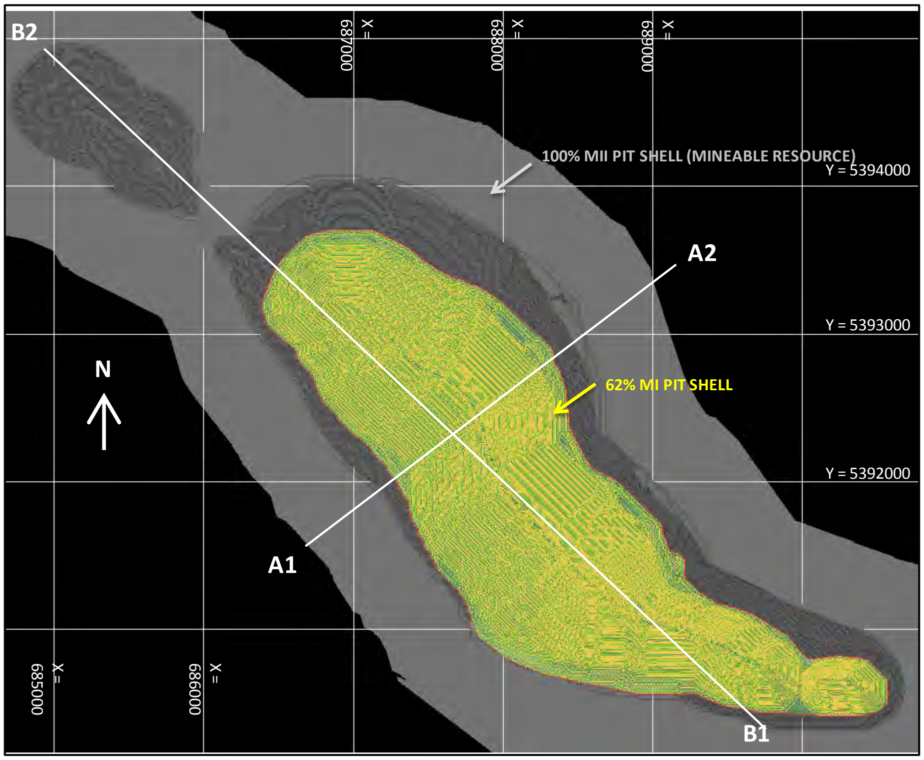
It can be seen that the nested shell generated with a 62% Revenue Factor has been selected as being closest to the Engineered Design. Using the LG parameters, the incremental profit for this Revenue Factor is $7.32/t ore (compared to the preceding 61% Revenue Factor shell, the 62% Revenue Factor shell contains 41.3 Mt ore that generate $790 million in NSR with associated costs of $209 million for mining and $280 million for processing and G&A). The selected shell is thus considered to be a conservative estimate of the economic limits for Dumont reserve inventory.

Table 15-6: Comparison of Engineered Design & Second Pass LG Output

LG Stage	Revenue Factor	LG Shell	Mill Feed (Mt)	Waste Rock (Mt)	Stripping Ratio	Grade Ni	% Contained Ni (Mlbs)	Avg. Recovery	NSR $/t
Engineered Design			**1,177**	**1,338**	**1.14**	**0.27**	**6,964**	**43.0%**	**$21.36**
1	30%	8	42	26	0.62	0.34	313	49.9%	$31.47
2	40%	18	255	146	0.57	0.29	1,646	46.4%	$25.48
3	50%	28	710	599	0.84	0.28	4,334	44.5%	$23.17
4	60%	38	1,087	1,187	1.09	0.27	6,515	43.8%	$22.41
4a	61%	39	1,131	1,243	1.10	0.27	6,759	43.6%	$22.25
4b	**62%**	**40**	**1,172**	**1,309**	**1.12**	**0.27**	**6,996**	**43.4%**	**$22.14**
4c	63%	41	1,187	1,339	1.13	0.27	7,078	43.4%	$22.12
4d	64%	42	1,205	1,373	1.14	0.27	7,183	43.4%	$22.08
4e	65%	43	1,237	1,439	1.16	0.27	7,367	43.3%	$22.02
4f	66%	44	1,245	1,459	1.17	0.27	7,413	43.3%	$22.01
4g	67%	45	1,267	1,505	1.19	0.27	7,545	43.2%	$21.96
4h	68%	46	1,282	1,533	1.20	0.27	7,629	43.1%	$21.92
4i	69%	47	1,299	1,574	1.21	0.27	7,724	43.1%	$21.89
5	70%	48	1,315	1,617	1.23	0.27	7,814	43.1%	$21.88
6	80%	58	1,413	1,914	1.35	0.27	8,376	42.9%	$21.73
7	90%	68	1,468	2,113	1.44	0.27	8,681	42.8%	$21.63
8	100%	78	1,492	2,236	1.50	0.27	8,819	42.8%	$21.60
Selected as Closest to Engineered Design									

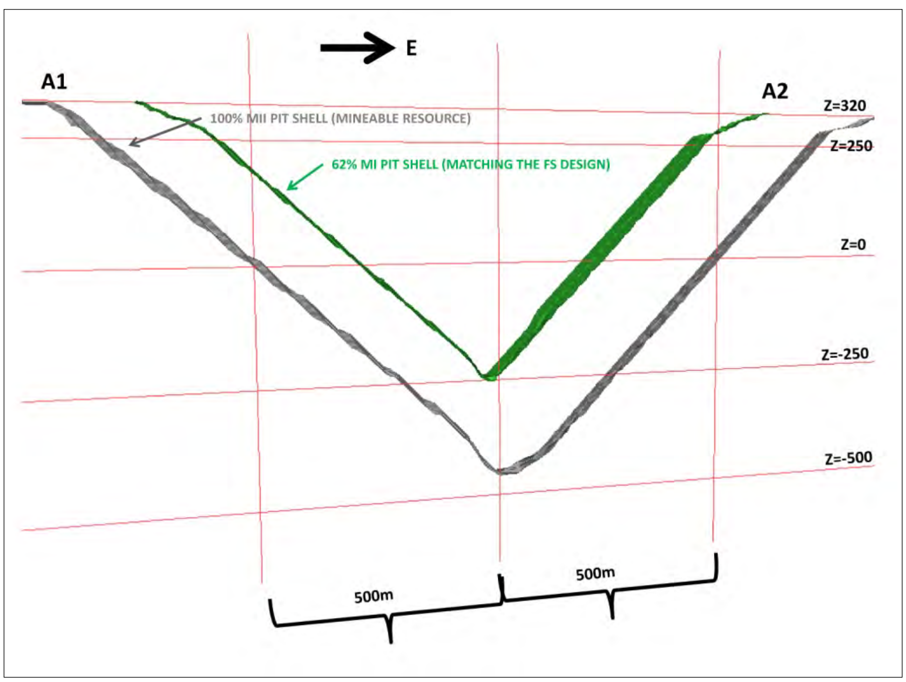
Plan and section views of the 62% Revenue Factor LG shell are given on the following page in Figures 15.5 to 15.7. These figures also includes a view of the 100% Revenue Factor shell generated using Measured, Indicated and Inferred (MII) Resources that was used to constrain the resource estimate.

Figure 15.5: Second Pass LG – Shell Corresponding to Engineered Design (Plan View)



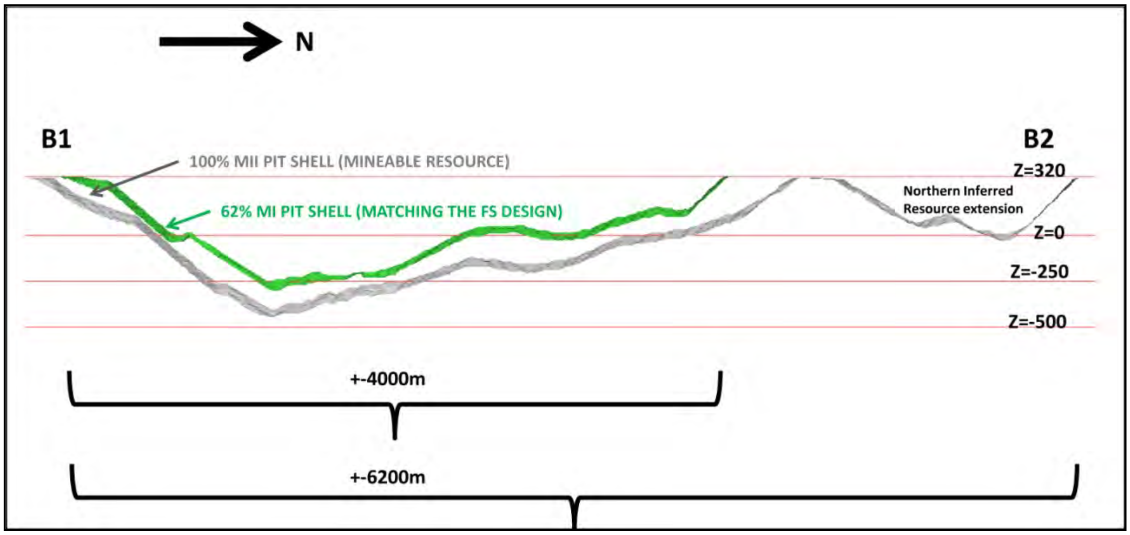
Source: RNC. **Note**: the scale is given in metres based on the local UTM coordinate grid (NAD83 datum, Zone 17).

Figure 15.6: Second Pass LG – Shell Corresponding to Engineered Design (X-Section)



Source: RNC.

Figure 15.7: Second Pass LG – Shell Corresponding to Engineered Design (Long-Section)



Source: RNC.

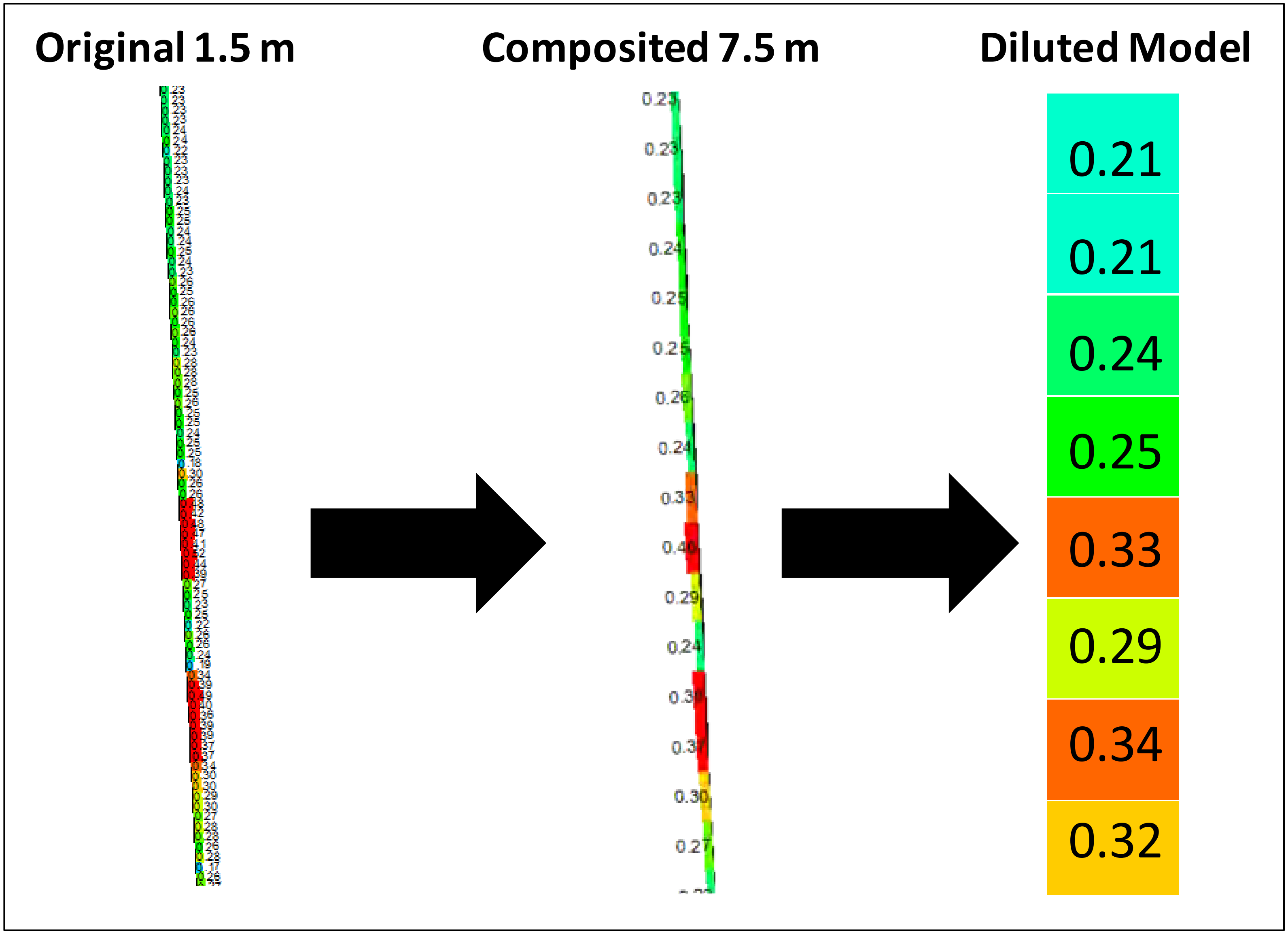
15.2.7 Dilution & Mining Losses

The drill hole sample compositing and block grade interpolation process used to construct the deposit block model is believed to incorporate sufficient dilution and hence, no additional planned dilution factors were applied. As shown in Figure 15.8, the original 1.5 m samples in the area of the arrow all grade between 0.39 to 0.52% Ni. Compositing these on 7.5 m intervals yields adjacent composites of 0.46 and 0.33% Ni, but the resulting block value generated through interpolation is only 0.33% Ni.

Unplanned dilution and mining losses (defined as dilution and mining recovery factors not already inherent within the block model) will potentially occur at the contacts between ore and waste.

Dumont mineralization is disseminated with large continuous ore zones between hanging wall and footwall waste rock contacts. There is no sharp contact between ore and waste at the hanging wall and footwall contacts. Rather, higher value mineralization hosted in dunite is bracketed by lower value mineralization hosted in peridotite (the engineered design includes 21.2 Mt of peridotite-hosted ore). This gradational nature of mineralization, coupled with the methodology for interpolating grades into 20m x 20m x 15m blocks (equal to the smallest mining unit for the shovels and excavators that will be used) was considered to have accounted for any possible dilution at the hanging wall and footwall contacts. No extra unplanned dilution or mining losses was added for these contacts. Similarly, internal blocks of waste within the ore zones do not display sharp contacts and the block model selectivity is considered achievable in mining. This selectivity basis is not only pertinent to waste mining, it also pertains to selectivity of grade bins for dispatching ore to stockpiles versus the ROM feed as represented in the LOM schedule.

Figure 15.8: Planned Dilution Accounted for In Block Model (from Hole 09-RN-219)



Source: RNC.

However, there is a sharp delineation of ore at the overlying sand and gravel contact. Moreover, the ore – sand and gravel interface can be an undulating uneven surface, leading to some unavoidable mixing of the two products during mining. Accordingly, within the contact horizon of 10m (5m ore and 5m sand and gravel) it was assumed that 2m of sand and gravel would be unavoidably mixed with ore and 1m of ore would be unavoidably lost with waste. Losses and dilution were calculated on a block-by-block basis taking account of the differences in SG (2.57 average for ore, 1.94 average for sand and gravel) and were scheduled as the affected blocks would be mined. The resulting impact is:

- Dilution is expected to average 0.49% of total ore tonnage over the life of mine, with a peak of 8.2% during the initial year of mill operation (when a significant tonnage of ore is located at the sand and gravel interface). Diluting material would carry no grade.
- Mining losses are expected to average 0.28% of total ore tonnage over the life of mine, with a peak of 5.2% that also occurs in the initial year of mil operation.

15.2.8 Cut-Off Grade

Cut-off values used for mine planning will be based on the NSR value of material, as determined using the grade, recovery and price of all economic metals (including Ni, Co, Pt and Pd). As is normal for open pit designs, the calculation of cut-off values ignores mining costs and includes only the following marginal costs:

- any incremental haulage costs from the pit rim that would be incurred for re-handling low-grade ore (as material at the marginal cut-off grade would initially be stockpiled);
- milling costs, including sustaining capital that would be effectively expended on a per-tonne basis (e.g., lifts of the TSF or annual maintenance of the mill); and
- G&A.

It has been reported previously in Section 15.2.4 that these marginal costs total $7.00/tonne. This value is an average and will vary as a function of the cut-off grade, as a lower cut-off will lead to a larger stockpile and associated higher hauling costs. Additionally, this value includes some sustaining capital costs, which are treated as unit costs (equally applied to all tonnage) when in reality sustaining costs for fleet and the TSF are incurred as step functions (no expenditure until a given threshold, then a large payment for a replacement truck or additional lift of the TSF) The theoretical calculation was tested iteratively, by adjusting the cut-off value upwards and rescheduling the mine plan. NPV was maximized with the $7/t cut-off, thus proving the validity of this value as a cut-off.

As a separate exercise, the associated cut-off grade in % Ni was estimated for purposes of defining potentially economic resources to be included in reserves. As 96% of the NSR is generated from Ni, this cut-off grade calculation was based solely on Ni. Table 15-7 summarizes this calculation, showing that the theoretical cut-off is 0.09% Ni.

A key variable in the calculation shown in Table 15-7 is metallurgical recovery. To account for the possibility that lower grade mineralization may exhibit lower recovery, the theoretical cut-off of 0.09% Ni has been increased by approximately 60% to the 0.15% Ni that is used in the resource estimate. It should be noted that the resource and reserve statement is based on this minimum 0.15% Ni grade – so blocks with an NSR value > $7/t but grading less 0.15% Ni have been excluded from the estimate and schedule. The tonnage of material within this category is small (approximately 7 kt) and therefore not material.

Table 15-7: Cut-Off Grade Calculation

Parameter	Units	Value
Ni	US$/lb	$9.00
TC/RC	US$/lb	$1.27
NSR	**US$/lb**	**$7.73**
F/X	C$=	$0.90
Net Smelter Return	$/lb	$8.59
Mine Re-Handle	$/t	$1.00
Mill Cost	$/t	$4.80
G&A Cost	$/t	$0.55
TSF	$/t	$0.30
Sustaining Capital	$/t	$0.35
Subtotal	**$/t**	**$7.00**
Payable Ni	lb/t ore	0.82
Payables	lb/t ore	93%
Recovered Ni	lb/t ore	0.88
Average Recovery		43%
Head Grade	**%Ni**	**0.09**

15.2.9 Reserve Classification

Measured resources that would be treated as ROM ore have been classified as Proven Reserves, while Measured Resources that would be initially stockpiled and all Indicated Resources have been classified as Probable Reserves.

The cut-off value used to define the material that would be treated as ROM ore will vary by year, as a function of the total tonnage and associated value of ore mined in a given year. The lowest value used as the ROM cut-off will be $15/t in Year 19, when a total of 41.8 Mt ex-pit ore is planned to be mined, or approximately 9% more than the mill capacity of 38.3 Mt/a. The highest value used as the ROM cut-off of $27/t will be used the following year, when waste stripping is complete and a total of 106.5 Mt ex-pit ore is planned to be mined, or 178% more than the mill capacity. The $27/t cut-off is also planned for Year 4 when the stockpile is being expanded in preparation for the transition from a single mill line at 52.5 kt/d to two lines totalling 105 kt/d. Ore mining of 58.6 Mt that year represents 206% more than the single line capacity. Over the 20 year life of the pit, a $17/t cut-off is applied the most years (eight in total), followed by $19/t (six years).

This variable cut-off is accounted in Table 15-8, which illustrates the conversion of resources to reserves. The open pit mine reserves declared in Table 15-8 have been carried forward into the LOM plan and production schedule as described in Section 16. As well, the project costs and economic analysis have been determined from this reserve and resulting LOM schedule. The results of LOM plan and financial analysis confirm that the appropriate parameters have been used in the LG runs (see Sections 21 and 22).

Table 15-8: Conversion of Resources to Reserves

Material Within	Measured Resources					Indicated Resources					Total				
Engineered Pit Shell	kt	% Ni	Ni M lbs	Recovery	NSR	kt	% Ni	Ni M lbs	Recovery	NSR	kt	% Ni	Ni M lbs	Recovery	NSR
Resource	354,000	0.28	2,216	43%	$22.84	822,600	0.26	4,749	43%	$20.59	1,176,600	0.27	6,964	43%	$21.26
ROM	179,400	0.32	1,277	50%	$30.12						179,400	0.32	1,277	50%	$30.12
Stockpile	174,600	0.24	939	34%	$15.35						174,600	0.24	939	34%	$15.35
Mining Losses	700	0.28	4	45%	$23.79	3,100	0.26	18	43%	$20.59	3,800	0.27	22	43%	$21.18
ROM	400	0.32	3	50%	$30.12						400	0.32	3	50%	$30.12
Stockpile	300	0.24	2	34%	$15.35						300	0.24	2	34%	$15.35
Dilution	1,100	0.00	0	0%	$0.00	4,700	0.00	0	0%	$0.00	5,800	0.00	0	0%	$0.00
ROM	600	0.00	0	0%	$0.00						600	0.00	0	0%	$0.00
Stockpile	500	0.00	0	0%	$0.00						500	0.00	0	0%	$0.00
Proven Reserves	179,600	0.32	1,275	50%	$30.03						179,600	0.32	1,275	50%	$30.03
Probable Reserves	174,800	0.24	938	34%	$15.31	824,200	0.26	4,731	43%	$20.47	999,000	0.26	5,668	41%	$19.57
Total Reserves	**354,400**	**0.28**	**2,212**	**43%**	**$22.77**	**824,200**	**0.26**	**4,731**	**43%**	**$20.47**	**1,178,600**	**0.27**	**6,943**	**43%**	**$21.16**

16 MINING METHODS

16.1 Hydrology & Hydrogeology

16.1.1 Hydrology

The proposed mine development will be largely confined to an unnamed stream tributary that drains out along the left-bank of the Villemontel River. For the purpose of the FS, this tributary has been designated Unnamed Creek. At its confluence with the Villemontel River, Unnamed Creek has a total drainage area of 52.3 km^2. The drainage catchment of this stream shares its northern and eastern boundaries with the divide between the Hudson Bay and St. Lawrence River watersheds. One of the key constraints on mine development will be preventing the transfer of surface water flow from the Unnamed Creek catchment to the Hudson Bay watershed.

Unnamed Creek has two main tributaries that flow in a southerly direction, each draining areas of similar size. These tributaries have been unofficially named West Creek and East Creek, consistent with the side of the Unnamed Creek catchment that each drains. The confluence of these two streams occurs over the ore deposit, some 2.5 km upstream of the mouth of Unnamed Creek.

A surface water management system will be constructed to direct the flows in West Creek, and East Creek around the open pit. Further information regarding the surface water management system is provided in Section 18.

16.1.2 Hydrogeology

Characterization of the Dumont hydrogeology was built on the work that was undertaken for the pre-feasibility study, and took into account the revised mine layout.

Groundwater level monitoring data was taken from a total of 55 wells across the concession (42 in overburden and 13 in shallow bedrock). Additional bedrock hydraulic testing was completed, bringing the total to 57 packer tests in 20 drill holes, as well as two long-term (>36-hour) injection tests. Overburden testing included a slug-testing program of 13 tests as well as a 41-hour pump test in the sand and gravel horizon at the west side of the proposed open pit.

A geographical information system (GIS) database was developed to compile and present the data collected. Surfaces were created for the dominant (overburden and bedrock) hydrogeological domains on a concession wide scale.

The surfaces generated in the GIS database were used to construct a 3D groundwater model for the project. Hydraulic parameters, taken from the field testing results, were assigned to the model domains. The output from model produced the following:

- groundwater inflow to the pit (site water balance input) was estimated to range between 3,500 and 5,400 m^3/d during mine operations; and
- boundary conditions and calibration data were used in pit slope pore pressure modelling for the geotechnical program.

Pore pressure modelling of the pit slopes indicated that active depressurization is not expected to be necessary, although further pore pressure monitoring is recommended along the periphery of the pit to ensure pressures do not exceed levels generated by the model.

16.2 Geotechnical Design Criteria

The geotechnical characteristics of rock types that will be encountered in the Dumont pit have been determined by the following drilling/data collection campaigns:

- dedicated geotechnical holes drilled during the Preliminary Assessment Study (three 500 m holes);
- dedicated geotechnical holes drilled during the Pre-Feasibility Study (ten 500 m holes);
- dedicated geotechnical holes drilled during the Feasibility Study (eleven 500 m holes); and
- geotechnical logging of resource holes drilled during the pre-feasibility and feasibility studies.

16.2.1 Geotechnical Model

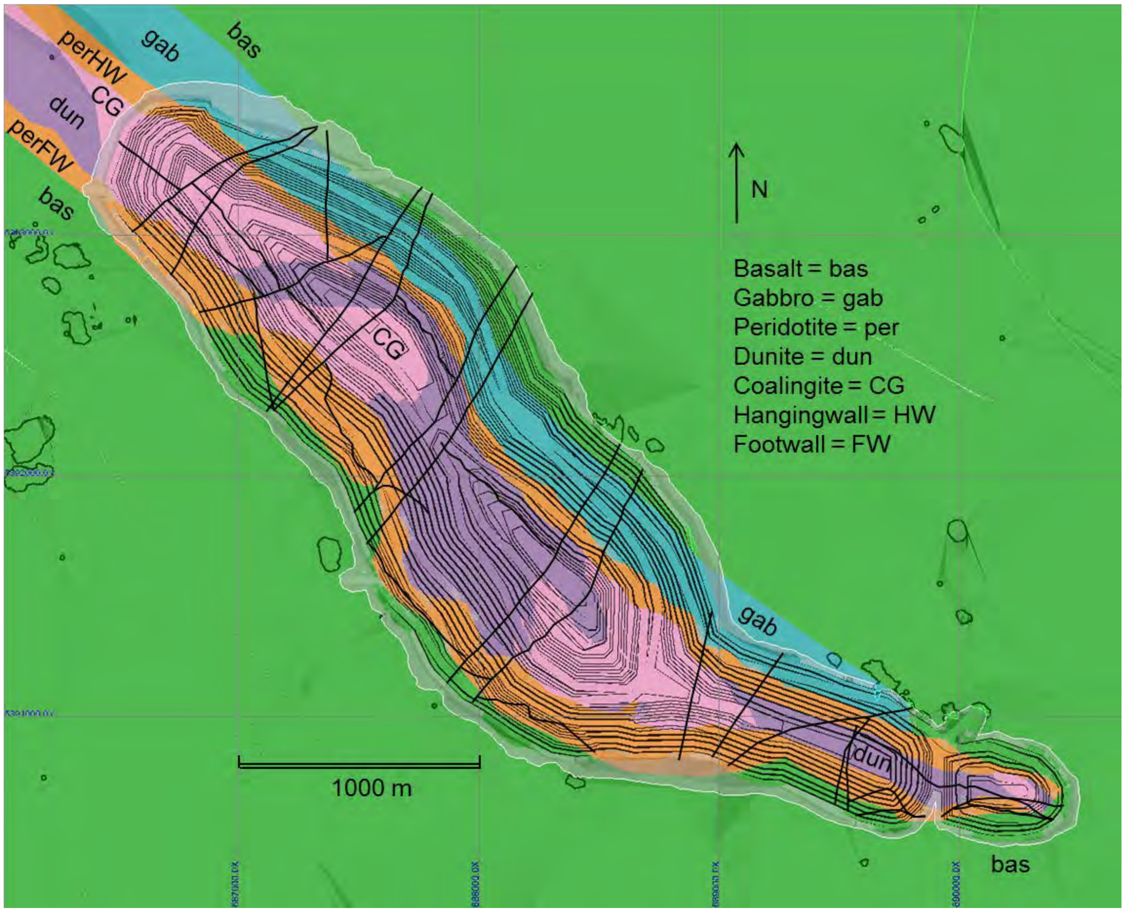
Partial geotechnical data exist for approximately 342 drill holes. Of these, 51 drill holes have been logged using oriented core for more detailed geotechnical parameters such as joint condition and orientation. The geotechnical model has been built using the geology (and alteration) wireframes, interpreted fault-network, and rock mass specific parameters.

Three detailed structural geology studies have been undertaken on the Dumont deposit area: one in 2010, another in 2011, and a feasibility level interpretation and consolidation of knowledge in 2012. The deposit-scale structural geometries were modelled using regional geophysical data interpretation tied to drill hole data from RNC's geological database.

Through this drilling, logging, and mapping, a consistent package of rock types has been identified. From hanging wall to footwall (as illustrated in Figure 16.1), they comprise the following:

- basalt (BasHW);
- gabbro (gab);
- peridotite (perHW);
- dunite (the host to mineralization, which includes dun and dun-CG);
- peridotite (perFW); and
- basalt (basFW).

Figure 16.1: Plan View of the Rock Types & Major Structures that may be Exposed in the Proposed Dumont Pit (Hanging wall is the Northeast Side of the Pit Shell)



Source: SRK.

Using the geological model as a framework, an analysis of geotechnical data was undertaken for the rock mass at the Dumont deposit. The assessed parameters include rock quality designation (RQD), fracture frequency per meter (FF/m), empirical field estimates of intact rock strength (IRS), field (point load) and laboratory (uniaxial compressive, triaxial, joint shear) strength, and RMR_{89} (Bieniawski, 1989). Representative geotechnical parameters for each of the four main rock types are given in Table 16-1, while a typical cross-section is given in Figure 16.2.

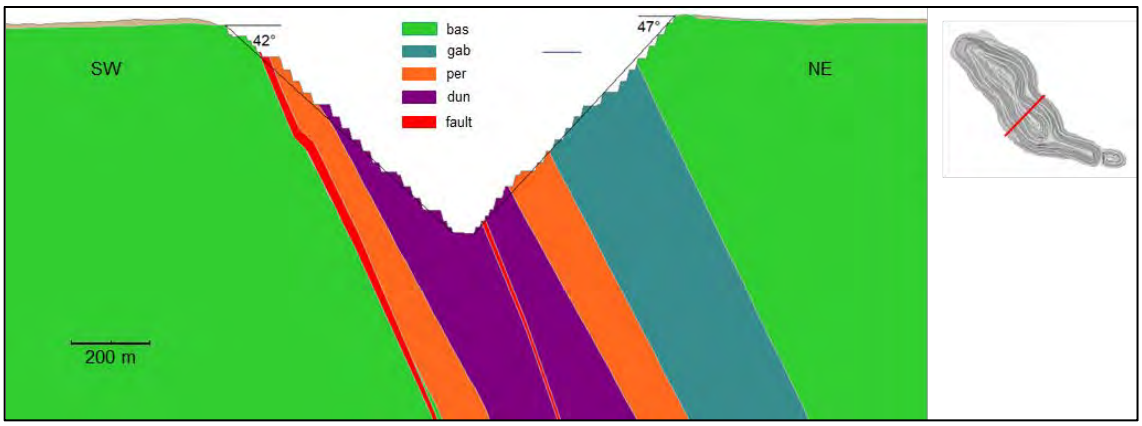
The structural investigation found three distinct structural domains for the Dumont open pit area. These domains are bounded by major structures and exhibit minor to moderate differences in joint and foliation data. The northeast-to-southwest-trending faults are steeply dipping towards the southeast. Damage-zones associated with faults oriented parallel and sub-parallel with the sill do occur. These sill-parallel faults are restricted to the basal-contact, footwall peridotites and the dunites – occurring throughout the strike-length of the pit, dipping towards the hanging wall.

The combined litho-structural and alteration model was used to construct the geotechnical domains, for which a typical cross-section is given in Figure 16.2.

Table 16-1: Representative Geotechnical Characteristics of Dumont Rock Types

Material	SG (t/m^3)	UCS (MPa)	Fracture Frequency (ff/m)	Rock Mass Rating
Basalt	2.9	130	1.8	75
Dunite	2.6	90	3.3	70
Gabbro	3.0	150	1.2	75
Peridotite	2.7	110	3.8	65

Figure 16.2: Typical Southwest to Northeast Cross-section through Dumont Pit (pit depth is approx. 500 m)



Source: SRK.

16.2.2 Rock Slope Design

Reviews of the site geology, structural geology findings, geotechnical evaluation and resource targets indicates in this relatively strong rock mass, that the dominant controls on pit stability are likely to be kinematic. The failure modes will most probably be planar sliding on the footwall at a bench and inter-ramp scale, minor bench-scale wedges throughout the pit, with a low probability of toppling on the hanging wall. For fault damage-zones and exposed (and activated) dun-CG domain rocks, unravelling on a bench-scale may occur.

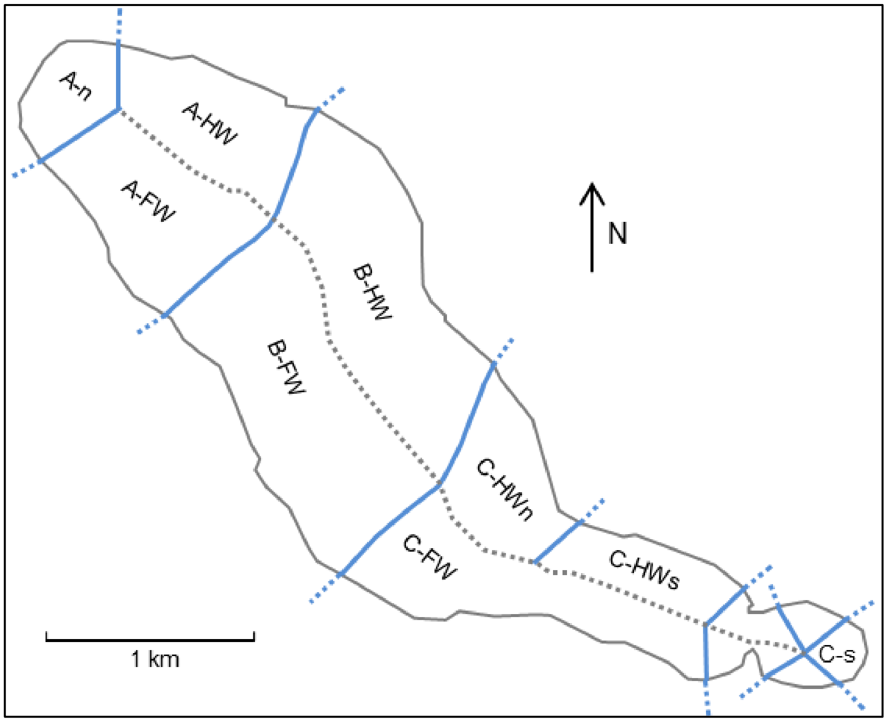
Considering the geotechnical domains and the likely slope directions, slope design sectors where generated and design parameters developed for each design sector. The design parameters were based on a maximum stack height of 120 m, separated by a geotechnical safety berm.

These design parameters included the following:

- single or double benching;
- bench width;
- bench face angle; and
- inter-ramp angle.

For each slope design sector, an overall slope angle was determined based on the combination of the above listed parameters, the geotechnical berm/ramp width and the stack height. This inter-ramp angle was used in the design (Table 16-2). The pit phases and annual shells were checked for interactions with major structures and geology, and no significant unfavourable conditions were found that cannot be operationally managed.

Figure 16.3: Dumont Pit Design Sectors



Source: SRK.

Table 16-2: Dumont Pit Design Guidelines by Sector

RNC Dumont Feasibility Study Slope Design Guidelines				
RNC Dumont FS Pit	Bench (m)			
Design Domain / Sector (Face Dip-Dir°)	Height	Width	Bench Face Angle (°)	Inter-Ramp Angle (°)
A-n (130), A-HW (200), C-HWn (250), C-s (270), and C-s' (090)	30	10.5	75	58
	15	7.5		52
B-HW (240) and C-HWs (200)	30	14.5	75	53
	15	9.5		48
A-FW (050) and C-FW (010)	30	10.5	70	54
	15	7.5		49
B-FW (060)	30	10.5	65	51
	15	7.5		46

Note that a) the maximum stack height allowed is 120 m, b) geotech berm-width is 20 m, c) single-benching is 15 m high, d) use single-benching one below and three above for faults oriented within ±015° to the bench-crest azimuth, e) single-benching for dun-CG domain, and f) double-bench only if pre-split blasting used and only in non-CG and non-faulted ground.

The footwall design has the bench-faces pre-split on finals to match the sill-parallel foliation, which will result in inter-ramp slopes which are parallel to the basal contact of the sill. Where slopes are built within, or in close proximity to the basal-fault damage zones, the benches and inter-ramp slopes may break back towards the fault-zone. In instances where this occurs, remediation and-or operational design adjustments of the affected slopes may need to be implemented.

For some of the hanging wall slopes, rock block toppling may be experienced. Allowance has been made for the possibility of this toppling in the design and it may be an area for improved slope angles once trial slopes are established and the rock mass performance in excavation is determined.

16.2.3 Recommendations

There is some potential for more faults to occur within the volume than have already been interpreted, and further work (drilling and geological/geotechnical mapping) is required to satisfactorily understand the structural geology of some portions of the deposit area.

During construction, the slopes established in the southeast pit should be used as an opportunity to investigate the behavior of the rock mass (in terms of failure mechanisms and mode) for each of the domains represented in the slopes. In particular, it will enable the footwall fault damage-zones' performance (with possible remediation measures) to be analysed prior to the northwestward pit advance.

Determine which slopes are likely to experience some level of instability due to elevated pore pressures during freshet, and assess the schedule impact if access needs to be restricted at these times of year.

16.2.4 Soil Geotechnical

The geotechnical characteristics of soils that will be encountered within the pit area have been determined primarily on the basis of field programs completed during Q1 2011 and Q1 2012. Based on these field programs, the following soil types, listed in descending stratigraphic order, were identified:

- Organic soil, which consists of a very weak organic mat and/or peat. This layer blankets much of the project area and extends to depths ranging from 0.5 to 4.0 m.
- Clay, which is generally found below the organic soil and typically ranges in thickness from 2 to 15 m. Two types of clay were encountered: a firm to stiff brown clay and a soft to very soft grey clay. The brown clay overlies the grey clay in areas where both are present.
- Silt, which is accompanied by a variable distribution of gravel, sand and clay, ranges in thickness between 1 and 16 m but is usually around 5 m thick. The silt ranges from soft to stiff.
- Sand and gravel, which are generally dense to very dense and range in thickness from 1 to 40 m.

Not all soil types are present in all areas of the pit.

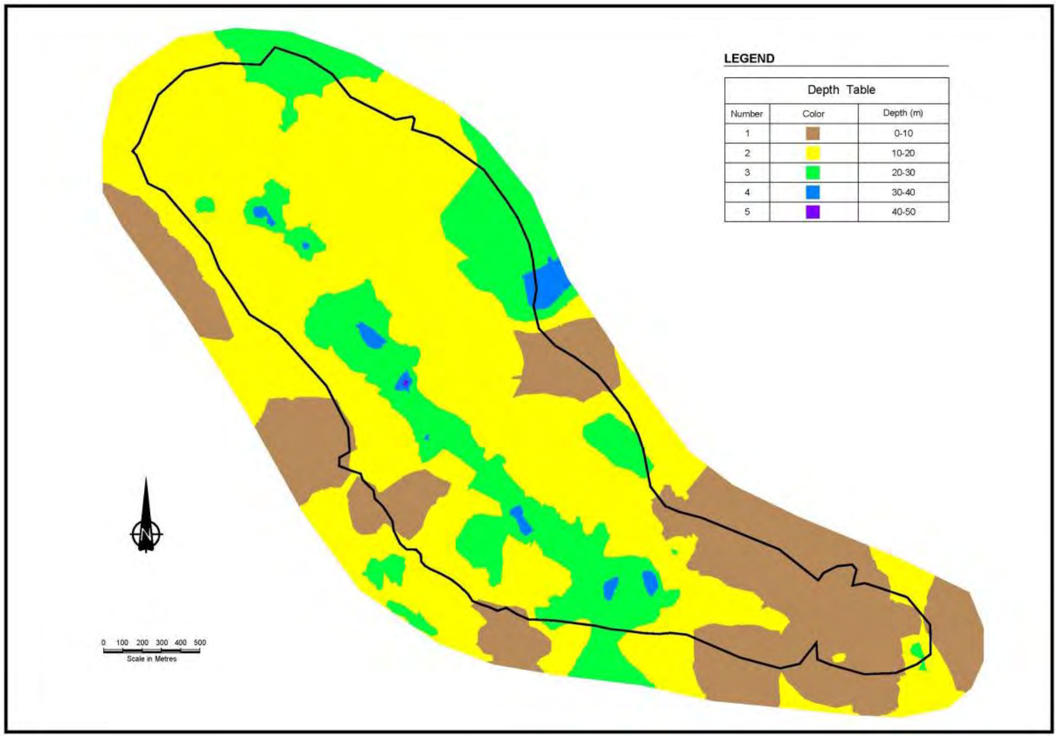
16.2.4.1 Database

The geotechnical database for the soils in the vicinity of the open pit is comprised mainly of 43 sonic drill holes, most of which extended to bedrock and were complemented by laboratory tests on selected samples from the sonic drill program. An additional 53 CPT probes, performed to refusal (typically on dense granular soils), complete the data base in the open pit.

16.2.4.2 General Stratigraphy & Geotechnical Conditions

The overburden (soil) thickness in the vicinity of the open pit is illustrated as a series of coloured isopachs on Figure 16.4. Glacier movements have scoured a depression in the bedrock that coincides generally with the northwest-southeast orientation of the ore body. The thickness of the overburden approaches its maximum, close to 50 m, in the central portion of the pit. Conversely, the overburden is generally the thinnest along the sides of the proposed open pit.

Figure 16.4: Isopachs of Overburden Thickness

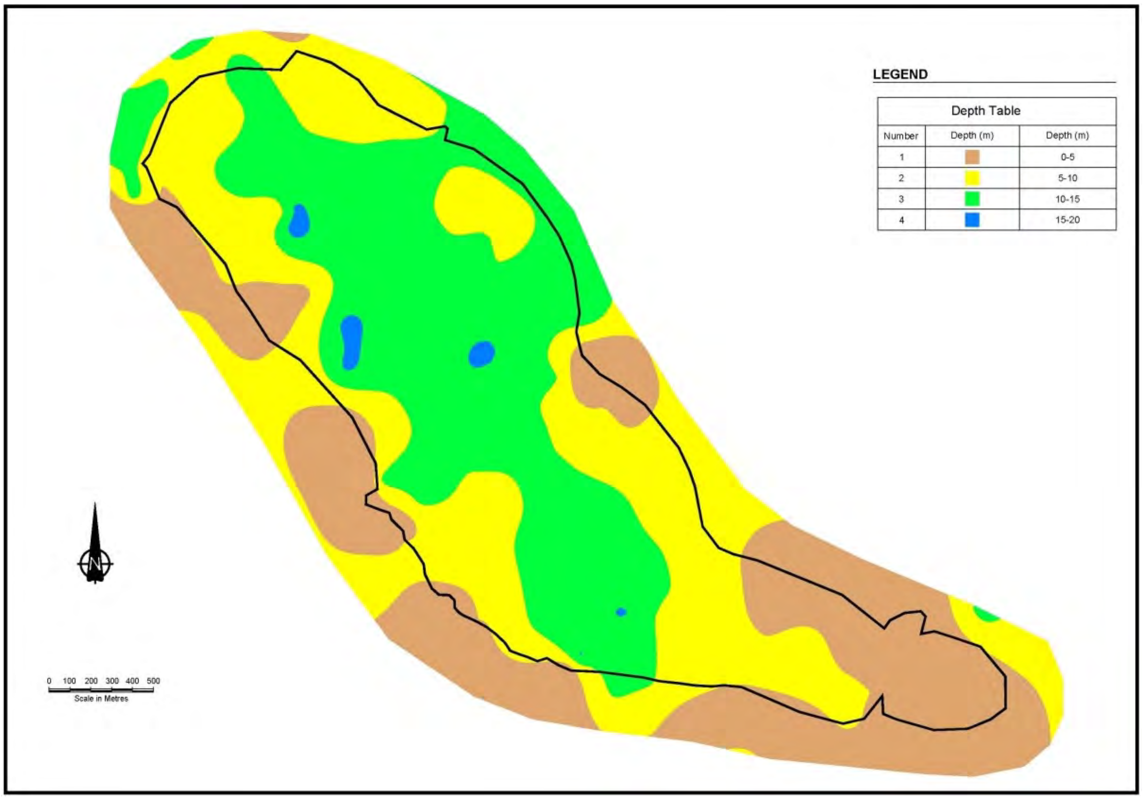


Source: SRK.

In general, where the overburden is less than 6 to 8 m in thickness, the soil profile consists of a thin layer of organic soil overlying a layered sequence of relatively stiff clay and silt, overlying dense gravelly sand or bedrock. However, where the overburden is greater than 6 to 8 m in thickness, the soil profile typically consists of a thin layer of organic soil overlying a 1 to 2 m thick layer of light brown, moist, firm to stiff clay overlying a layer of grey, wet to saturated, very soft to firm clay of variable thickness. A relatively thin layer of soft silt usually underlies the grey clay and a dense gravelly sand underlies the silt, or the clay where the silt is absent.

The combined thickness of the organic soils, clay and soft silt deposits in the vicinity of the open pit is illustrated as a series of coloured isopachs on Figure 16.5. The thickness of these deposits is typically 2 to 10 m over most of the pit area, but is greater than 15 m in a few locations.

Figure 16.5: Isopachs of Organic & Fine-grained Soil Thickness



Source: SRK.

The grey clay, due to its low undrained shear strength, is the weakest unit within the overburden materials. Table 16-3 summarizes the average geotechnical properties of the grey clay based on laboratory testing. The CPT data compares well with the laboratory test results, but also confirms the undrained shear strength of the grey clay varies over the footprint of the proposed open pit. This variation is summarized in Figure 16.6, which shows three zones of grey clay based on the undrained strength results from the CPT probes.

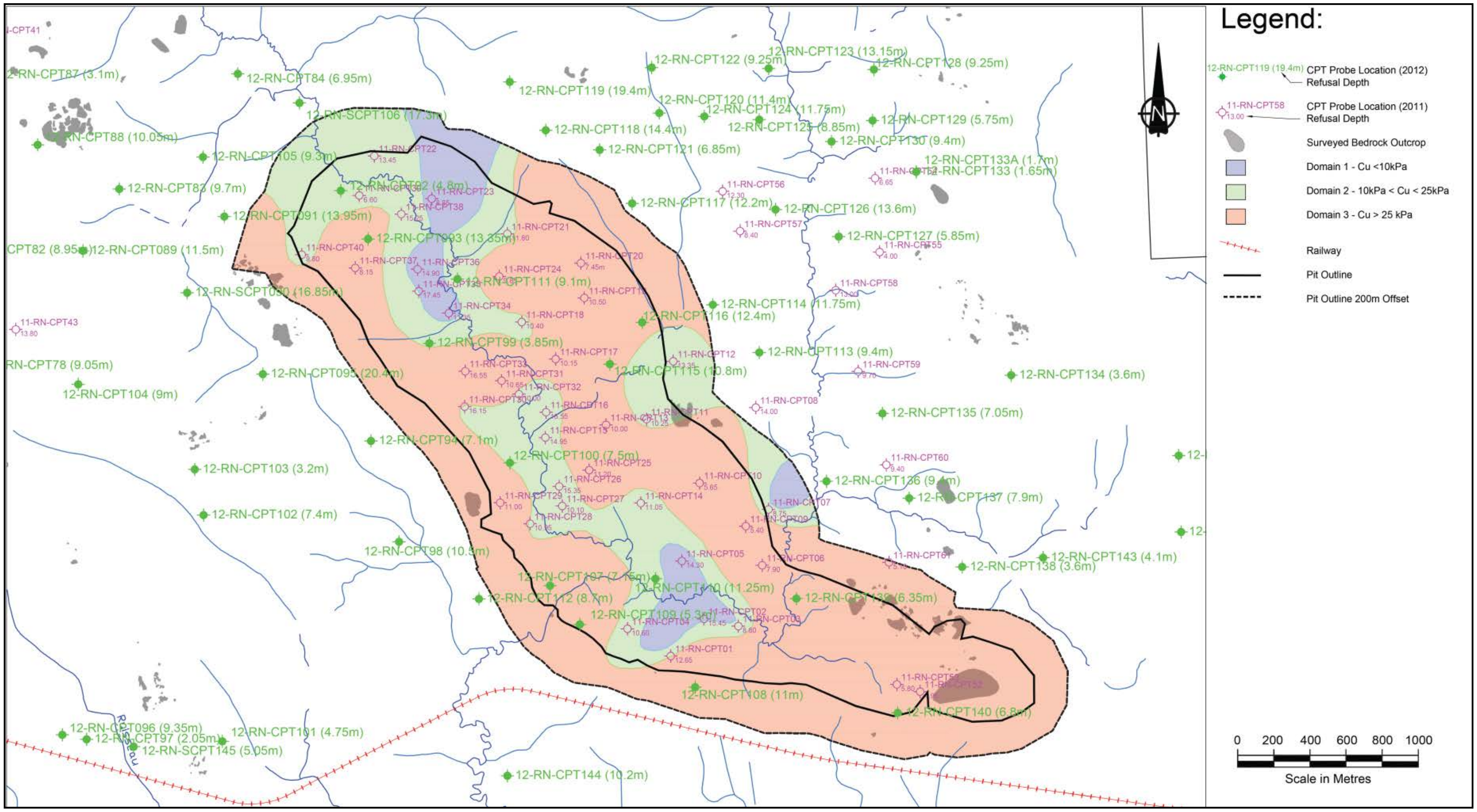
Table 16-3: Average Properties of the Grey, Wet Clay

USCS Classification	W [%]	w_L [%]	w_P [%]	k [m/s]	e_0 [-]	C_c [-]	σ_p [kPa]	c_u [kPa]
CH	92	70	27	3.8E-09	2.5	2.7	40	20

Note: w_L: Liquid limit, w_P: Plastic limit, w_L: Moisture content, k: Hydraulic conductivity, e_0: In-situ void ratio, C_c: Compression index, C_v: Coefficient of consolidation, σ_p: Pre-consolidation pressure, c_u: Undrained shear strength.

Ausenco

Figure 16.6: Overburden Domains Based on the Undrained Strength of the Grey Clay



Source: SRK.

16.2.4.3 Trafficability

The expected trafficability of the various overburden soils is summarized below:

- The organic soils, clays and soft silts will not support normal mining equipment unless a waste rock-bearing layer at least 1 to 2 m thick is placed over them.
- The relatively stiff silt will typically require a layer of waste rock to provide trafficability, particularly if this material becomes wet due to precipitation or runoff. The thickness of the waste rock layer will depend on factors such as the moisture content and undrained strength of the soil, as well as the equipment size.
- The sand and gravel materials are generally dense to very dense and will afford reasonable trafficability for mining equipment, except where localized layers or lenses of silt or clay may be present within the sand and gravel materials.

16.2.4.4 Slope Design

As noted previously, the low undrained shear strength (c_u) of the grey clay is the key to the design of the overburden slopes. Three generalized overburden domains for slope stability analysis were established within the open pit area based on the undrained strength characteristics of the clay (Figure 16.6 above).

Stability analyses under static and seismic loads were undertaken on simplified cross-sections through the overburden domains that were intended to address the typical range in soil stratigraphy. This produced a range of results that varied depending on the stratigraphy, the undrained strength of the fine-grained soil (i.e., the clay and/or silt) and the effective stress parameters for the coarse-grained soil (i.e., the sand and gravel materials). Based on these results, Table 16-4 presents the design of the open pit slopes for the overburden.

Table 16-4: Open Pit Soil Slope Design Recommendations.

Domain	Clay Slope	Other Stratigraphies (Sandy Silt, Sand & Gravel)
Domain 1 Thick Clays (C_u < 10 kPa)	Complete removal of the clay material or 8H:1V	2.5H:1V
Domain 2 Moderately thick Clays (10 kPa < C_u < 25 kPa)	5H: 1V	2.5H:1V
Domain 3 Mainly Sands and Silts (C_u > 25 kPa)	4H:1V	2.5H:1V

These recommended slope angles have been used as the basis of the design slopes adopted in the feasibility study mine plan.

16.3 Open Pit Mine Plan

16.3.1 Introduction

The Dumont pit measures approximately 4.9 km along strike, 1.4 km at the widest point and reaches a maximum depth of 560 m. A total of 2,514 Mt of material will be excavated, using large surface mining equipment that will operate at high production rates. Many of the mining concepts resemble practices currently used at large open pit copper, iron ore and coal mines.

The low-grade of the ore implies that the mining operation must achieve high productivity and efficiency.

As described in section 16.2.4 above, the ore body is covered with overburden of varying depth. Overburden, which represents 7% of the total material that will be excavated, is comprised of differing soil types. Overburden will be stripped in advance of the ore mining operation and impounded in different areas depending on the geotechnical parameters of the specific soil type. Waste rock, which represents 46% of the total material excavated, will mainly be stored in a single large dump, with a portion of the waste rock used for construction of various infrastructure including roads and the tailings storage facility (TSF). Ore, representing 47% of the total tonnage excavated, will be fed to the mill, either directly as run-of-mine (ROM) feed or after being temporarily impounded in a low-grade stockpile. Tailings from the treatment of ore will be impounded in the TSF while the pit is operational then in the depleted pit shell in later years when mill feed is sourced entirely from the stockpiles.

The mine will have de-watering systems and an electrical supply system for the electrified mining equipment. Unit operations will consist of drilling, blasting, loading and hauling. Key criteria used in the design of the open pit reflect the size of equipment that will be used and include:

- bench height of 15 m;
- ramp gradient of 10%;
- pushbacks have been designed using a target minimum mining width of 100 m, with the absolute minimum in isolated areas being 60 m; and
- All final walls will be pre-split.

The Dumont pit and mine plan was developed with standard mine planning practices following the steps of:

- LG optimization;
- shell selection and initial schedule;
- pit phase design; and
- final mine scheduling.

These steps are described in the following sections.

16.3.2 LG Optimization

The LG Optimization has been described in some detail in Section 15.2.4 previously. This work is summarized in the following paragraphs.

The LG optimization was initiated by calculating the net value of each block in the model by subtracting estimated costs for mining, processing and administration from the NSR of each block. These estimated costs were based on the forecast cost structure at the full mill production rate of 105 kt/d.

Overall slope angles were assigned to the various sectors based on the recommendations provided in the Section 16.2, along with the ramp geometry from the earlier Revised PFS and 35m ramps that will be used for 230 t class haul trucks. These slopes ranged from 42° to 50°.

The LG algorithm then selected a ‘cone’ of ore and associated waste stripping that maximises NPV. By varying the metal price, it was possible to generate higher value nested cones that can be used to identify the optimal development sequence. Nested shells were generated for each subsequent 1% increment in the Ni price then aggregated into 10 potential stages of mine development.

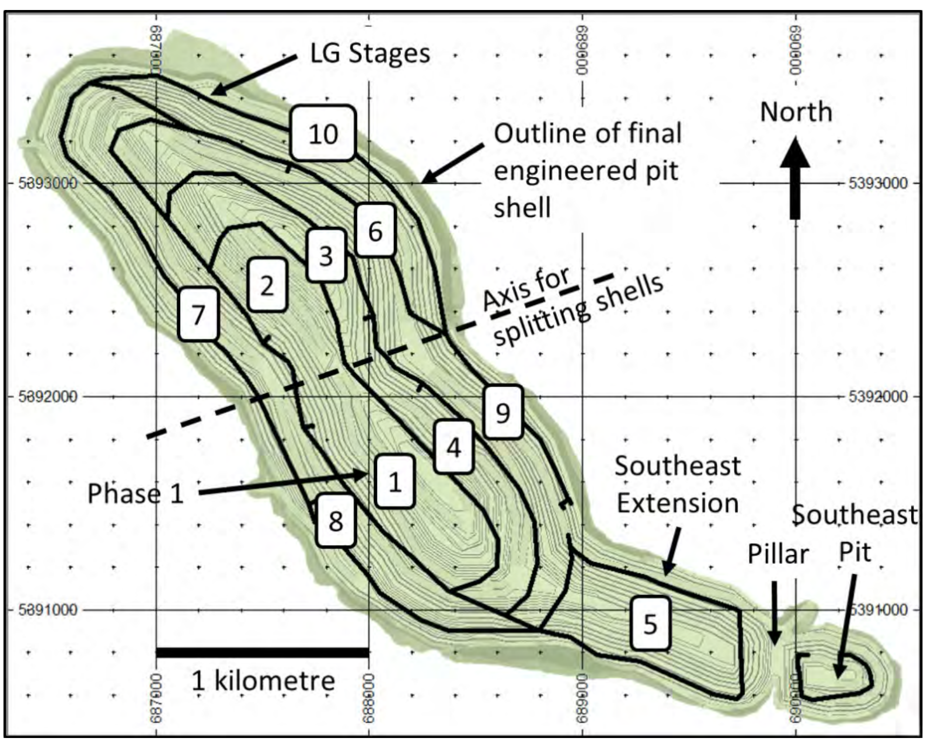
The stages were evaluated using a spreadsheet techno-economic model. This model showed that NPV increased fairly rapidly until Stage 6, which contained 1,175 Mt ore or 10% more than the Revised Prefeasibility Study (Revised PFS) design. Beyond Stage 6, the increase in NPV moderated but continued to increase through Stage 8 (13% more ore than Stage 6), then declined moderately through Stage 10 (27% more ore than Stage 6). It should be noted that the NPV for Stage 10 remained higher than that for Stage 6. The impact of discount rate should also be noted – the trajectory in NPV in part reflects the impact of an 8% discount rate being applied to the longer life that would result with the later stages.

The engineered mine design was based on Stage 6 (see Figure 15.2 for details).

16.3.3 Shell Selection & Initial Schedule

The mining sequence was developed based on the first pass nested LG shells that were discussed in Section 15.2.4. Five intermediate nested shells spaced by the target 100 m minimum mining width and the final pit shell were selected for the phase designs. All shells were then bisected by an approximate mid-point along the long axis of the pit so that the tonnage of individual pushbacks and associated instantaneous stripping rates could be minimized. Splitting the shell increased the number of LG stages to 11 (including 10 in the main pit and the southeast pit as a separate stage). The optimal sequence for mining these was determined by iteration, based on post-tax net present value. Of the 15 different permutations tested, the sequence pictured in Figure 16.7 (Sequence 'O') was determined to be optimal.

Figure 16.7: General Mining Sequence from LG Stages



Source: RNC.

This optimal sequence not only considered ore revenues, stockpiles and waste stripping economics, it also considered the haulage costs. As such, a conceptual ramp system was developed for all the phases.

16.3.4 Pit Phase Design

Engineered design pits were produced for each LG phase shell. All internal phases (1 to 8) used single bench designs while the final pit employs double benching when allowed. Wall slope designs were based on the geotechnical criteria for each sector. The guidelines given in Table 16-2 were simplified with use of a constant 70° bench face angle throughout and a variable safety berm width that was employed to arrive at the specified inter-ramp slope angle. All designs have 3D toe, crest and ramp lines to make a clean accurate design shape for every pit phase. These 3D shapes were intersected at sub-crop and surface topography to make concise pit designs and thus volumes for final scheduling and production calculations.

The phase designs also included ramps of 35 m width, which is sufficient for the 230 t class haul trucks that are planned for use. The overall ramp concept followed the previous PFS ramp design with some adjustments required to ensure that pit exits aligned with access roads to the crusher and dumps, and that four-way intersections (‘butterflies’) are located to allow access to subsequent phases.

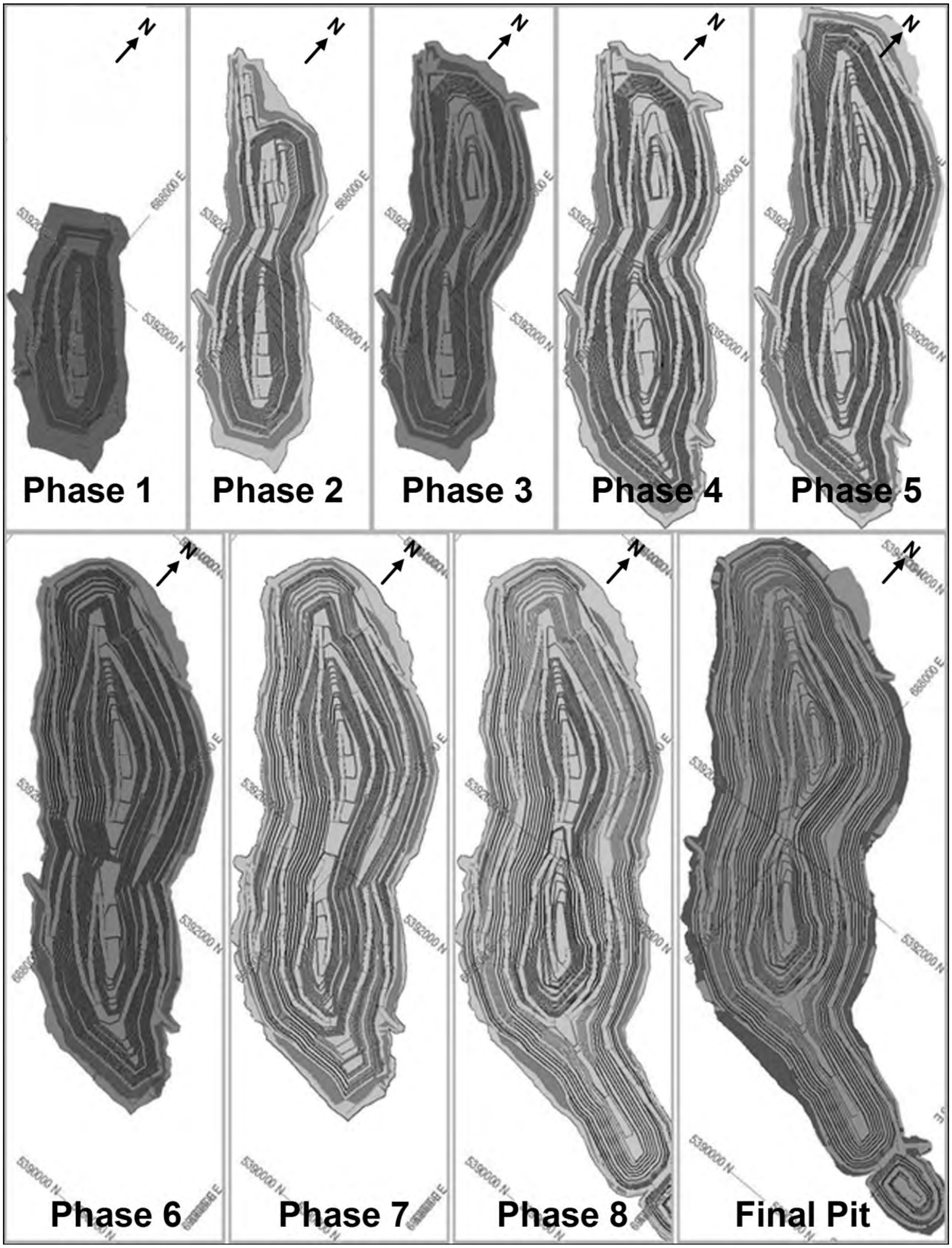
Inclusion of practical considerations such as ramps and crest / toes for walls resulted in some differences between the LG design and engineered pit phases.

These differences were minimized with the engineered design for the ultimate pit representing an improvement on overall stripping ratio with minimal losses of ore as follows:

- the undiluted engineered design contains 99% of the LG design nickel (6,964 Mlb vs. 7,024 Mlb) in 100% of the ore (1,178 Mt vs. 1,175 Mt);
- the engineered design contains 11% less waste (1,337 Mt vs. 1,502 Mt); and
- the engineered design stripping ratio is also 12% lower (1.13 vs. 1.28).

The phase sequencing also follows the optimal LG sequence (Sequence ‘O’ pictured in Figure 16.7 previously). The only material difference is that LG stage 5 is deferred until Phase 8 of the main pit, as shown in Figure 16.8. However, as will be shown in Figures 16.9 through to Figure 16.29, a large portion of Phase 8 is accelerated, with the final annual plans very closely matching the optimal LG sequence. Figure 16.8 shows the phase design pits and final pit. All pits are same scale.

Figure 16.8: Sequence of Engineered Phase Designs



Source: RNC

16.3.5 Annual Plans & Mine Schedule

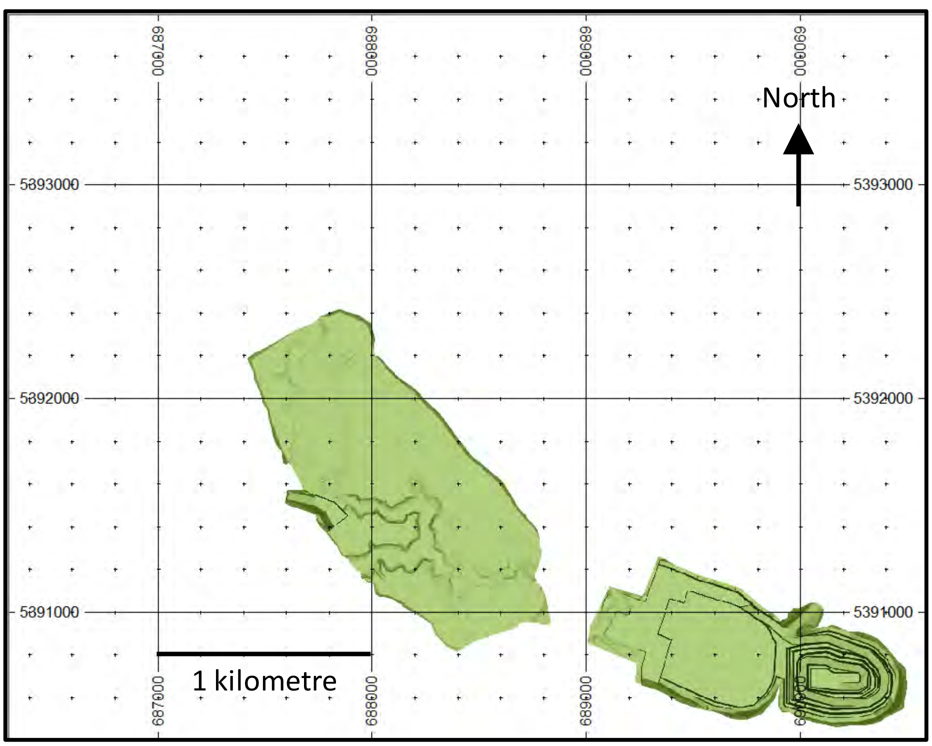
The phase designs were then used as the basis for annual plans that are illustrated in Figures 16.9 through 16.29 on the following pages. The annual 'cuts' were developed around the LG initial sequence described above. Each design pit was depleted in sequence obeying the following guidelines:

- mill feed closely following the LG schedule;
- mine production closely following the LG schedule but was smoothed in some instances to optimize utilization of mine fleet; and
- bench sinking rates kept at a maximum of 10 benches per year (preferably less).

As identified in the preceding section, the annual plans are able to accelerate a portion of the southeast extension (SEE) in order to more closely match the sequence of LG stages. The upper benches of the SEE are excavated early in pre-production in order to obtain construction rock and early ore with a low associated stripping ratio as the depth of overburden in this area is limited. This zone sees additional excavation in the early - mid years to maintain the schedule along the lines of the LG optimization, as well as providing operational flexibility with additional loading faces to facilitate achievement of planned production tonnages.

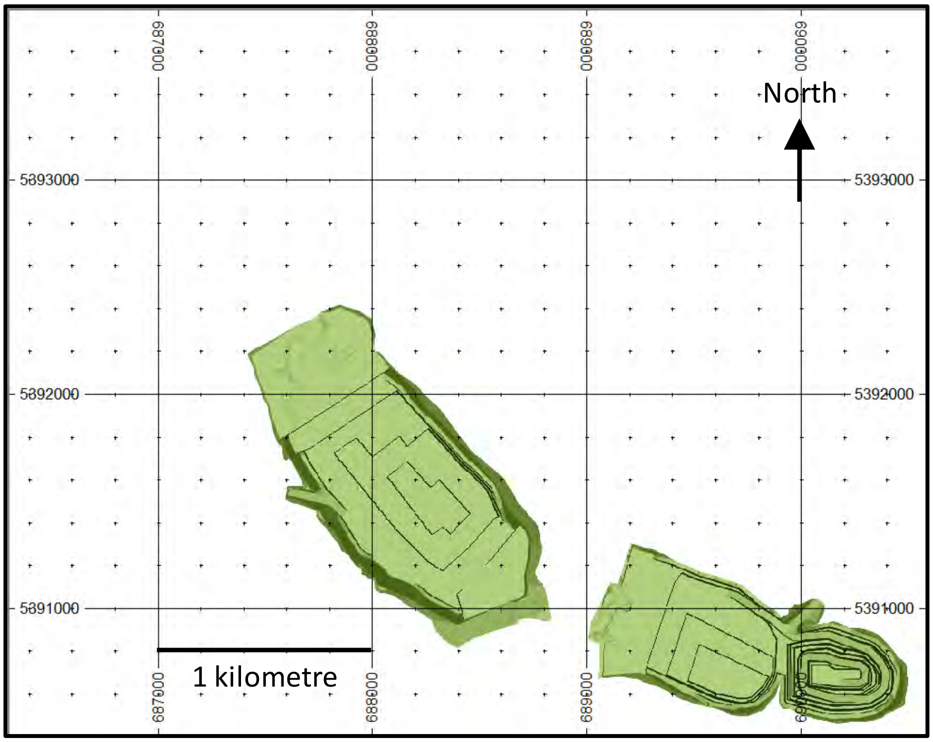
A typical year has two phases in production, while three is maximum and occurs regularly throughout the mine life.

Figure 16.9: Mine Development – End of Pre-Strip



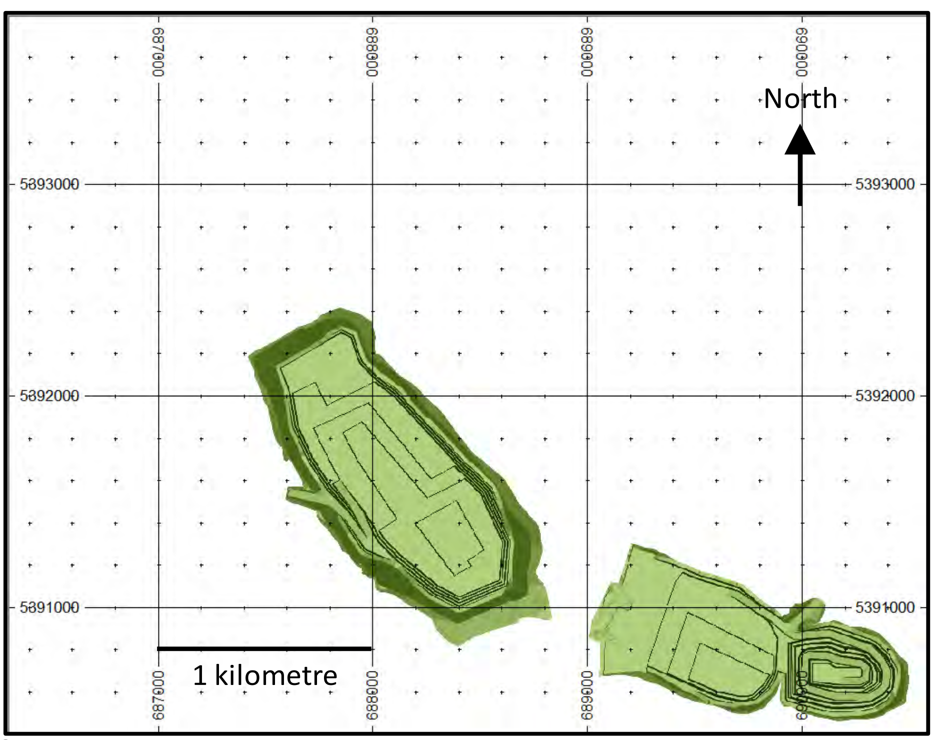
Source: RNC.

Figure 16.10: Mine Development – End of Year 1



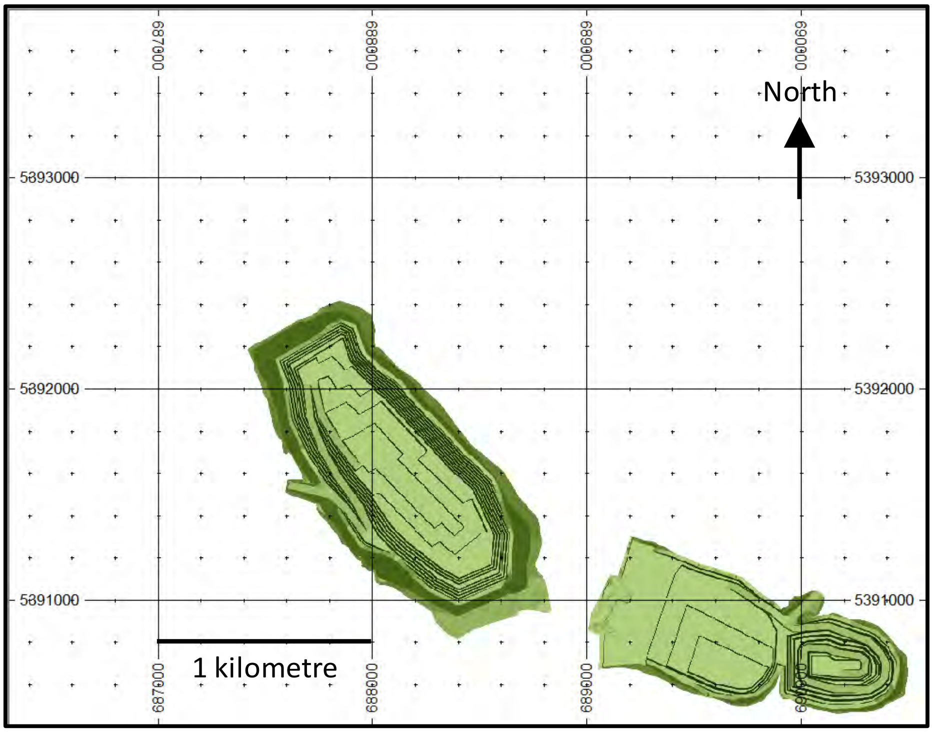
Source: RNC.

Figure 16.11: Mine Development – End of Year 2



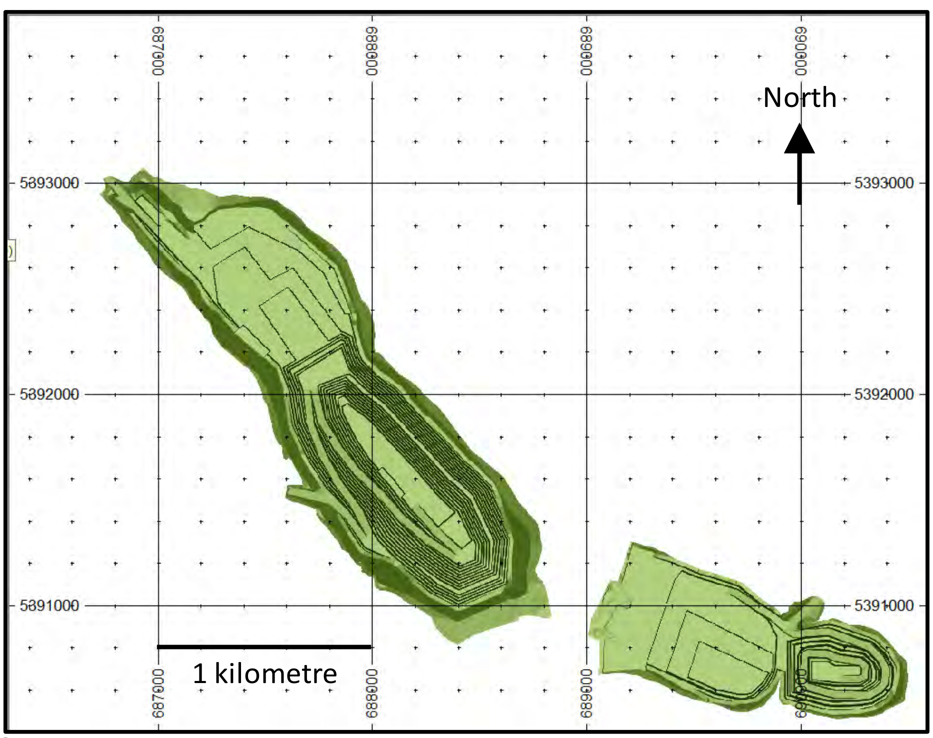
Source: RNC.

Figure 16.12: Mine Development – End of Year 3



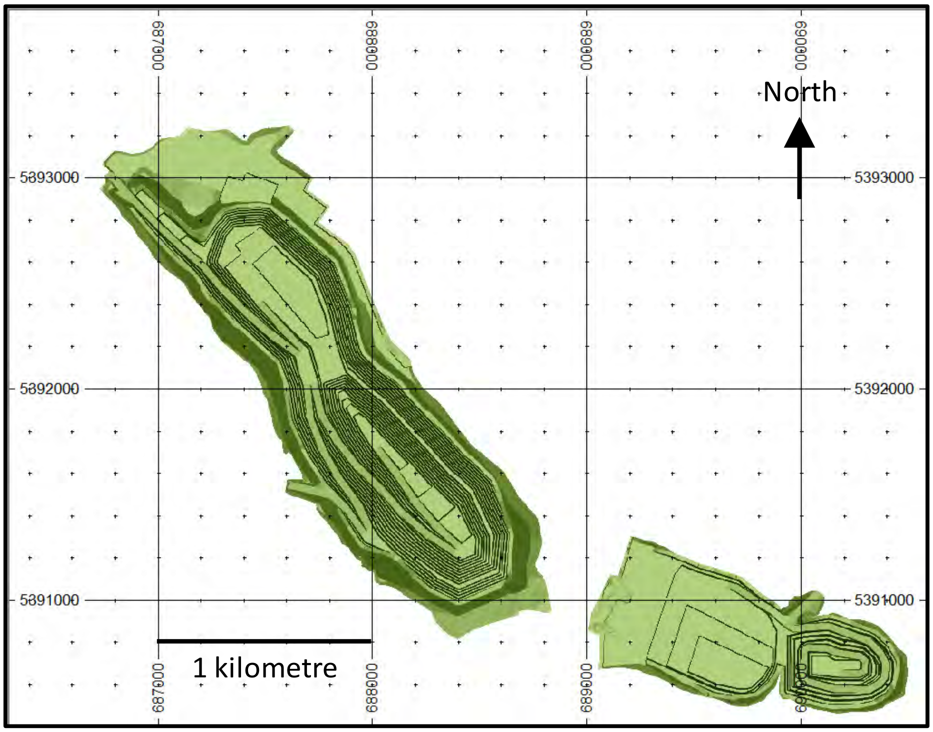
Source: RNC.

Figure 16.13: Mine Development – End of Year 4



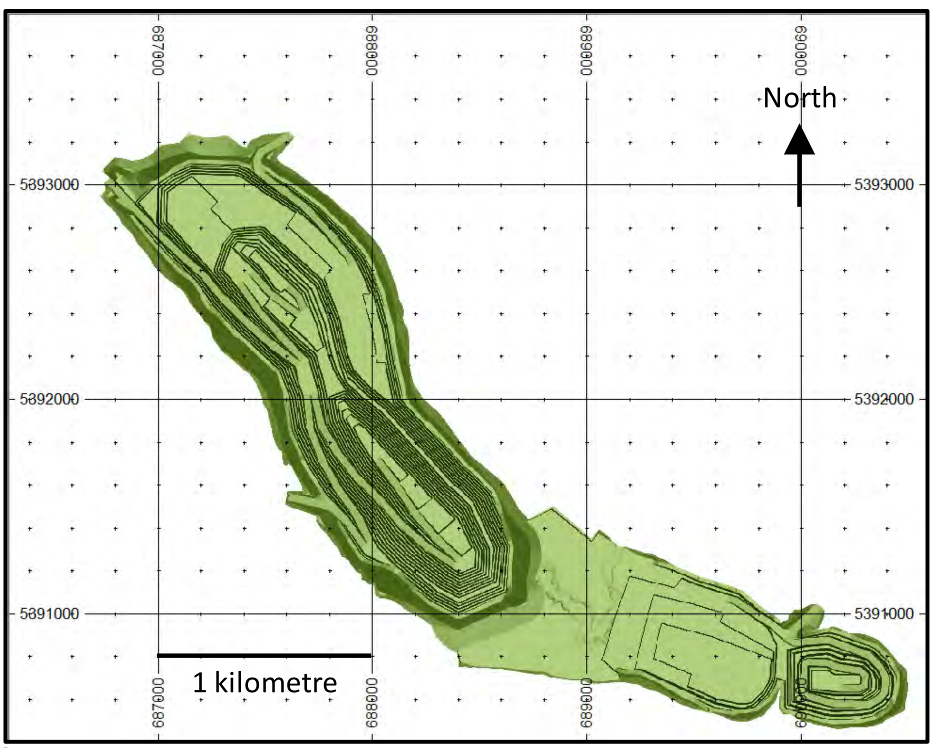
Source: RNC.

Figure 16.14: Mine Development – End of Year 5



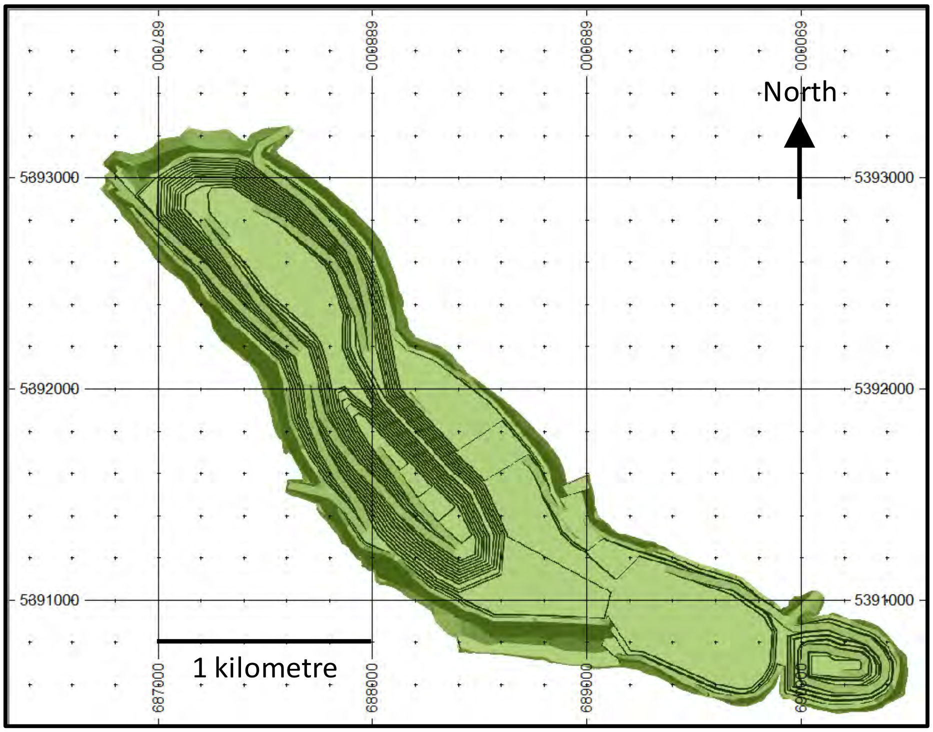
Source: RNC.

Figure 16.15: Mine Development – End of Year 6



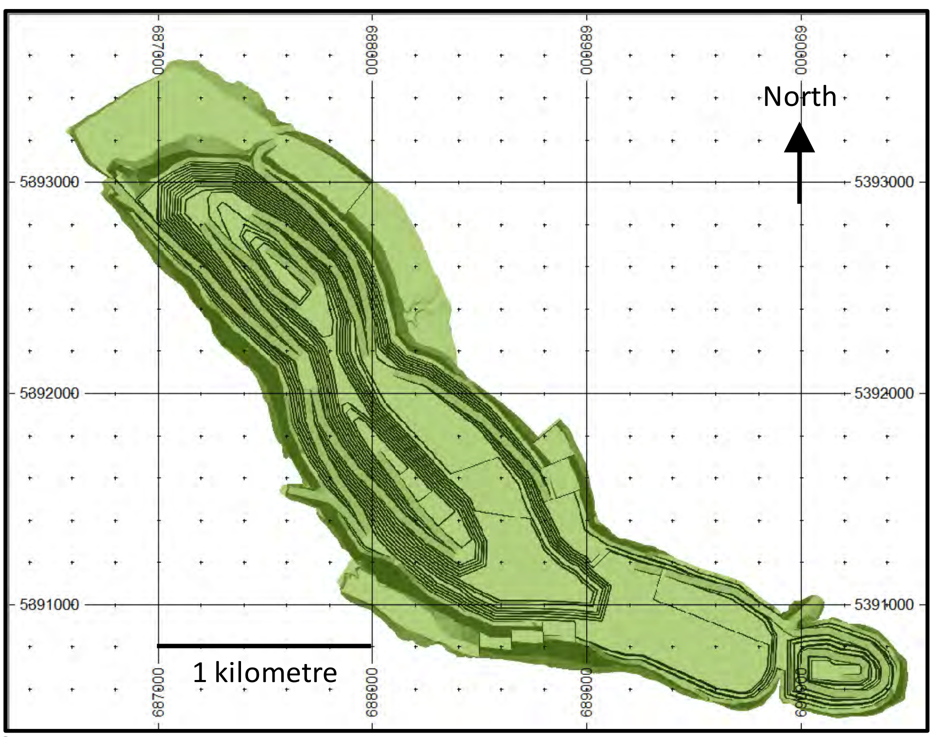
Source: RNC.

Figure 16.16: Mine Development – End of Year 7



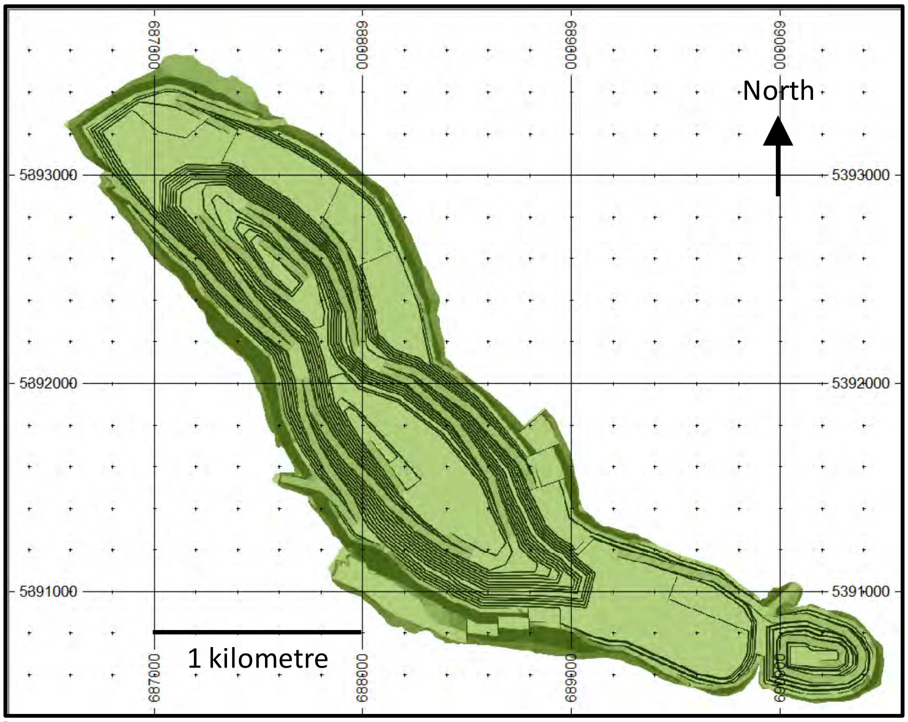
Source: RNC.

Figure 16.17: Mine Development – End of Year 8



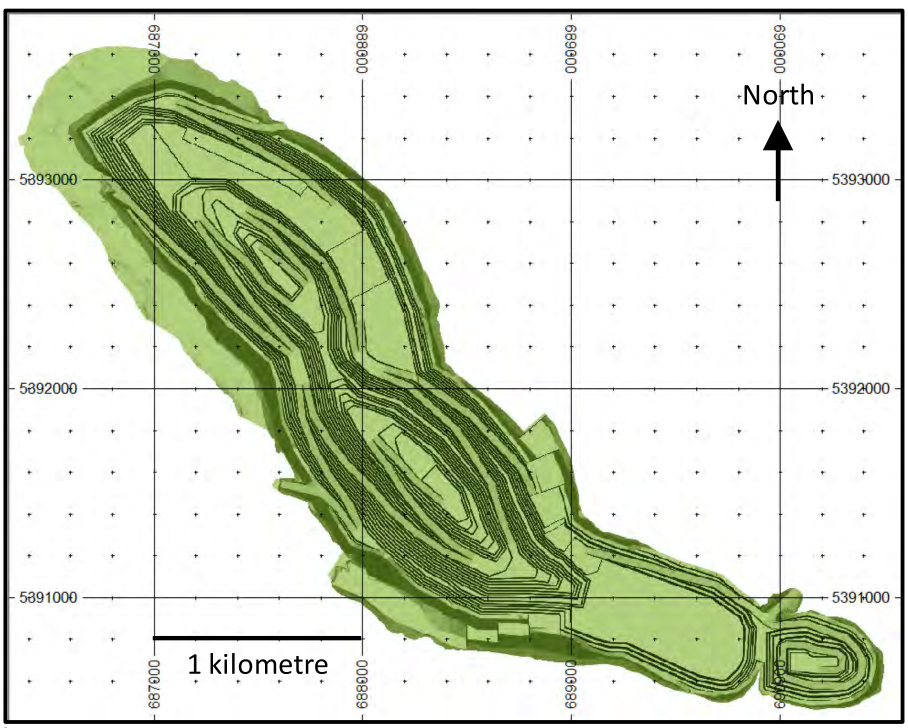
Source: RNC.

Figure 16.18: Mine Development – End of Year 9



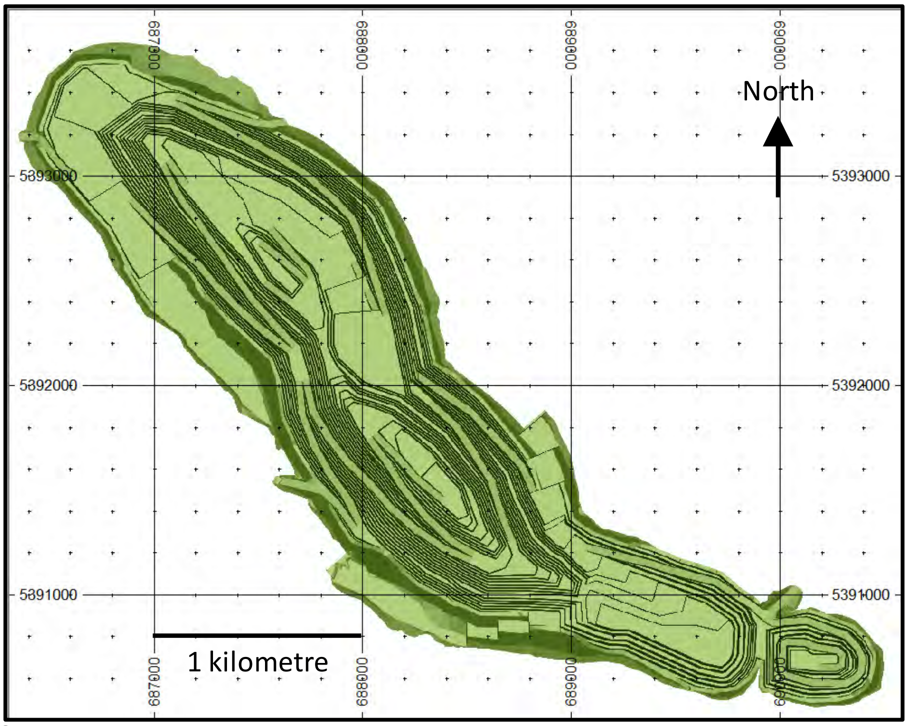
Source: RNC.

Figure 16.19: Mine Development – End of Year 10



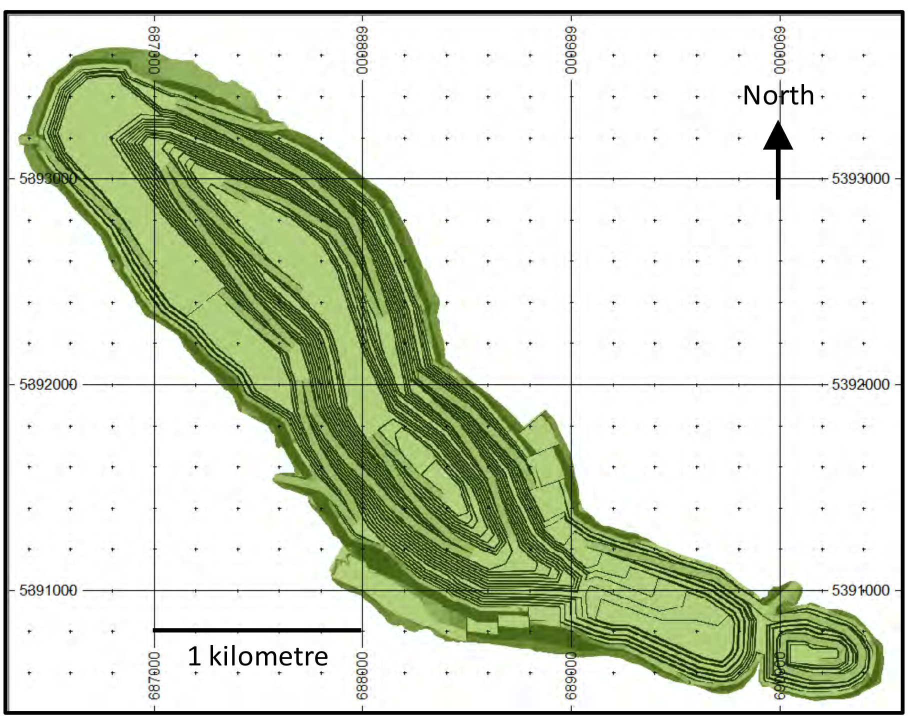
Source: RNC.

Figure 16.20: Mine Development – End of Year 11



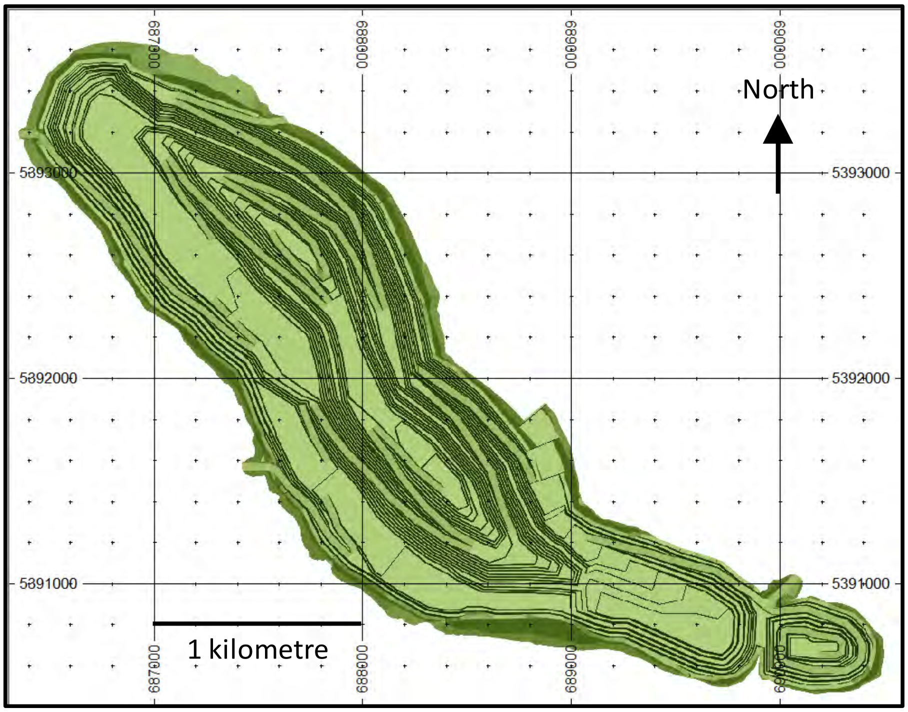
Source: RNC.

Figure 16.21: Mine Development – End of Year 12



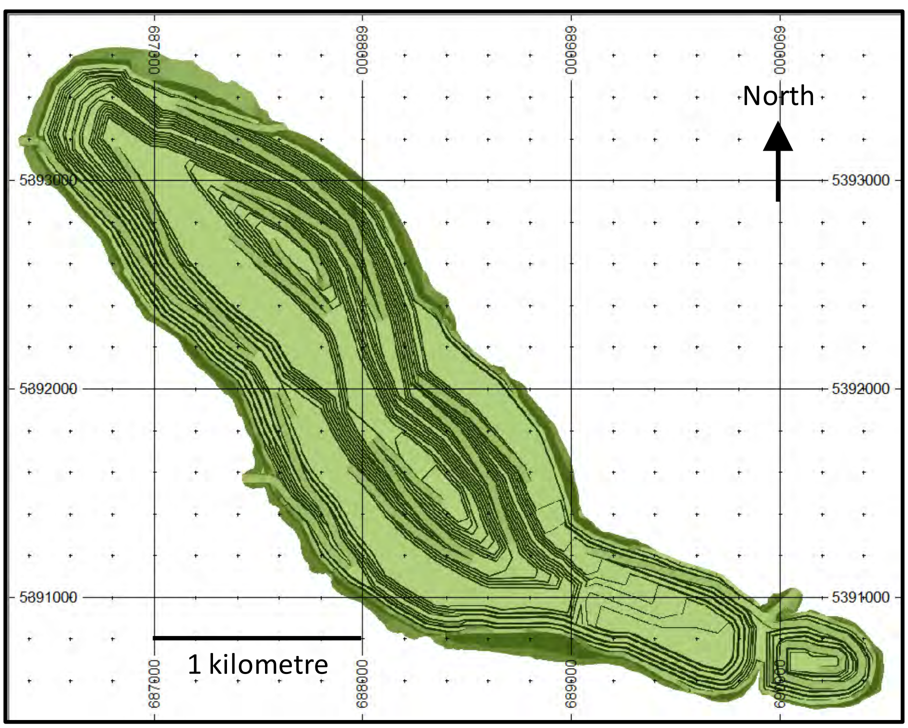
Source: RNC.

Figure 16.22: Mine Development – End of Year 13



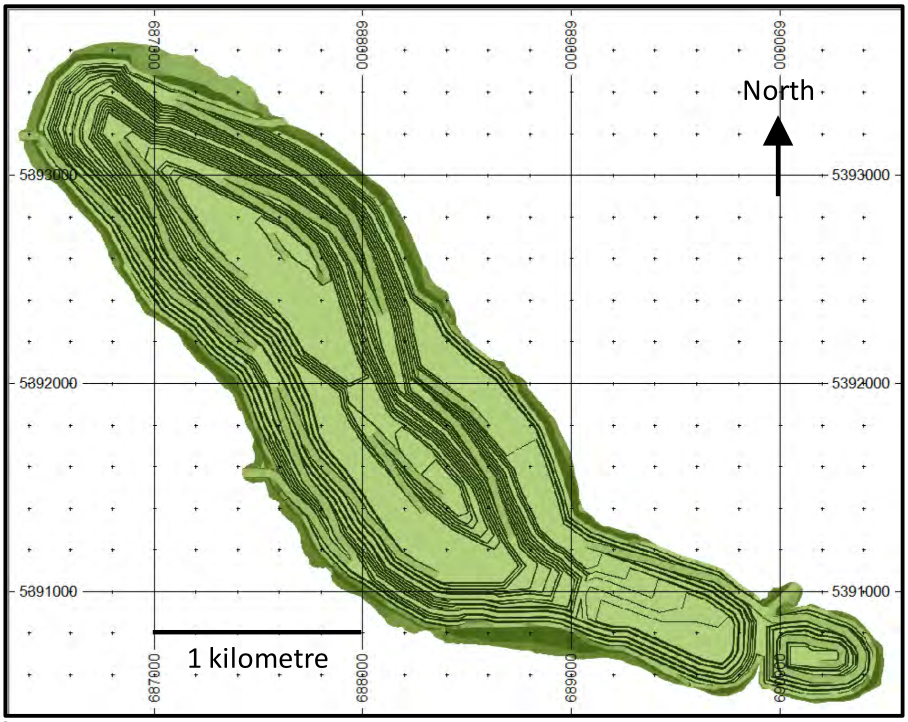
Source: RNC.

Figure 16.23: Mine Development – End of Year 14



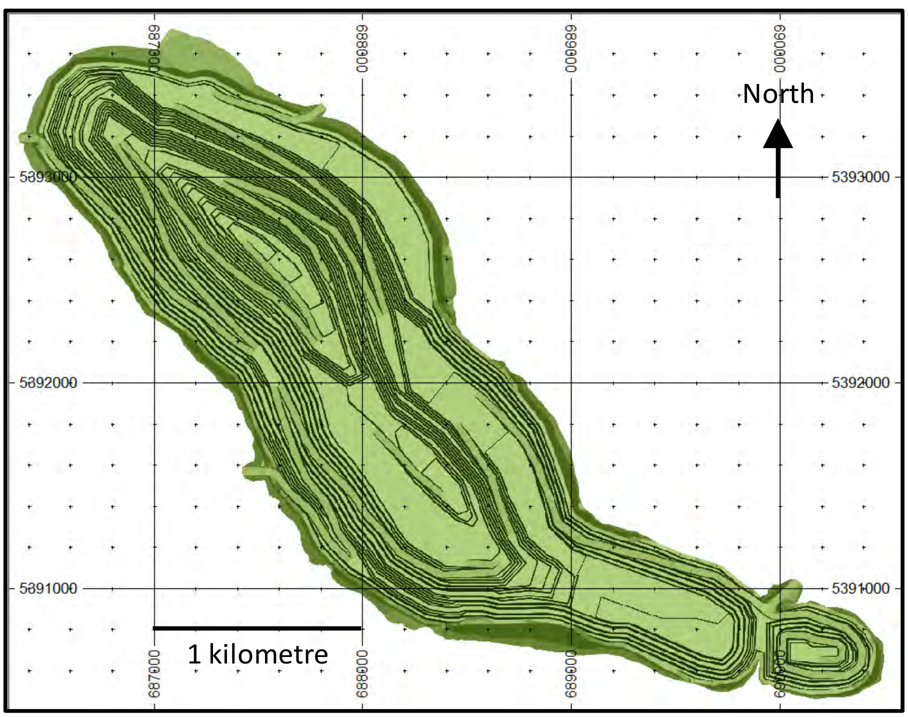
Source: RNC.

Figure 16.24: Mine Development – End of Year 15



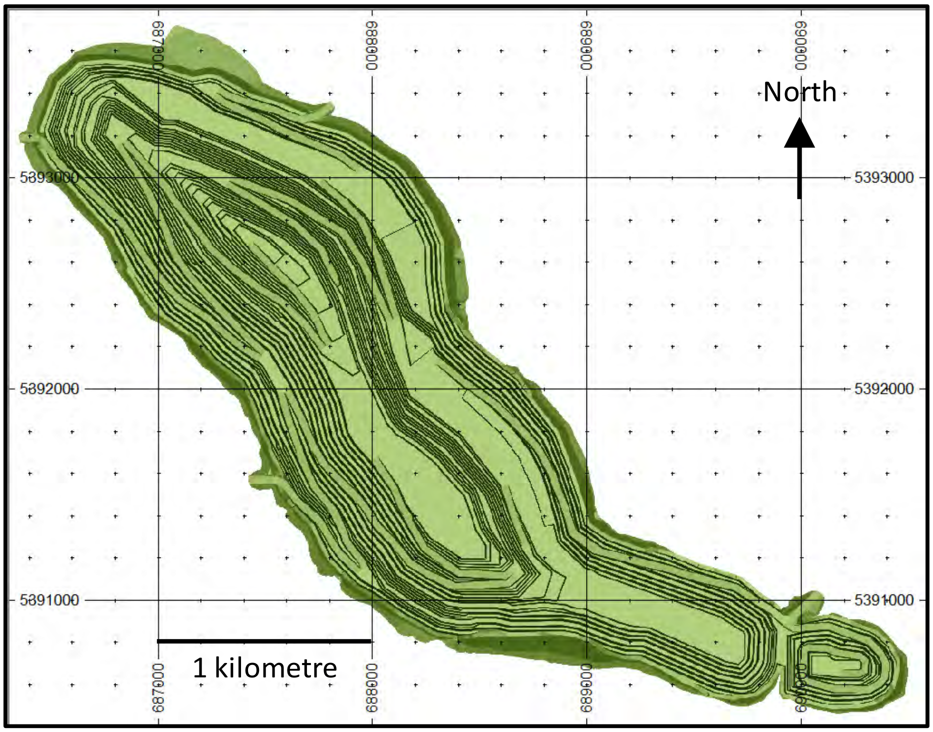
Source: RNC.

Figure 16.25: Mine Development – End of Year 16



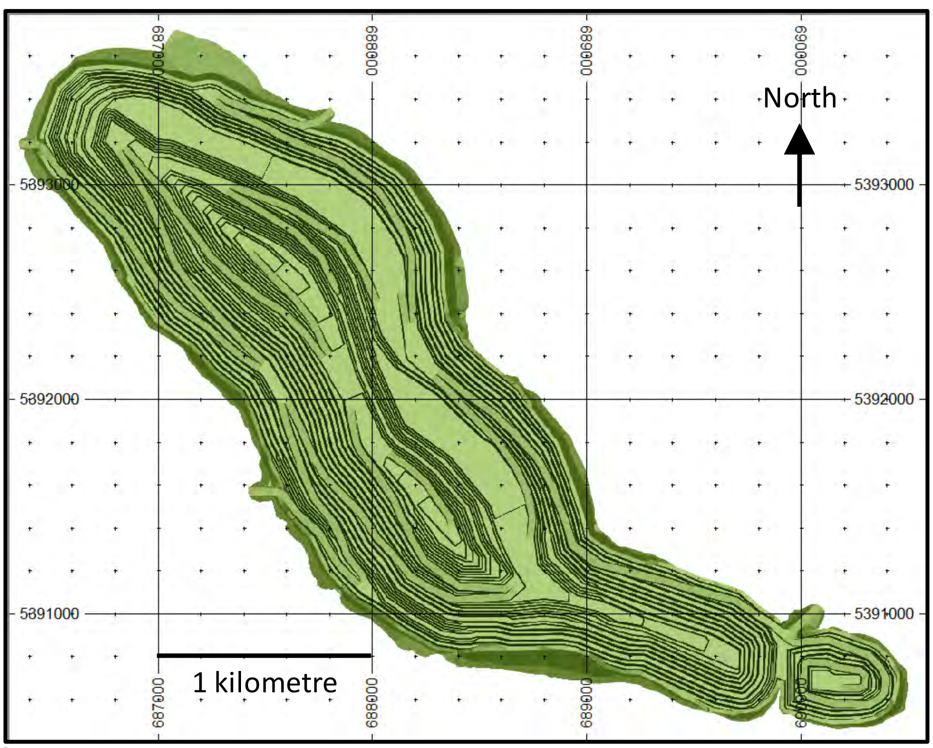
Source: RNC.

Figure 16.26: Mine Development – End of Year 17



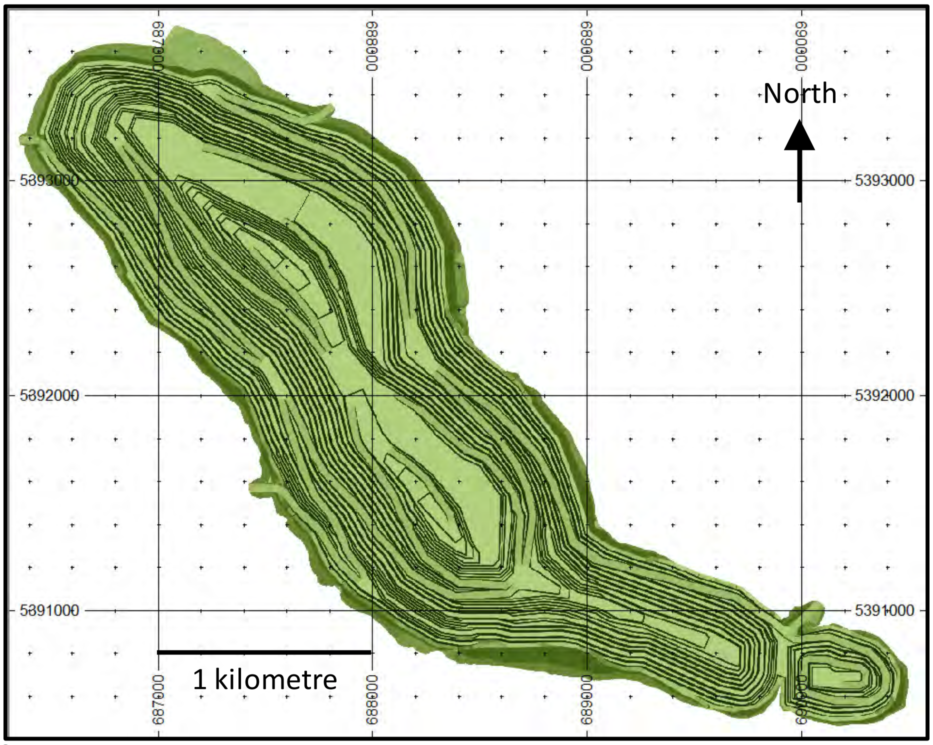
Source: RNC.

Figure 16.27: Mine Development – End of Year 18



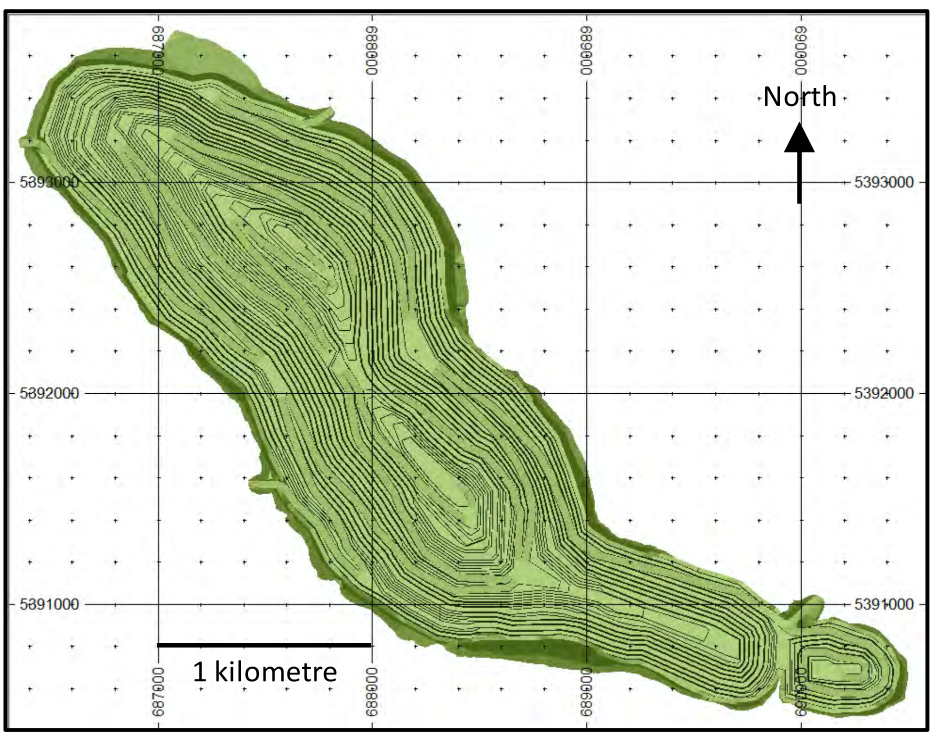
Source: RNC.

Figure 16.28: Mine Development – End of Year 19



Source: RNC.

Figure 16.29: Mine Development – End of Year 20 (End of Mine Life)



Source: RNC.

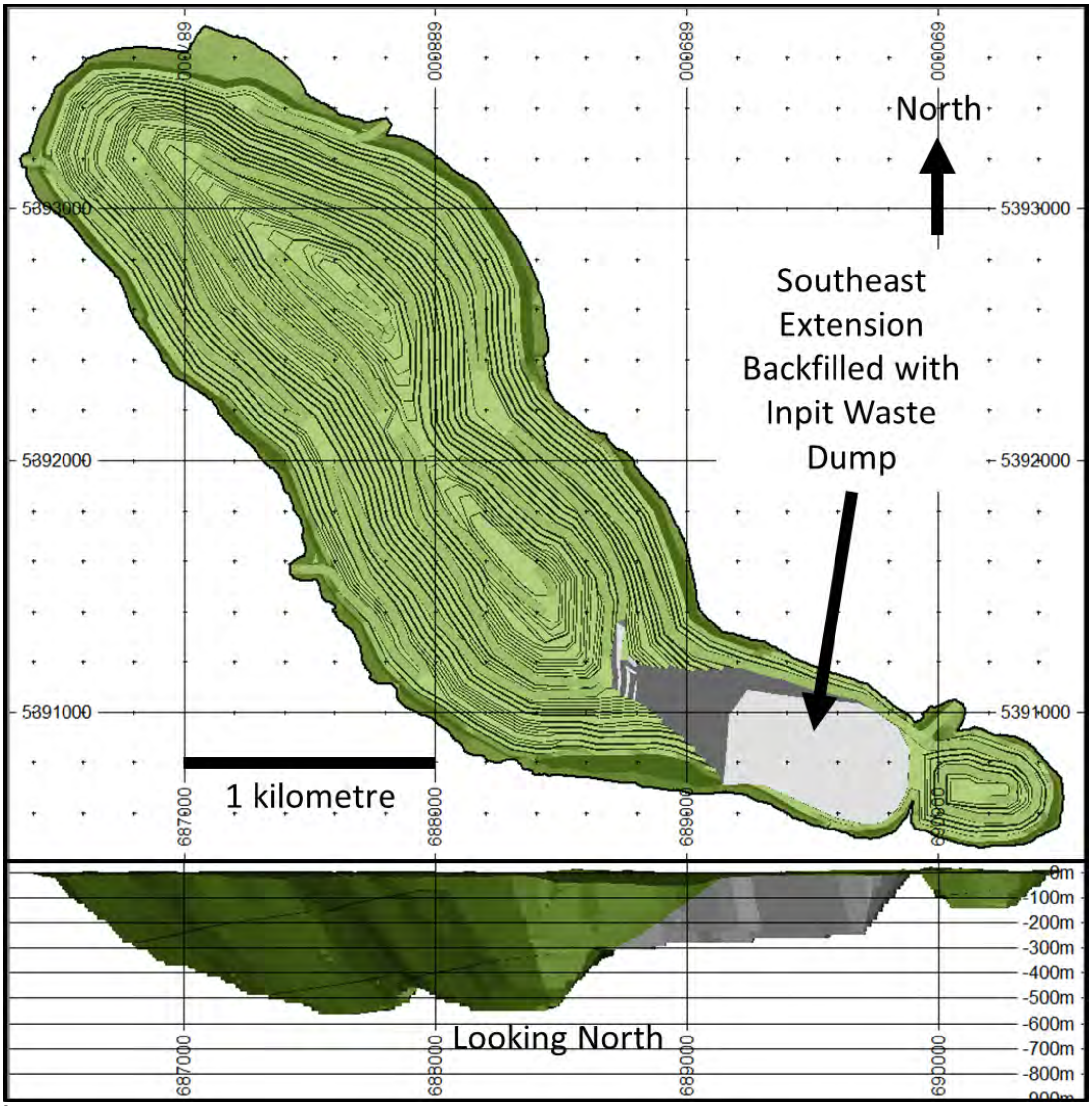
A high level summary of the mining sequence is as follows:

- Mining initiates at the southeast pit, which is at the extreme south-east of the deposit and separated from the main pit by a pillar. The primary focus of the pre-strip plan is to excavate the entire 37 Mt (of which 95% is ore or waste rock, with overburden only 5%) contained within the southeast pit prior to mill start-up, in order to provide a water reservoir of 10 Mm^3 capacity and supply rock for construction. This will be achieved by employing both production excavators from the outset.
- As mining in the southeast pit nears completion, one excavator will be re-allocated to the SEE and primarily target waste rock that will be used for construction. This unit will be active in the SEE until the end of the Year 1 of mill production.
- Upon completion of the southeast pit, the second excavator will be re-allocated to Phase 1 of the main pit, which will have been stripped of clay by the contractor while the southeast pit was being mined.
- At the end of Year 1 (of mill production), both excavators will be active in Phase 1, where they will be joined by the first rope shovel. A second rope shovel will be added one year later. The average daily production rate for this fleet will be approximately 200 kt/d. This production rate will be maintained until the end of Year 6.
- In Year 7, a third rope shovel is added, followed by a fourth in Year 10. With the increased fleet, daily production increases to average approximately 375 kt/d. The excavators will be reserved mainly for loading sand and gravel, as well as sinking new benches and more cost effective rope shovels will be used for the bulk of rock mining. Clay will be mined using much smaller equipment.
- Mining is intermittently active in the SEE from Years 6 to 17. With the completion of mining during Year 18, the void will be backfilled with waste rock from the final phases of mining to the north (Figure 16.30). The tonnage of waste rock planned to be tipped in the SEE is 114 Mt, compared to 189 Mt of waste rock that will be mined after this dump becomes available for tipping.

Figure 16.31 provides a summary of the annual mined tonnages.

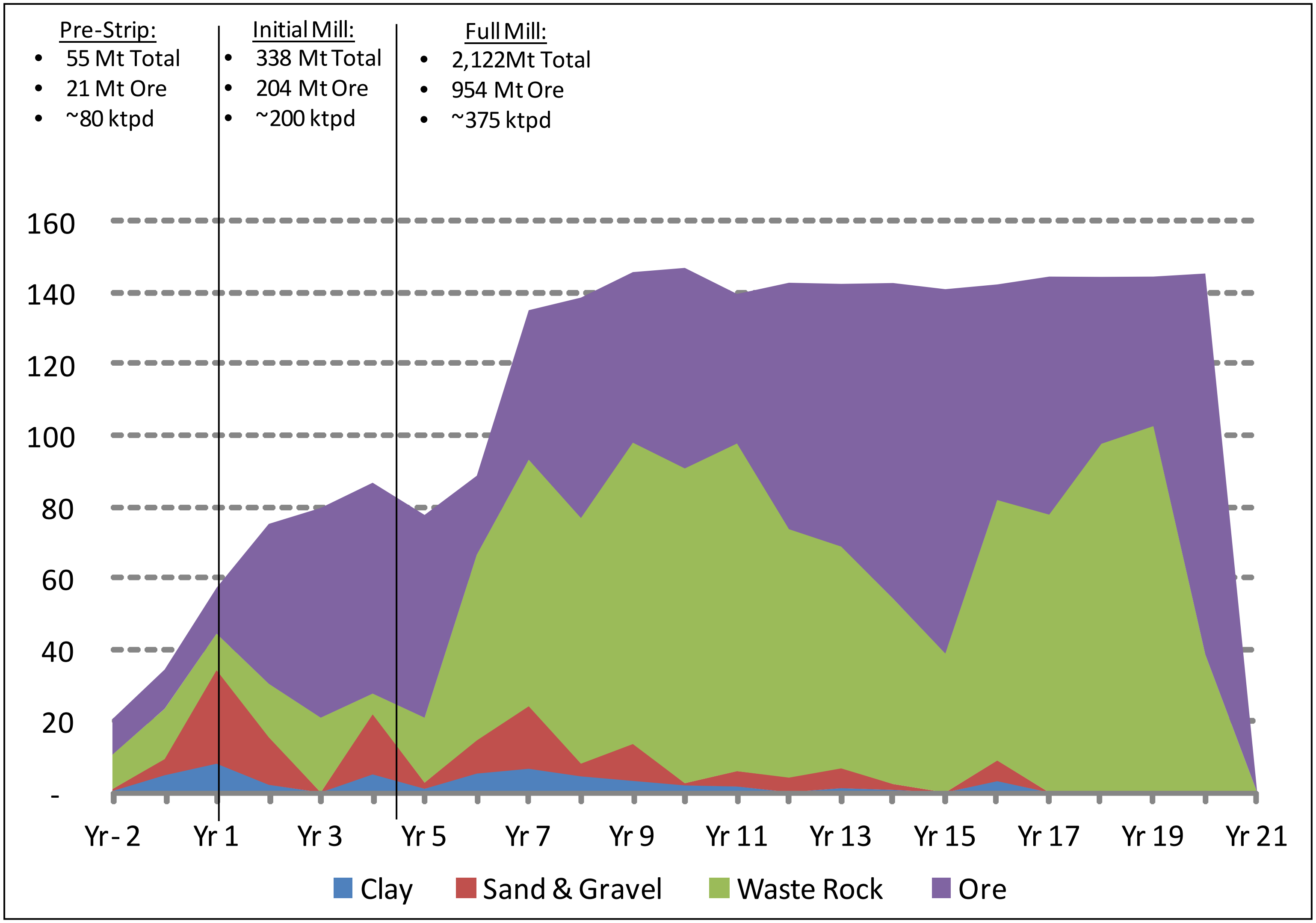
In addition to the re-handle of low-grade ore at surface stockpiles, some overburden (including both clay and sand and gravel) will be re-handled during the construction of the tailings storage facility (TSF). Details of the production schedule, including both ex-pit mining and rehandling of low-grade ore and overburden for construction, are given in Table 16-5. For simplicity of presentation, Table 16-5 also includes a summary of metallurgical production.

Figure 16.30: Mine Development at the End of Mine Life, Showing Waste Rock Dump in SEE



Source: RNC.

Figure 16.31: Summary Mine Production Schedule



Source: RNC.

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Table 16-5: Life-of-Mine Plan

MINING

Ore Mining	units	TOTAL	'14	'15	'16	'17	'18	'19	'20	'21	'22	'23	'24	'25	'26	'27	'28	'29	'30	'31	'32	'33	'34	'35	'36	'37	'38	'39	'40	'41	'42	'43	'44	'45	'46	'47	'48	'49
Diluted Ore	kt	1,178,648	2,547	12,941	11,944	28,770	51,661	58,800	57,841	39,434	32,029	51,763	54,680	51,897	48,915	55,328	71,168	80,896	95,113	81,123	63,486	56,640	44,248	74,174	53,250	-	-	-	-	-	-	-	-	-	-	-	-	-
Grade Ni	% Ni	0.267	0.250	0.245	0.247	0.274	0.286	0.295	0.287	0.275	0.263	0.266	0.271	0.278	0.270	0.253	0.250	0.240	0.243	0.256	0.268	0.266	0.264	0.288	0.298	-	-	-	-	-	-	-	-	-	-	-	-	-
Grade Co	% Co	0.011	0.010	0.009	0.010	0.011	0.011	0.011	0.011	0.010	0.010	0.010	0.011	0.010	0.010	0.010	0.010	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.011	-	-	-	-	-	-	-	-	-	-	-	-	-
Grade Pt	g/.t Pt	0.009	0.004	0.004	0.005	0.009	0.009	0.009	0.009	0.008	0.006	0.007	0.007	0.007	0.006	0.006	0.007	0.009	0.011	0.012	0.011	0.009	0.007	0.009	0.010	-	-	-	-	-	-	-	-	-	-	-	-	-
Grade Pd	g/t Pd	0.019	0.011	0.011	0.013	0.021	0.022	0.022	0.022	0.018	0.014	0.017	0.018	0.017	0.014	0.011	0.013	0.017	0.022	0.025	0.025	0.022	0.016	0.020	0.022	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Ni	klbs	6,942,453	14,040	69,872	64,994	173,661	325,206	381,834	366,614	239,121	185,710	303,712	326,373	318,008	291,115	308,134	392,270	428,525	508,659	458,257	374,917	331,776	257,918	471,780	349,958	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Co	klbs	278,132	539	2,708	2,514	6,723	12,423	14,161	13,673	9,066	7,178	11,981	12,819	11,942	10,939	12,388	16,474	19,295	23,307	20,126	15,587	13,628	10,343	17,563	12,754	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Pt	koz	328	0	2	2	9	16	18	17	10	7	12	13	12	9	10	16	24	35	32	23	17	10	20	17	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Pd	koz	716	1	4	5	19	36	41	38	22	15	28	31	28	21	20	30	43	67	66	51	39	23	47	38	-	-	-	-	-	-	-	-	-	-	-	-	-

Total Expit Mining	units	TOTAL	'14	'15	'16	'17	'18	'19	'20	'21	'22	'23	'24	'25	'26	'27	'28	'29	'30	'31	'32	'33	'34	'35	'36	'37	'38	'39	'40	'41	'42	'43	'44	'45	'46	'47	'48	'49
Ore	kt	1,178,648	2,547	12,941	11,944	28,770	51,661	58,800	57,841	39,434	32,029	51,763	54,680	51,897	48,915	55,328	71,168	80,896	95,113	81,123	63,486	56,640	44,248	74,174	53,250	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste Rock	kt	1,159,450	2,472	14,245	12,154	12,623	18,004	13,412	11,993	35,050	60,461	68,864	76,511	86,156	89,877	80,605	65,785	56,951	45,368	55,892	75,324	87,696	100,068	70,599	19,340	-	-	-	-	-	-	-	-	-	-	-	-	-
Clay	kt	49,940	112	2,750	6,405	5,067	1,075	2,524	3,030	3,140	5,944	5,552	3,870	2,627	1,837	887	650	965	368	1,568	1,568	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sand & Gravel	kt	126,352	132	2,654	15,394	19,780	6,664	8,431	9,323	5,578	13,417	10,554	6,999	5,513	2,483	4,231	4,873	3,641	847	2,919	2,919	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Expit	**kt**	**2,514,390**	**5,264**	**32,590**	**45,897**	**66,239**	**77,405**	**83,167**	**82,188**	**83,201**	**111,851**	**136,733**	**142,059**	**146,193**	**143,112**	**141,051**	**142,476**	**142,454**	**141,697**	**141,503**	**143,297**	**144,335**	**144,316**	**144,774**	**72,591**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**

Rehandle / Construction	units	TOTAL	'14	'15	'16	'17	'18	'19	'20	'21	'22	'23	'24	'25	'26	'27	'28	'29	'30	'31	'32	'33	'34	'35	'36	'37	'38	'39	'40	'41	'42	'43	'44	'45	'46	'47	'48	'49
Low Grade Ore Rehandle	kt	605,842	-	-	3,445	5,052	-	-	-	14,195	22,903	14,548	10,052	5,682	4,927	7,007	4,502	2,422	-	-	-	1,402	4,200	2,798	19,176	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	23,317
Rehandle Clay	kt	17,096	267	734	337	552	816	516	213	1,358	4,982	3,731	-	94	166	310	435	479	444	214	344	474	474	158	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Rehandle Sand & Gravel	kt	2,502	-	-	-	-	-	2,502	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Rehandle	**kt**	**625,441**	**267**	**734**	**3,781**	**5,604**	**816**	**3,019**	**213**	**15,552**	**27,885**	**18,279**	**10,052**	**5,776**	**5,093**	**7,317**	**4,937**	**2,901**	**444**	**214**	**344**	**1,876**	**4,674**	**2,955**	**19,176**	**38,351**	**38,351**	**38,351**	**38,351**	**38,351**	**38,351**	**38,351**	**38,351**	**38,351**	**38,351**	**38,351**	**38,351**	**23,317**

Hauling Distance	units	TOTAL	'14	'15	'16	'17	'18	'19	'20	'21	'22	'23	'24	'25	'26	'27	'28	'29	'30	'31	'32	'33	'34	'35	'36	'37	'38	'39	'40	'41	'42	'43	'44	'45	'46	'47	'48	'49
Expit Rock (Ore & Waste)	1-way metres	4,720	3,785	4,714	3,876	3,412	3,944	4,337	3,686	3,085	3,073	3,483	4,669	4,689	4,737	4,404	4,531	5,258	5,116	6,100	5,582	4,858	4,757	5,659	6,589	-	-	-	-	-	-	-	-	-	-	-	-	-
Clay	1-way metres	2,263	1,168	1,914	1,518	1,486	1,543	1,953	1,947	2,473	2,356	2,374	2,723	2,850	2,784	2,958	3,344	3,459	3,536	3,223	3,429	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sand & Gravel	1-way metres	3,537	907	2,386	4,466	3,695	3,236	3,067	4,765	3,517	2,403	2,003	2,686	2,865	3,441	4,124	6,002	6,468	4,820	3,151	2,452	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Expit	**1-way metres**	**4,613**	**3,657**	**4,290**	**3,745**	**3,349**	**3,850**	**4,136**	**3,744**	**3,090**	**2,954**	**3,324**	**4,518**	**4,588**	**4,689**	**4,387**	**4,575**	**5,276**	**5,111**	**6,008**	**5,496**	**4,858**	**4,757**	**5,659**	**6,589**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Low Grade Ore Rehandle	1-way metres	2,585	-	-	160	160	-	-	-	397	1,781	1,646	1,798	1,894	1,712	2,417	2,112	1,893	-	-	-	1,452	1,334	2,036	1,798	1,755	1,932	3,387	3,421	3,739	3,403	3,370	2,508	2,720	2,594	2,543	2,904	2,723
Clay Rehandle	1-way metres	3,660	2,868	3,854	7,946	6,720	3,675	3,125	3,420	3,603	3,437	3,320	-	3,310	3,339	3,382	3,705	3,693	3,498	3,521	3,587	3,617	3,657	3,670	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sand & Gravel Rehandle	1-way metres	3,370	-	-	-	-	-	3,370	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Rehandle	**1-way metres**	**2,617**	**2,868**	**3,854**	**854**	**806**	**3,675**	**3,328**	**3,420**	**677**	**2,077**	**1,988**	**1,798**	**1,917**	**1,765**	**2,458**	**2,252**	**2,190**	**3,498**	**3,521**	**3,587**	**1,999**	**1,569**	**2,124**	**1,798**	**1,755**	**1,932**	**3,387**	**3,421**	**3,739**	**3,403**	**3,370**	**2,508**	**2,720**	**2,594**	**2,543**	**2,904**	**2,723**

PROCESSING

Ore Milled	units	TOTAL	'14	'15	'16	'17	'18	'19	'20	'21	'22	'23	'24	'25	'26	'27	'28	'29	'30	'31	'32	'33	'34	'35	'36	'37	'38	'39	'40	'41	'42	'43	'44	'45	'46	'47	'48	'49
Diluted Ore	kt	1,178,648	-	-	7,191	19,176	19,176	19,176	19,176	35,954	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	38,351	23,317
Grade Ni	% Ni	0.267	-	-	0.263	0.300	0.358	0.374	0.351	0.280	0.274	0.282	0.286	0.294	0.282	0.270	0.272	0.259	0.268	0.286	0.288	0.277	0.269	0.310	0.310	0.271	0.269	0.232	0.232	0.232	0.232	0.232	0.232	0.232	0.232	0.231	0.221	0.221
Grade Co	% Co	0.011	-	-	0.010	0.010	0.011	0.011	0.011	0.010	0.011	0.011	0.011	0.011	0.010	0.010	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.011	0.010	0.010	0.010	0.010	0.010	0.010	0.010	0.010	0.010	0.010	0.010
Grade Pt	g/.t Pt	0.009	-	-	0.006	0.009	0.014	0.014	0.012	0.008	0.008	0.008	0.009	0.009	0.007	0.007	0.009	0.009	0.012	0.014	0.013	0.011	0.008	0.010	0.011	0.009	0.009	0.007	0.007	0.007	0.007	0.007	0.007	0.007	0.007	0.007	0.005	0.005
Grade Pd	g/t Pd	0.019	-	-	0.015	0.023	0.033	0.034	0.029	0.020	0.018	0.020	0.021	0.021	0.017	0.015	0.017	0.019	0.026	0.031	0.030	0.025	0.018	0.024	0.025	0.019	0.019	0.014	0.014	0.014	0.014	0.014	0.013	0.013	0.013	0.013	0.007	0.007
Contained Ni	klbs	6,942,453	-	-	41,717	126,618	151,545	158,058	148,463	221,653	231,486	238,628	241,821	248,547	238,423	228,120	230,368	219,033	226,601	241,819	243,857	234,204	227,111	261,714	261,980	229,134	226,879	195,589	195,589	195,589	195,589	195,589	196,286	196,353	196,353	195,240	187,069	115,434
Contained Co	klbs	278,132	-	-	1,541	4,375	4,751	4,809	4,682	8,316	8,904	9,056	9,170	9,047	8,759	8,768	9,008	9,075	9,318	9,551	9,517	9,338	9,082	9,304	9,505	9,336	9,305	8,873	8,873	8,873	8,873	8,873	8,826	8,821	8,821	8,801	8,647	5,335
Contained Pt	koz	328	-	-	1	6	8	9	7	10	10	10	11	11	9	9	11	12	15	17	16	13	10	13	14	11	11	9	9	9	9	9	9	9	9	8	6	4
Contained Pd	koz	715	-	-	4	14	20	21	18	23	22	24	26	26	20	19	21	24	33	38	36	31	23	29	31	23	23	18	18	18	18	18	17	16	16	16	9	6
Recovery Ni		43.0%	0.00%	0.00%	46.20%	50.37%	52.54%	55.09%	53.93%	46.85%	45.08%	46.22%	45.86%	46.64%	46.85%	46.20%	47.05%	45.77%	47.52%	49.72%	50.42%	48.35%	44.43%	51.81%	52.50%	44.99%	44.42%	35.16%	35.16%	35.16%	35.16%	35.16%	29.35%	28.80%	28.80%	28.16%	23.22%	23.22%
Recovery Co		42.1%	0.00%	0.00%	46.24%	50.39%	52.47%	55.09%	53.87%	46.75%	45.08%	46.21%	45.85%	46.54%	46.73%	46.17%	47.03%	45.76%	47.45%	49.72%	50.42%	48.28%	44.46%	51.16%	51.77%	44.96%	44.34%	35.13%	35.13%	35.13%	35.13%	35.13%	29.34%	28.77%	28.77%	28.12%	23.21%	23.21%
Recovery Pt		62.5%	0.00%	0.00%	52.20%	63.47%	65.61%	65.25%	64.83%	63.30%	60.20%	62.19%	63.56%	62.59%	59.46%	58.34%	58.67%	55.63%	57.84%	62.10%	63.06%	62.28%	61.64%	64.36%	64.89%	62.57%	62.58%	62.71%	62.71%	62.71%	62.71%	62.71%	65.56%	65.84%	65.84%	65.90%	66.53%	66.53%
Recovery Pd		60.7%	0.00%	0.00%	46.37%	61.79%	66.32%	65.59%	64.74%	60.11%	56.37%	60.20%	62.80%	60.89%	55.08%	53.37%	54.96%	47.87%	51.43%	59.49%	61.12%	59.67%	58.75%	63.94%	65.65%	62.14%	62.11%	61.47%	61.47%	61.47%	61.47%	61.47%	66.29%	66.78%	66.78%	66.86%	67.92%	67.92%
Recovered Ni	klbs	2,982,700	-	-	19,275	63,782	79,619	87,072	80,068	103,851	104,364	110,292	110,899	115,922	111,710	105,385	108,398	100,251	107,684	120,231	122,950	113,240	100,914	135,604	137,546	103,081	100,775	68,771	68,771	68,771	68,771	68,771	57,616	56,544	56,544	54,974	43,447	26,805
Recovered Co	klbs	117,104	-	-	713	2,205	2,493	2,649	2,522	3,888	4,014	4,185	4,205	4,210	4,093	4,049	4,237	4,152	4,422	4,748	4,798	4,508	4,038	4,760	4,920	4,197	4,126	3,117	3,117	3,117	3,117	3,117	2,589	2,538	2,538	2,475	2,007	1,238
Recovered Pt	koz	205	-	-	1	4	6	6	5	6	6	6	7	7	5	5	6	6	9	11	10	8	6	8	9	7	7	5	5	5	5	5	6	6	6	5	4	2
Recovered Pd	koz	434	-	-	2	9	14	14	11	14	12	14	16	16	11	10	12	11	17	22	22	18	13	19	20	14	14	11	11	11	11	11	11	11	11	10	6	4

Treatment & Refining	units	TOTAL	'14	'15	'16	'17	'18	'19	'20	'21	'22	'23	'24	'25	'26	'27	'28	'29	'30	'31	'32	'33	'34	'35	'36	'37	'38	'39	'40	'41	'42	'43	'44	'45	'46	'47	'48	'49
Concentrate Treated	kt	4,635	-	-	32	105	134	142	123	152	155	172	189	194	166	146	152	140	158	188	195	177	157	208	206	153	150	107	107	107	107	107	97	96	96	94	76	47
Concentrate Grade - Ni	% Ni	29.2	-	-	27.4	27.6	26.9	27.9	29.5	30.9	30.6	29.1	26.6	27.1	30.5	32.8	32.4	32.5	30.9	29.0	28.5	29.1	29.1	29.6	30.3	30.5	30.5	29.1	29.1	29.1	29.1	29.1	27.0	26.7	26.7	26.7	25.9	25.9
Concentrate Grade - Co	% Co	1.1	-	-	1.0	1.0	0.8	0.8	0.9	1.2	1.2	1.1	1.0	1.0	1.1	1.3	1.3	1.3	1.3	1.1	1.1	1.2	1.2	1.0	1.1	1.2	1.2	1.3	1.3	1.3	1.3	1.3	1.2	1.2	1.2	1.2	1.2	1.2
Concentrate Grade - PGE	PGE g/t	4.3	-	-	2.3	3.7	4.4	4.2	4.1	4.0	3.7	3.8	3.8	3.5	3.1	3.2	3.7	3.9	5.0	5.5	5.1	4.7	3.8	4.0	4.4	4.3	4.3	4.7	4.7	4.7	4.7	4.7	5.3	5.4	5.4	5.3	4.1	4.1
Payable Ni	klbs	2,773,911	-	-	17,926	59,317	74,046	80,977	74,463	96,581	97,059	102,572	103,136	107,808	103,890	98,008	100,810	93,234	100,146	111,815	114,344	105,313	93,850	126,111	127,918	95,866	93,721	63,957	63,957	63,957	63,957	63,957	53,583	52,586	52,586	51,126	40,405	24,929
Payable Co	klbs	58,552	-	-	356	1,102	1,246	1,325	1,261	1,944	2,007	2,092	2,102	2,105	2,047	2,024	2,118	2,076	2,211	2,374	2,399	2,254	2,019	2,380	2,460	2,099	2,063	1,559	1,559	1,559	1,559	1,559	1,295	1,269	1,269	1,237	1,003	619
Payable Pt	koz	157	-	-	0	3	4	4	4	5	4	5	5	5	4	4	4	5	7	9	8	6	4	6	7	5	5	4	4	4	4	4	5	5	5	4	3	2
Payable Pd	koz	332	-	-	1	6	10	10	9	10	9	11	12	11	8	7	9	8	13	18	18	14	10	14	16	11	11	8	8	8	8	8	9	9	9	8	5	3

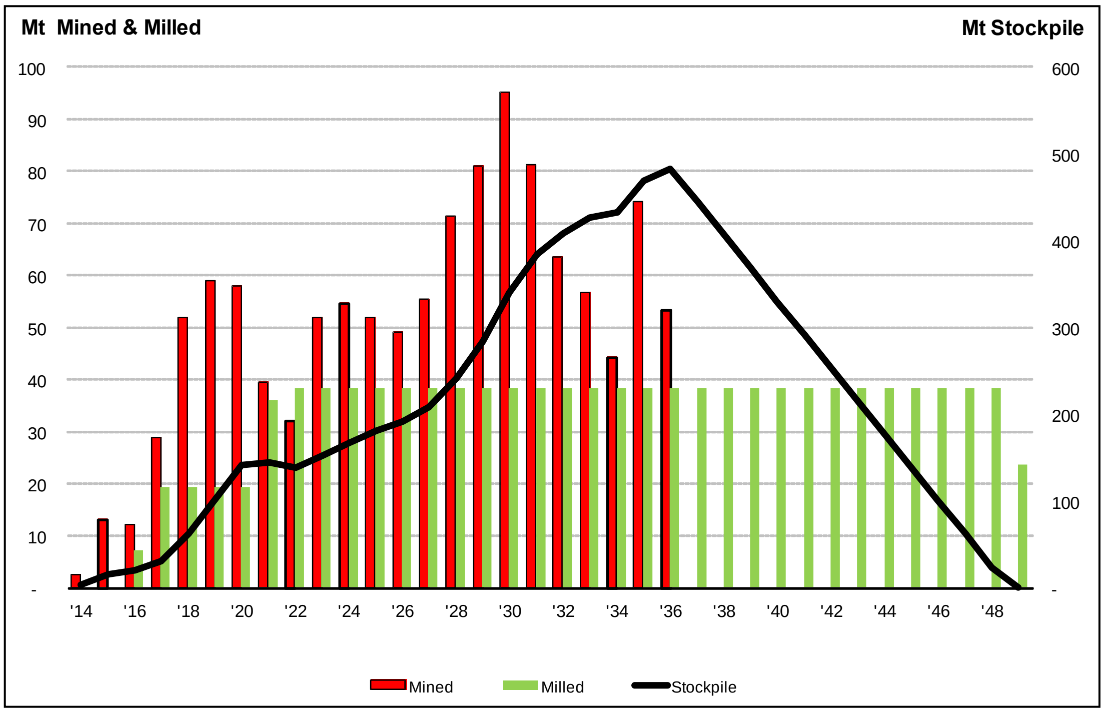
Source: RNC

16.3.6 Low-grade Ore Stockpiles

A key component of the mine plan is the accelerated mining of ore from the pit, with higher value ore being fed directly to the mill and lower value material being temporarily stockpiled. During the life of pit, a total of 606 Mt will be loaded to the low-grade stockpiles. The philosophy of ensuring that the mill is fed with the highest value ore available results in 103 Mt of low-grade material being reclaimed while the pit is still active using either a front end loader (during the first year of operation) or production excavator (Years 2 to 20). The remaining 503 Mt will be reclaimed after pit closure, using rope shovels. Reclaiming the low-grade material extends the life of project until 2049, for a total of 33 years mill production (see Figure 16.32).

The strategy of accelerated mining has the additional advantage of creating a void (i.e., the mined-out open pit), which would accommodate approximately 498 Mt or 43% of the total tailings produced, thus reducing the surface footprint of operations.

Figure 16.32: Mill Production & Low-grade Stockpile

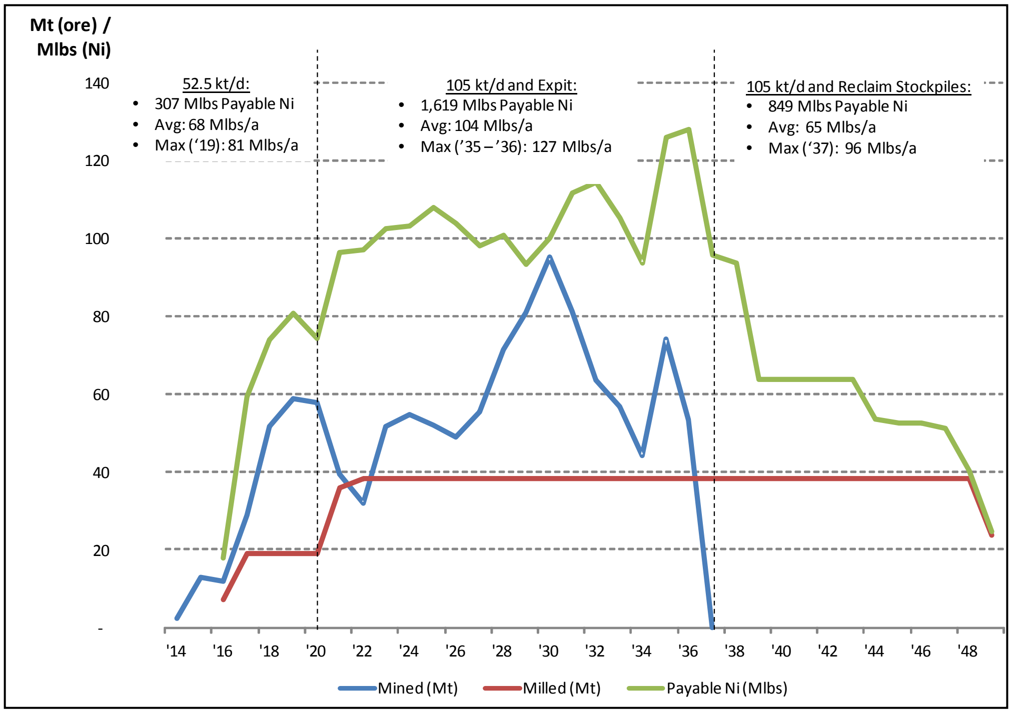


Source: RNC.

The strategy of stockpiling lower-value material allows the value of material treated during the initial years to be maximized. As a result, annual output averages 68 Mlbs payable Ni during the first 4.5 years of production (2016 to 2020) when the concentrator throughput is 52.5 kt/d. Maximum output during this time is 81 Mlbs in 2019.

After throughput is increased to 105 kt/d, output increases to an average of 104 Mlbs payable Ni (maximum is 127 Mlbs) for the period 2021 to 2036 when the pit is active. After the pit is depleted and processing of lower grade stockpiles commences in July 2036, output drops to an average of 65 Mlbs payable Ni, as shown in Figure 16.33.

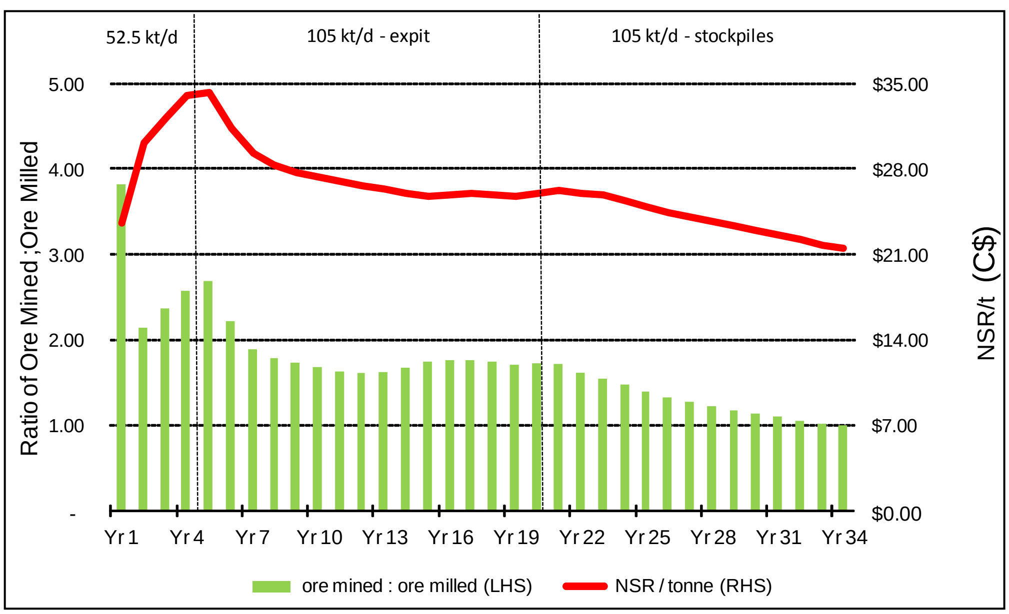
Figure 16.33: Ore Mined + Milled & Ni Output



Source: RNC.

Figure 16.34 illustrates the cumulative NSR value of ore treated through the mill as a function of the accelerated release of ore from the pit.

Figure 16.34: Cumulative Value of Ore Milled vs. Ex-pit Ore Release



Source: RNC.

As a result of the low strip-ratio material mined during the initial 4.5 years when the milling rate will be 52.5 kt/d, the mine plan releases 2.3 tonnes of ore for every tonne milled and the value of ore treated reaches $35/t by the time the expansion is commissioned. Following expansion and the move into higher stripping ratio areas of the pit, the ratio of ore mined to ore milled drops to 1.70, with a commensurate drop in value of ore mined to $26/t. Excluding the higher value material during the initial 4.5 years, the ratio of ore mined to ore milled during the expansion years is 1.6 and the value of ore treated is $25/t.

Material impounded in the low-grade stockpiles will range in value from $27/t down to the cut-off of $7/t, with the overall average being $15/t. As a result of this range in value, low-grade ore will be located in three distinct dumps described below:

- LGO3, which is a small dump with maximum capacity of 12 Mt, will be located closest to the crusher and within the final pit shell. This will be used to impound higher grade material in the initial years of mining and will be depleted by the end of Year 6. A total of 20 Mt of average value $21/t passes through this dump.
- LGO2, which is a larger dump (maximum capacity of 93 Mt), will also be located close to the crusher and used for impounding higher value material. Approximately 50% of the 176 Mt ore tipped on this dump will be reclaimed while the pit is still active. The remainder will be reclaimed before any material is reclaimed from LGO1. The total transiting through this dump will be 176 Mt of average value $20/t.
- LGO1 is the largest low-grade stockpile, with a capacity of 410 Mt. This dump will be sub-divided into three or more areas based on value of ore. Higher value ore (198 Mt of average value $14/t) will be reclaimed first, followed by intermediate value (145 Mt averaging $12/t). The lowest value material (67 Mt averaging $9/t) will be treated last.

Note that Figure 16.36 illustrates the relative location of the various stockpiles and a typical cross-section.

The design and capacity of stockpiles has been based on geotechnical parameters and requirements of the mine plan. The manner in which the stockpiles will be operated, including the division of ore stockpile 1 (LGO1) in three sub-areas and the exact sequence for reclamation, will also be governed by operating parameters that will be established once production commences.

16.3.7 Waste Dumps

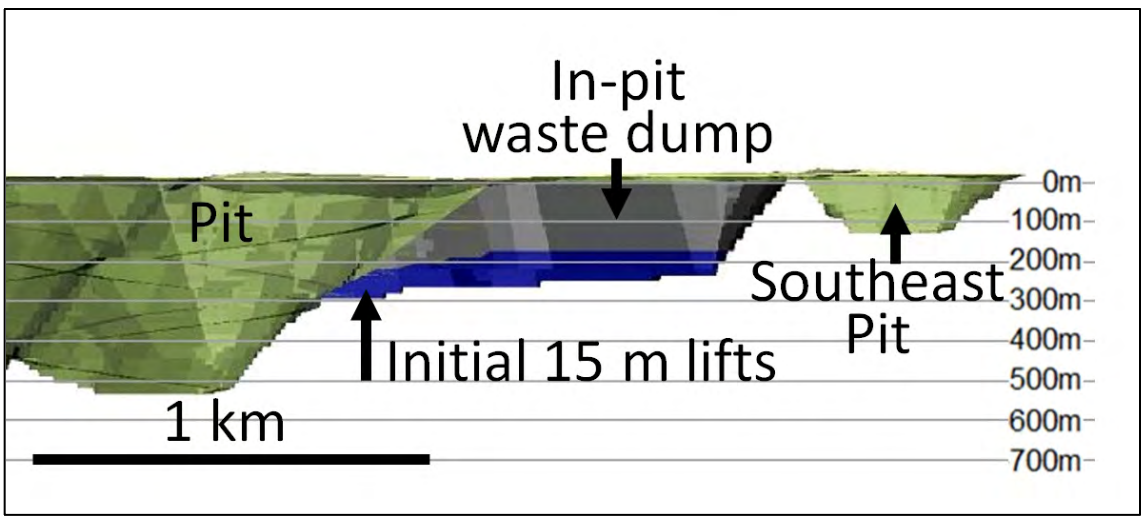
The 2,514 Mt excavated from the pit includes 1,179 Mt ore, along with waste materials comprising 50 Mt clay, 126 Mt overburden that is mainly sand and gravel, and 1,159 Mt waste rock. In addition to the 50 Mt of clay excavated from the pit, a further 13 Mt will be excavated from the key trench beneath the TSF dam wall for a total of 63 Mt. As discussed in Section 16.2.4 previously, there are two forms of clay at Dumont. Brown clay, which typically extends to a depth of 2 m, can be used for construction of the clay core in the TSF and for reclamation of dumps at the end of pit life and will thus not be impounded in waste dumps. Brown clay is estimated to total 9 Mt, including 5 Mt from the pit and 4 Mt from the key trench. The remaining 54 Mt grey clay (45 Mt from the pit and 9 Mt from the key trench) has no productive use and will be impounded in cells constructed out of sand and gravel overburden and/or waste rock. Cells will measure 200 m by 200 m in plan view and will be raised in four (4) lifts of 5 m. Approximately 75% of these cells will be contained within the larger overburden dump 1 (OB1) that is located centrally on the hanging wall side of the pit. A single clay cell will be contained within the smaller overburden dump 2 (OB2) at the southeast extremity of the property. The remainder will located within the southeast extremity of the main waste rock dump (WR1).

Approximately 17% of the remaining non-clay overburden (including organics, till and sand and gravel) will be used for TSF construction or reclamation of dumps. Of the remaining 105 Mt, approximately 74% will be impounded in OB1 (including that used for construction of the clay cells). The remainder (approximately 26 Mt) will be impounded in overburden dump 2 (OB2), which is a much smaller dump located at the far south-east end of the property and will serve to mitigate the impact of noise from the operation on communities to the east of the property. OB2 will have an approximate height of 40 m and will be constructed in 6 lifts, with the initial 4 lifts having a height of 5 m followed by two lifts of 10 m.

Approximately 20% of total waste rock will be used for construction of the TSF and roads, including roadstone that will be used to continually re-surface roads. Of the remaining 940 Mt waste rock, approximately 103 Mt will be impounded along with sand and gravel and clay in OB1. The combined tonnage of clay, sand and gravel, and rock for this impoundment will be 225 Mt and it will extend approximately 3.4 km along strike and to an approximate height of 40 m (as with OB2, it will be constructed in 6 lifts of either 5 m or 10 m). To minimize haulage distances, OB1 will be accessed by 4 separate ramps. The northern and southernmost will be aligned with the hanging wall north (HW-N) and hanging wall south (HW-S) pit exits, with the remaining two spaced evenly between.

A further 10% of waste rock (114 Mt) will be impounded within the pit, after mining in the SEE is completed. This dump will be constructed in two phases. The initial phase of approximately 12 Mt, contained within eight lifts of 15 m each, will be established in a bottom-up sequence (the blue levels in Figure 16.35). With this initial phase in place to act as a catchment, the remaining 102 Mt will be tipped from the top (grey in Figure 16.35) to achieve an overall face slope of 1.5H:1V.

Figure 16.35: In-pit Waste Rock Dump



Note: Looking North. Initially eight 15 m lifts shown in blue are placed from the bottom of the pit at an overall slope of 2.25:1. Subsequently, lifts are dumped from the top down at an overall slope of 1.5:1. **Source:** RNC.

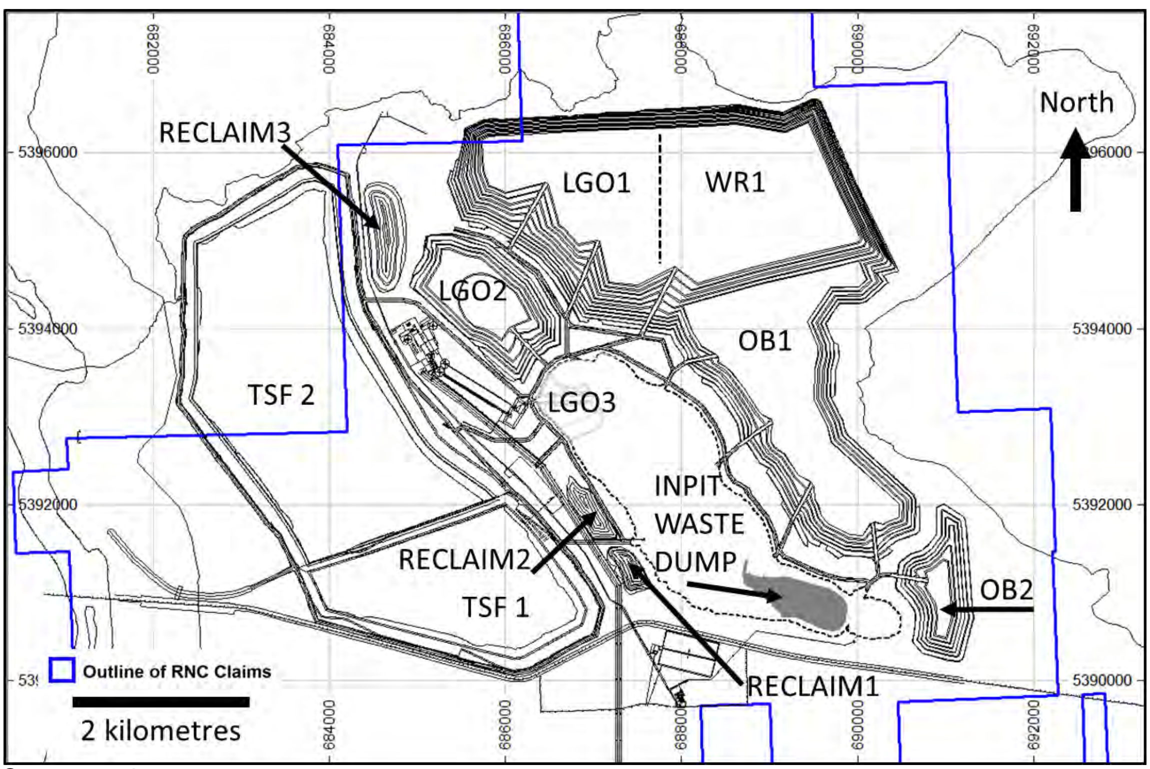
The majority of waste rock (723 Mt) will be stored in WR1, which is located between OB1 and LGO1 (Figure 16.36). With the compacted bulk density of 2.15 t/m^3, this dump will occupy 336 Mm^3 (the design allows for 5% extra capacity, or 353 Mm^3 total). It will be constructed in 11 lifts and have an approximate height of 85 m. Given the dump will be constructed on top of overburden, to ensure stability the initial lift will be only 2 m and will be followed by 4 lifts of 5m each. Additionally, the pit-facing slopes will be a relatively flat 6H:1V, compared to 3H:1V used on slopes not facing the pit. It may prove possible to improve the design (i.e., reduce associated operating costs) by steepening dump slopes and/or depositing material in greater lift heights initially – especially given that initial rate of deposition will be moderate, with delivery of the 228 Mt contained in the lower 5 lifts only completed after Year 10, for an average deposition rate of 22 Mt/a. The remaining 6 lifts will each be 10 m in height and will be deposited an average rate of 52 Mt/a.

The boundary between WR1 (which will be a permanent impoundment) and LGO1 (which will be reclaimed) will not be vertical but follow the 3H:1V final slope of the dumps. This face will be reclaimed following the end of stockpile reclaim operations. All other dump faces will be reclaimed during normal operations, as soon as the lift is complete. In addition to mitigating any environmental issues, early reclamation will allow maximum delivery of reclaim material (either brown clay or organic overburden) from ROM operations rather than more costly stockpiling and subsequent rehandle.

The calculation of haulage distances and associated cycle times assumes that each lift of the dump will be tipped to completion before the next lift is initiated. In practice, the dump would likely operate on multiple lifts simultaneously, which would serve to defer longer hauls and have a positive impact on the NPV of the operation.

Figure 16.36 provides a plan view of the various dump and stockpiles (including temporary stockpiles of material that will be used for reclaiming dumps and the TSF).

Figure 16.36: Layout of Dumps & Stockpiles



Source: RNC

16.3.8 TSF

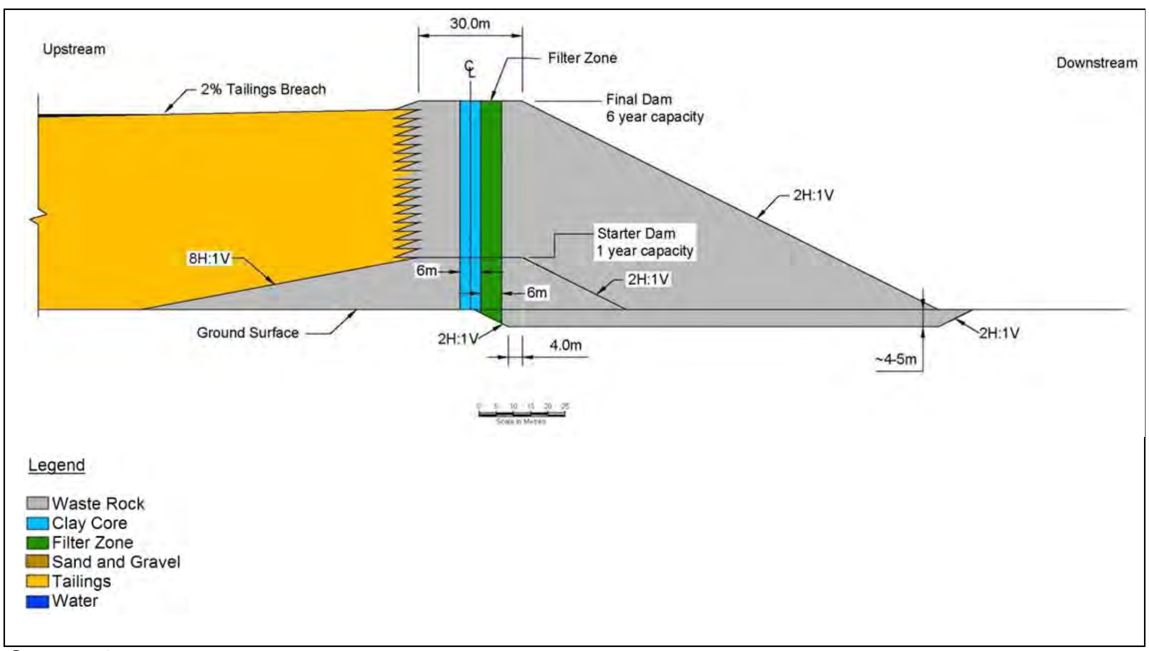
The TSF will be constructed using waste rock, sand and gravel, and clay excavated from the pit. Both TSF1 and TSF2 will be built in two stages as summarized in Figure 16.37:

- The starter dams will be constructed prior to delivery of any tailings to the impoundment. Both TSF1 and TSF2 starter dams will be built to a nominal height of 15 m (varies along the perimeter due to topography), which will be sufficient to store approximately one year of tailings production. The starter dams will be comprised of a 30 m wide core, an upstream component with a slope of 6H:1V and a downstream component with slope of 2H:1V. The core will include a 6 m wide clay zone (with very low permeability) and adjacent 6 m wide filter (constructed using sand and gravel). The remainder of core and both upstream and downstream components will be constructed using waste rock.
- After deposition of tailings commences, the dams will be raised using the centre-line methodology, where the upstream component toes into the tailings surface, while the downstream component continues to be tipped at a slope of 2H:1V. The rate of rise will be approximately 6 m/a for TSF1 and 4 m/a for the larger TSF2.

The dam would be constructed by first excavating any clay that underlies the dam wall and backfilling this excavation (the key trench) with sand and gravel stripped from the pit. Rock used in the construction of the TSF will be delivered on a 12-month basis, and will be sourced entirely from ROM operations (no stockpiling required). To allow rock to be delivered with cost-effective 230 t class trucks, the central core will be tipped solidly with rock. The TSF will be preferentially constructed using gabbro and basalt, but peridotite and dunite are also acceptable.

The clay core and filter layer would be constructed on a six-month basis (during warm months, when clay was soft enough to be handled) and peak requirements will exceed instantaneous ROM output, resulting in some stockpiling and rehandle being required.

Figure 16.37: Typical Cross-section through TSF Dam



Source: SRK

16.3.9 Surface Haul Roads

Pit operations will require the construction of 46.6 km of haul roads on surface, with 12.4 km being temporary and removed as either the pit expands or TSF2 is constructed. The remaining 34.2 km will be permanent. To minimize dust and maximize tire life, roads will be constructed using only gabbro and basalt rock types. Additionally, allowance has been made to cover all roads (including ramps in the pit and roads on dumps) with 50 mm of roadstone annually, resulting in production of approximately 40 Mt of roadstone over the life of mine.

All main haul roads will be 35 m wide, which is suitable for the 230 t class trucks planned for use. Roads will be filled with a minimum of 2 m at the centreline and sloped at -2% to the edges. Roads will be located a minimum of 40 m from the crest of the pit. To minimize dust generated on nearby communities, no haulage roads will be located on the south side of the pit.

16.4 Mining Process Description

16.4.1 Overview

Ex-pit mining operations at Dumont will be conducted by the following fleets of production mining equipment (in order of lithology that will be mined):

- Clay will be mined using small hydraulic excavators with 7 m^3 dippers (nominal 12 t payload) and 55 t payload rigid body haul trucks. No drilling and blasting will be required.
- The bulk of sand and gravel below the clay layer will be mined using large diesel-powered hydraulic excavators with 34 m^3 dippers (nominal 60 t payload) and 230 t payload rigid body haul trucks. No drilling and blasting will be required. The bench height will be 10 m.
- At the interface between rock and sand and gavel, rock will be loaded and hauled using the same size equipment as will be used for clay. Rock will be drilled using percussion drills with a nominal hole diameter of 102 mm on a bench height of up to 5 m.
- Below the sand and gravel interface, rock will be drilled using rotary blast hole units with holes measuring 270 to 311 mm in diameter. The bulk of rock will be loaded using large electric rope shovels with 43 m^3 dippers (nominal 75 t payload) though some rock will be mined using the 34 m^3 hydraulic excavators. All rock will be hauled using 230 t payload rigid body haul trucks. A bench height of 10 m will be used on any bench within some occurrence of sand and gavel. Below this horizon, benches will have a height of 15 m as per the pit design.

Production equipment will be supported by various units of support equipment, including tracked dozers, wheel dozers, front end loaders, graders, water tankers and utility excavators.

The bulk of the mining fleet will be purchased and operated by the Owner. The duty cycle for production units was estimated by first principles, based on the production plan.

Norascon, a local mining contractor with experience operating in similar environments has been pre-selected to assist in the mining operation. The contractor will operate mainly a fleet of small hydraulic excavators and 55 t trucks and will be tasked with performing the following:

- establish the initial working faces for the Owner's large hydraulic excavators;
- mine all clay and rock – sand and gravel contact material for the initial five years of operation (including 22 months of pre-strip and subsequent 38 months); and
- following the initial five-year period, the contractor will supplement the Owner's clay mining fleet as required according to the mine plan.

The following infrastructure would be provided to support mining activities:

- workshop and associated warehouse; equipment will be maintained under a maintenance contract initially, with a phased hand-over to in-house personnel as experience is gained;
- fuel farm and associated fuelling bays;
- explosives manufacture facility and magazine; as is the norm in Canada, this will be operated by the explosives supplier;
- in-pit sump and associated dewatering system; and
- electrical reticulation system.

The Owner's mining labour complement will average 331 persons during the life of the project, reaching a peak of 650 persons while the pit is active then dropping to an average of 116, while the low-grade stockpile is being reclaimed. The mining contractor workforce will average 95 persons over the eight years that the contractor will be active, with a peak of 178 persons in the early years.

16.4.2 Mining Fleet

Fleet sizes were based on the following assumptions:

- The mine will operate 24 hours per day, 365 days per year.
- The mechanical availability and operator utilization of equipment would vary according to the particular unit of equipment. Average annual engine hours (product of availability and utilization) for the main production equipment would range from a high of 7,000 (cable shovels) to 6,300 (230 t haul trucks) to 4,900 (diesel-powered percussion drill).
- An efficiency factor of 90% was applied to utilized time, meaning that 10% of total engine hours (incurring costs) would not be directed towards completing useful work.

Table 16-6 and Table 16-7 summarize the main units of the mining fleet that will be used by the Contractor and Owner, while Tables 16-8 and Table 16-9 summarize the size of mining fleet by year over the life of the project during ex-pit operations and stockpile reclaim periods, respectively. Examples of the specific fleet units have been provided for reference, but these do not in any way indicate that a decision has been made on the actual Original Equipment Manufacturers (OEMs) that will be selected to supply equipment to the project. The OEMs will be selected following a competitive tendering process.

There is some overlap between Contractor and Owner fleets (e.g., the Contractor fleet of small rigid body trucks would peak at 21 units and would average 19,000 engine hours or ~20% of their economic life at the end of the contract when they would thus be available for sale to the Owner). However, the capital cost estimates given in Section 21 do not include any synergies between the two fleets; all Contractor equipment is assumed to be leased at a rate reflecting the current purchase cost of a new machine while all Owner equipment is assumed to be purchased new.

Table 16-6: Dumont Mining Fleet – Contractor Equipment

Process	Unit	Application	Size	Example
Drilling	Percussion Drill	Rock at sand and gravel interface	102 mm hole	Sandvik DX800
Loading	Small Diesel Excavator - 1 Small Diesel Excavator - 2	Rehandle Overburden (Clay + sand and gravel) Ex-pit Clay + Rock at sand and gravel interface	4.5 m^3 bucket (8 t) 7 m^3 bucket (12 t)	Caterpillar 390 Komatsu PC1250
Hauling	Articulated Truck Small Rigid Body Truck	Rehandle Overburden (Clay + sand and gravel) Clay + Rock at sand and gravel interface	36 t payload 55 t payload	Caterpillar 740 Caterpillar 773
Support Equipment	Small Front End Loader Small Track Dozer Small Grader Small Water Tanker Small Utility Excavator	Clean-Up / Secondary Loading Clean-Up Maintain Roads Dust Suppression General Construction	11 t payload 8 m^3 blade 4.2 m blade (14ft) 35 m^3 capacity 6 m boom	Komatsu WA600 Komatsu D155 Komatsu GD655 Caterpillar 735 Komatsu PC 490

Table 16-7: Dumont Mining Fleet – Owner Equipment

Process	Unit	Application	Size	Example
Drilling	Percussion Drill	Rock at sand and gravel interface	102 mm hole	Sandvik DX800
	Diesel Rotary	Rock (pre- grid power & sinking new benches)	270 mm hole	Sandvik D90
	Electric Rotary	Rock (bulk mining)	311 mm hole	P&H 320 XPC
	Down-The-Hole Hammer	Pre-Splitting	165 mm hole	Sandvik DI550
Loading	Large Utility Excavator	Rehandle Overburden (Clay + sand and gravel)	4.5 m^3 bucket (8 t)	Caterpillar 390
	Small Diesel Excavator - 2	Clay + Rock at sand and gravel interface	7 m^3 bucket (12 t)	Komatsu PC1250
	Large Diesel Excavator	sand and gravel + Rock (pre- grid power, sinking new benches, LGO)	34 m^3 bucket (60 t)	Hitachi Ex 5600
	Electric Rope Shovel	Rock (bulk mining)	43 m^3 bucket (75 t)	P&H 4100 XPC
Hauling	Articulated Truck	Rehandle Overburden (Clay + sand and gravel)	36 t payload	Caterpillar 740
	Small Rigid Body Truck	Clay + Rock at sand and gravel interface	55 t payload	Caterpillar 773
	Large Rigid Body Truck	sand and gravel + Rock	230 t payload	Caterpillar 793
Support Equipment	Large Front End Loader	Support Large Excavator & Rope Shovel	35 t payload	Komatsu WA1200
	Large Track Dozer	Support Large Excavator & Rope Shovel	18 m^3 blade	Komatsu D375
	Wheel Dozer	Road Clean Up	8 m^3 blade	Caterpillar 834
	Large Grader	Maintain Permanent Roads (large fleet)	4.8 m blade (16 ft)	Caterpillar 16M
	Large Water Tanker	Dust Suppression	130 m^3 capacity	Komatsu HD 1500
	Large Utility Excavator	Trenching & Construction + Scaling Final Walls + Portable Rock Breaker	10 m boom	Komatsu PC 800

Table 16-8: Dumont Mining Fleet by Year during Ex-pit Operations

		Ex-pit Operations																						
Contractor Fleet	**example**	**'14**	**'15**	**'16**	**'17**	**'18**	**'19**	**'20**	**'21**	**'22**	**'23**	**'24**	**'25**	**'26**	**'27**	**'28**	**'29**	**'30**	**'31**	**'32**	**'33**	**'34**	**'35**	**'36**
Percussion Drill	Sandvik DX800	1	2	2	2	2	1	0	1	1	1	1	1	0	0	0	0	0	0	0	0	0	0	0
Small Diesel Excavator	Komatsu PC1250	1	3	5	4	2	1	0	1	1	1	1	1	0	0	0	0	0	0	0	0	0	0	0
Small Rigid Body Truck	Caterpillar 773	3	13	20	16	7	3	0	2	4	2	3	4	0	0	0	0	0	0	0	0	0	0	0
Small Front End Loader	Komatsu WA600	1	1	1	1	1	1	0	1	1	1	1	1	0	0	0	0	0	0	0	0	0	0	0
Small Track Dozer	Komatsu D155	1	2	3	3	2	1	0	1	1	1	1	1	0	0	0	0	0	0	0	0	0	0	0
Small Grader	Komatsu GD655	2	2	2	2	2	1	0	1	2	1	1	1	0	0	0	0	0	0	0	0	0	0	0
Small Utility Excavator	Komatsu PC 490	1	1	1	1	1	1	0	1	1	1	1	1	0	0	0	0	0	0	0	0	0	0	0
Small Water Tanker	Caterpillar 735	2	2	2	2	2	1	0	1	2	1	1	1	0	0	0	0	0	0	0	0	0	0	0
		Ex-pit Operations																						
Owner Fleet	**example**	**'14**	**'15**	**'16**	**'17**	**'18**	**'19**	**'20**	**'21**	**'22**	**'23**	**'24**	**'25**	**'26**	**'27**	**'28**	**'29**	**'30**	**'31**	**'32**	**'33**	**'34**	**'35**	**'36**
Percussion Drill	Sandvik DX800	0	0	0	0	0	1	2	2	2	2	2	2	1	2	2	2	2	1	1	2	1	0	0
Diesel Rotary	Sandvik D90	1	2	2	2	2	1	1	2	3	3	3	2	1	2	2	2	1	1	1	1	0	0	0
Electric Rotary	P&H 320 XPC	1	1	1	1	2	3	2	2	2	3	3	4	4	4	4	4	4	4	4	4	4	4	2
Down-The-Hole Hammer	Sandvik DI550	0	0	0	0	0	0	0	0	0	1	2	2	2	2	2	2	2	2	2	2	2	2	0
Small Diesel Excavator	Komatsu PC1250	0	0	0	0	0	1	2	2	3	3	3	2	1	2	2	2	2	1	1	1	1	0	0
Large Diesel Excavator	Hitachi Ex 5600	1	1	2	2	2	1	1	2	2	2	2	2	1	1	1	1	0	1	1	1	1	1	1
Electric Rope Shovel	P&H 4100 XPC	0	0	0	1	2	2	2	2	3	3	3	4	4	4	4	4	4	4	4	4	4	4	3
Articulated Truck	Caterpillar 740	2	2	2	3	2	4	1	3	11	8	0	0	1	1	1	1	1	1	1	2	2	1	0
Small Rigid Body Truck	Caterpillar 773	0	0	0	0	0	6	11	11	18	11	13	12	7	10	7	8	7	4	6	4	1	0	0
Large Rigid Body Truck	Caterpillar 793	6	12	13	19	27	30	26	28	44	52	55	56	59	57	57	63	63	68	64	61	62	72	40
Large Front End Loader	Komatsu WA1200	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Large Track Dozer	Komatsu D375	1	1	1	2	2	2	2	2	3	3	3	3	3	3	3	3	2	3	3	3	3	3	2
Wheel Dozer	Caterpillar 834	1	1	1	2	2	2	3	3	4	4	4	4	3	4	4	3	3	3	3	3	3	3	2
Large Grader	Caterpillar 16M	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2
Large Water Tanker	Komatsu HD 1500	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2
Large Utility Excavator	Komatsu PC800	3	3	3	3	3	4	3	3	5	4	3	3	3	3	3	3	3	3	3	3	3	2	2

Table 16-9: Dumont Mining Fleet by Year during Stockpile Reclaim

		Stockpile Reclaim												
Owner Fleet	**example**	**'37**	**'38**	**'39**	**'40**	**'41**	**'42**	**'43**	**'44**	**'45**	**'46**	**'47**	**'48**	**'49**
Electric Rope Shovel	P&H 4100 XPC	2	2	2	2	2	2	2	2	2	2	2	2	1
Large Rigid Body Truck	Caterpillar 793	8	8	11	11	11	11	11	9	10	9	9	10	6
Wheel Dozer	Caterpillar 834	1	1	1	1	1	1	1	1	1	1	1	1	1
Large Grader	Caterpillar 16M	1	1	1	1	1	1	1	1	1	1	1	1	1
Large Utility Excavator	Komatsu PC 800	1	1	1	1	1	1	1	1	1	1	1	1	1
Large Water Tanker	Komatsu HD 1500	1	1	1	1	1	1	1	1	1	1	1	1	1

16.4.2.1 Production Drilling & Blasting

Geotechnical parameters for the various Dumont rock types were input into simulations by two different explosives suppliers (Dyno-Nobel and Orica) to predict the required powder factor. These simulations indicated that an acceptable particle size distribution for all rock types could be achieved with a powder factor of 0.25 kg/t. Typical blast patterns are approximately 8 m x 8 m on 10 m benches and approximately 10 m x 10 m on 15 m benches, yielding a weighted average of approximately 3,400 t per hole drilled. In a typical year following the expansion to a milling rate of 105 Mt/a, mining of 140 Mt rock results in drilling of approximately 40,000 holes with approximately 590,000 m of drilling.

The same data regarding rock properties was provided to the suppliers of rotary blast hole drills. Based on feedback from these OEMs, the following instantaneous penetration rates were estimated:

- 102 mm percussion:
  - basalt and gabbro = 50 m/h
  - dunite and peridotite = 60 m/h
- 270 mm and 311 mm rotary:
  - basalt and gabbro = 35 m/h
  - dunite and peridotite = 42 m/h

As dunite and peridotite combined represent 72% of rock that will be mined and 98% of all rock will be drilled with rotary drills, the life of mine average penetration rate is 41.8 m/h. The calculation of total drill hours also includes the following:

- an allowance for re-drilling 3% of holes;
- delays for moving between holes of 5 min for percussion drills and 7.5 min for rotary drills; and
- an allowance for moving between patterns equal to 15% total operating time.

After including the operator efficiency factor of 90%, rotary drill productivity is estimated at 23.3 m/h. With this productivity, the production plan can be achieved with a fleet that reaches a maximum strength of four electric rotary units, supplemented by two diesel rotary units and two percussion drills. At the end of mine life, the average age of the four units' electric rotary machines is 48,000 hours, while the diesel rotary units will have each reached approximately 35,000 hours. This is less than the economic life of these units (120,000 and 60,000 hours for the electric and diesel units, respectively), and no replacement machines would be required.

Similarly, the Owner's two percussion units will have achieved approximately 12,000 engine hours or 20% of their economic life. There is an opportunity to reduce the capital cost of these machines by purchasing one or more of the Contractor's units, which will have reached ~8,000 hours each at the end of the contract.

16.4.2.2 Pre-Splitting

The mine design assumes that all final walls will be pre-split. The design of pre-split blasts was based on simulations performed by Dyno-Nobel, which indicated that with 20 kg of explosive placed in a 15 m x 165 mm diameter hole, the following hole spacing by rock type would be required: basalt = 1.75 m, gabbro = 1.85 m, peridotite = 2.22 m, dunite = 2.67 m. A weighted average of 2.39 m was then estimated, based on the volumes of the four different rock types.

The total pre-splitting requirement was based on an estimated 407 km of total final wall perimeter over the 39 benches that would be mined. The resulting 171,000 presplit holes would require 2,729 km drilling. Pre-splitting was assumed to start in Year 8, when the initial final walls are established in the southeast extension. The total metres of pre-splitting was then evenly divided by the remaining duration of ex-pit mining (with pre-splitting assumed to be finished 6 months before the final production blast), resulting in monthly rates of 1,145 holes or 18.3 km.

Pre-split drilling would be accomplished by a fleet of percussion rigs equipped with down-the-hole hammers. Similar penetration rates as estimated for the 102 mm percussion drill have been assumed above and a fleet of two units would be required.

16.4.2.3 Loading & Hauling

A trade of study conducted during the PFS determined that the optimal match of load and haul equipment at Dumont would be the largest class of rope shovel (≥ 40m3 dipper and payload of up to 110 tonnes) loading 230 t payload trucks in three passes. During the FS, this analysis was updated using the most up to date cost and productivity data from all the OEMs potentially able to supply equipment to the project. The earlier PFS findings were confirmed as:

- The ratio of tare weight-to-payload for trucks larger than 230 t payload is generally inferior, resulting in slower uphill travel speeds and increased diesel consumption per tonne transported. Additionally, larger trucks require wider ramps and thus flatter slope angles with more waste stripping. Wages in the Abitibi are such that the savings in labour resulting from use of larger trucks does not offset these two factors and trucks larger than the 230 t class cost more to operate on a unit basis.
- There is very little savings in capital costs or the hourly operating cost for a smaller class of rope shovels that are equipped with 34 m^3 dippers that can load a payload of up to 60 t. This class of shovel would require four passes to load the 230 t truck, which would necessitate the purchase of additional units to achieve the Dumont mine plan.

Note that while the large rope shovel that has been selected could theoretically load a 230 t truck in two passes, there is a large body of anecdotal evidence suggesting this has detrimental effects on both truck life and truck driver health. Consequently, a more conservative three-pass arrangement has been assumed. Furthermore, loading design criteria (Table 16-10) assume that 25% of trucks would be loaded in four passes to reflect likely variability in loading conditions. In the event that two-pass loading were feasible, the impact to the overall project would be to improve post-tax NPV by approximately 3%.

The excavators are not fully utilized, with the FS mine plan resulting in average production of 3.3 and 10.8 Mt/a for the clay and rock units, respectively (77% and 42% of theoretical productivity). The rope shovels will be utilized closer to their maximum, with average production of 35.4 Mt/a.

Criteria used to calculate the productivity of various hauling units are given in Table 16-11. These have been provided by OEMs based on the rimpull curves of the different units under consideration, and assume 10% ramp gradients and 2% rolling resistance. Calculated values such as cycle time, tramming speed and productivity are primarily a function of the haulage profile.

Table 16-10: Loading Design Criteria

Material Excavated		**Clay[5] Excavator**	**Sand & Gravel + Rock Excavator**	**Sand & Gravel + Rock Rope Shovel**
Example Unit		Komatsu PC 1250	Hitachi Ex 5600	P&H 4100
Average Bucket Factor[1,3]	tonnes	12.5	61.4	76.7
Average Truck Payload[2,3]	tonnes	54.1	230.1	230.1
Theoretical Passes per Load	number	4.30	3.70	3.00
Additional Passes per Load	number	0.25	0.25	0.25
Total Passes per Load	number	4.55	3.95	3.25
Cycle Time per Bucket	seconds	40	35	28
Spot Time	seconds	30	30	30
Total time to Load Truck	seconds	212	168	121
Engine hrs per year	hours	6,000	6,500	7,000
Non-productive time per year[4]	hours	1,410	1,528	1,960
Theoretical productivity per unit	**Mt/a**	**4.3**	**25.9**	**35.9**

Notes: 1. 95% of rated 2: 97.5% of rated, 3. assumes load monitoring systems would be utilized, 4: includes non-productive utilized time of 10% and equipment moves, blast delays etc of 15% for excavator and 20% for rope shovel. 5. includes rock at Sand & Gravel interface.

Table 16-11: Hauling Design Criteria

Material Hauled		**Clay[3]**	**Sand & Gravel + Rock**	**Sand & Gravel + Rock**
Example Unit		Cat 773	Cat 793	Cat 793
Loaded with:		Excavator	Excavator	Rope Shovel
Payload	tonnes	54.1	230.1	230.1
Loading Time	min / load	3.53	2.80	2.02
Dumping Time	min / load	2	2	2
Queuing Time	min / load	2	2	2
Speed - in-pit flat (empty & full)	km/h	20	20	20
Speed - ex-pit normal flat (empty & full)	km/h	30	35	35
Speed - ex-pit TSF flat (empty & full) [1]	km/h	15	15	15
Speed - uphill loaded	km/h	12	13.7	13.7
Speed - downhill empty	km/h	35	37.5	37.5
Average Cycle Time	**min / load**	**29.3**	**33.2**	**33.2**
Average Speed [2]	**km/h**	**14.1**	**17**	**17**
Average Fuel Burn[2]	**L/h**	**40.8**	**181.4**	**181.4**
Average productivity per unit [2]	**Mt/a**	**0.6**	**2.6**	**2.6**

Notes: 1. Slower speed travel on top of TSF dam wall, 2. Average for all large trucks (loaded by excavator and rope shovels), 3. includes rock at sand & gravel interface.

With the parameters given in Tables 16.10 and 16.11, the Owner's clay fleet (including rock mined at the sand and gravel interface) peaks at 3 excavators and 18 trucks (supplemented by Contractor fleet of up to 5 excavators and 20 trucks to handle peaks in the schedule). Use of this fleet over the entire mine life averages 39,000 hours for excavators (compared to an economic life of 60,000 hours) and 41,000 hours for the trucks (100,000 hr life). No replacements are required and there is likely an opportunity to smooth the schedule and rationalize fleet purchases.

The Owner's fleet of large production excavators totals two units that average 44,000 hours at the end of mine life. No replacements of these units are required.

The fleet of rope shovels reaches a maximum of four units, which can log a total of 511,000 hours, resulting in one unit slightly exceeding the economic life of 120,000 hours that has been assumed. The evaluation thus includes purchase of a replacement unit. It should be noted there are instances of rope shovels exceeding 130,000 hours and being planned for 175,000 hours so it may be possible to avoid this purchase.

The fleet of large haul trucks reaches a maximum of 72 units and in aggregate logs 7.1 million hours. While the average of 98,600 hours per machine is less than the economic life of 100,000 hours, the timing of purchases for the existing mine plan is sub-optimal and a total of 16 units are planned for replacement. There may be an opportunity to smooth the production schedule and reduce or eliminate some of these replacement purchases.

A computerised truck dispatching system will be employed to maximize the utilization and efficiency of the haul truck fleet.

16.4.2.4 Support Equipment

Open-pit haul roads and working faces would be maintained with a fleet of support equipment that includes:

- Track dozers, for ripping footwalls and for heavy construction work. The fleet requirements were estimated based on the empirical relationship of 0.5 operating dozers for every operating production loading unit (for both the Contractor and Owner fleets). The contractor fleet would be supported by smaller 8 m^3 blade for example, while the Owner fleet would be supported by an 18 m^3 blade (i.e., class) unit.
- Rubber-tired dozers, for lighter construction and general cleanup. The fleet requirements were estimated based on the empirical relationship of 0.5 operating dozers for every operating loading unit (Owner only). Equipment with an 8 m^3 blade would be utilized.
- Graders. The fleet requirements were estimated to be two graders for the contractor fleet and three graders for the Owner fleet. The contractor fleet would use units with a 14 ft blade, while Owner fleet would utilize larger units with a 16 ft blade.
- Water tankers, for suppressing dust. An allowance was made for two units to support the Contractor fleet and two units to support the Owner fleet. The Owner units would be converted 130 t class haul trucks (130 m^3 capacity) while the Contractor units would be converted 35 t class articulated trucks (35 m^3 capacity).
- Front end loaders, for construction and clean-up activities including the loading of roadstone into trucks. Front end loaders would also be available to supplement to main production fleet, if required. One unit has been assumed for each of the Contractor (11 t payload) and Owner fleets (35 t payload).

- Utility excavators, for construction activities such as would take place at the TSF. The Owner utility excavators would also be equipped with attachments for scaling highwalls and breaking oversize, as required. The Contractor fleet would require a single unit while the Owner fleet would vary in size according to the required duty, with a maximum of 5 machines required.

16.4.3 Infrastructure

16.4.3.1 Workshop

A workshop and associated warehouse would be provided to maintain the fleet of equipment. The size of this workshop was based on the following empirical factors:

- one maintenance bay for every five production trucks; and
- one auxiliary bay for every 12 production trucks.

The workshop is initially six bays at start-up, expanding to 10 bays when the concentrator increases to 105 kt/d. As the fleet continues to increase due to longer hauls, the workshop would ultimately reach 20 bays.

Equipment would be maintained under a maintenance contract initially, with a phased handover to in-house personnel as experience was gained. A more complete description of the workshop is given in Section 18.

16.4.3.2 Fuel Farm / Diesel Bay

Diesel consumption has been estimated from first principles, based on the burn rate for the various pieces of equipment that would be operated and specific duty cycle (in the case of haul trucks, burn rates were estimated for each of the different profiles listed in Table 16-11). Figure 16.38 illustrates that the large (230 tonne) haul trucks operated by the Owner on ex-pit hauls are expected to consume 69% of the 1,500 ML diesel required over the life of mine and a further 11% when rehandle of low-grade stockpiles and construction of the TSF are included. Diesel used as fuel for other Owner mining equipment represents another 13% of the total requirement while diesel used by the Contractor and in explosives is each 2% of the total. The remaining 3% of consumption is by light equipment operated at the Concentrator and by General and Administration (G&A) personnel.

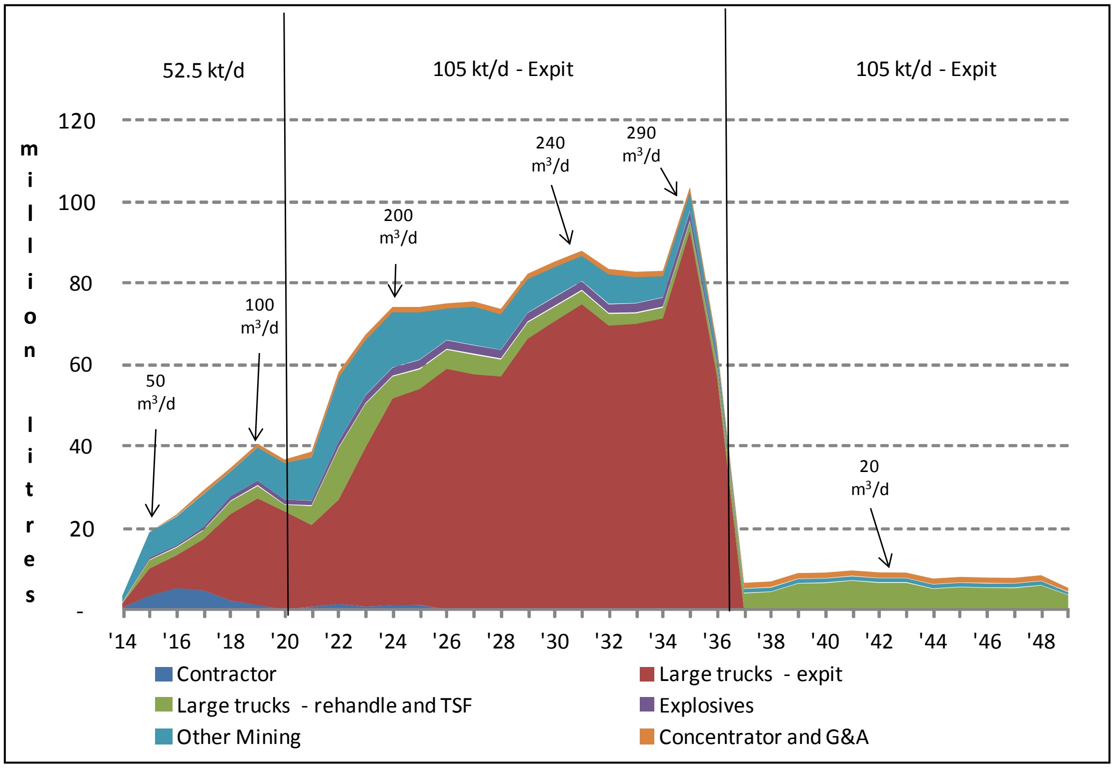
As will be discussed in Section 24, there is an opportunity to use trolley-assist to reduce diesel consumption. A study completed as part of the FS identified the potential to reduce diesel consumption by large trucks on ex-pit uphill hauls by more than 350 ML (or 23% of base case consumption).

For the conventional haulage base case, average daily consumption over the initial period of operation (mill rate at 52.5 kt/d) starts at 50 m^3/d and increases steadily to 100 m^3/d at the time of expansion, with a maximum daily consumption of 122 m^3/d during this time. The fuel farm has been sized to allow for surge, with the initial capacity of 900 m^3 providing over 1 week storage during periods of maximum consumption. With the expansion of mill capacity to 105 kt/d, the fuel farm will be expanded to 1,650 m^3, providing over five days of storage for the year of peak consumption. After the end of ex-pit operations, consumption drops to 20 m^3/d.

Equipment would be fuelled at a diesel fueling station located adjacent to the workshop complex. A modified 140t haul truck would be equipped with a fuel tank to refill equipment in the pit, if necessary.

A more complete description of the fuel farm is given in Section 18.

Figure 16.38: Diesel Consumption



Source: RNC.

16.4.3.3 Explosives Manufacture Plant

Rock will be blasted using emulsion explosives. For the initial 34 months of operation (including the 22-month pre-strip and initial 12 months of mill production), the consumption rate would be low enough (<20 t explosives per day) that explosives could be supplied from one of the following existing plants located within the Abitibi:

- Orica – Plant located at the Canadian Malartic mine, approximately 90 km from the Dumont mine site; or
- Dyno-Nobel – Plant near in Val d'Or, approximately 100 km from the Dumont mine site.

Explosives would be trucked using 14 t bulk delivery trucks that would deliver explosives directly to the blast hole. The daily traffic would thus be 1 to 2 trucks.

By the second year of mill production, the daily consumption of explosives would be sufficient to justify the construction of a plant at the Dumont mine site. Raw materials (including ammonium nitrate prills, ammonium nitrate solution, emulsifier, and diesel) would be non-explosive until combined at the facility and delivered by trucks with payload 37 to 40 t. In line with Canadian regulations, the facility will be located at least 1 km from all infrastructure (buildings, public roads) and 670 m from an active dump.

The explosives manufacture facility will use intellectual property owned by the explosives supplier. In line with North American practices, the facility would thus be owned and operated by the explosives supplier. Based on budgetary quotations provided by Orica and Dyno-Nobel, the financial model assumes that this facility would be rented for the entire life of mine. The decommissioning of this facility at the end of mine life would be the responsibility of the explosives supplier.

16.4.3.4 Roadstone Crusher

To ensure the truck fleet achieves high productivity (including an average life of 8,000 hours for tires on the large haul trucks), roads would be continually re-surfaced with crushed waste rock. Rock would be crushed to a nominal size of 20 mm through a two-stage plant (primary jaw and secondary cone crusher). This is approximately the same size product as would be required for the concrete batch plant during construction, and a single crushing plant would be used for both construction and roadstone.

The duty cycle of the roadstone plant has been based on the following:

- All haul roads will receive 50 mm of crushed material annually (equivalent to two treatments of 25 mm).
- All blast holes will be stemmed using crushed roadstone.
- The total requirement for roadstone has been estimated at 39.3 Mt over the life of project, comprising:
  - 8.6 Mt of roadstone on in-pit roads that will be reloaded as the roads are mined out
  - 30.3 Mt of roadstone on surface roads that will be permanent in nature (not reloaded)
  - 0.4 Mt of roadstone used for blast hole stemming.

Feed to the roadstone plant will be delivered using large haul trucks while crushed roadstone will be loaded into large haul trucks using a front end loader.

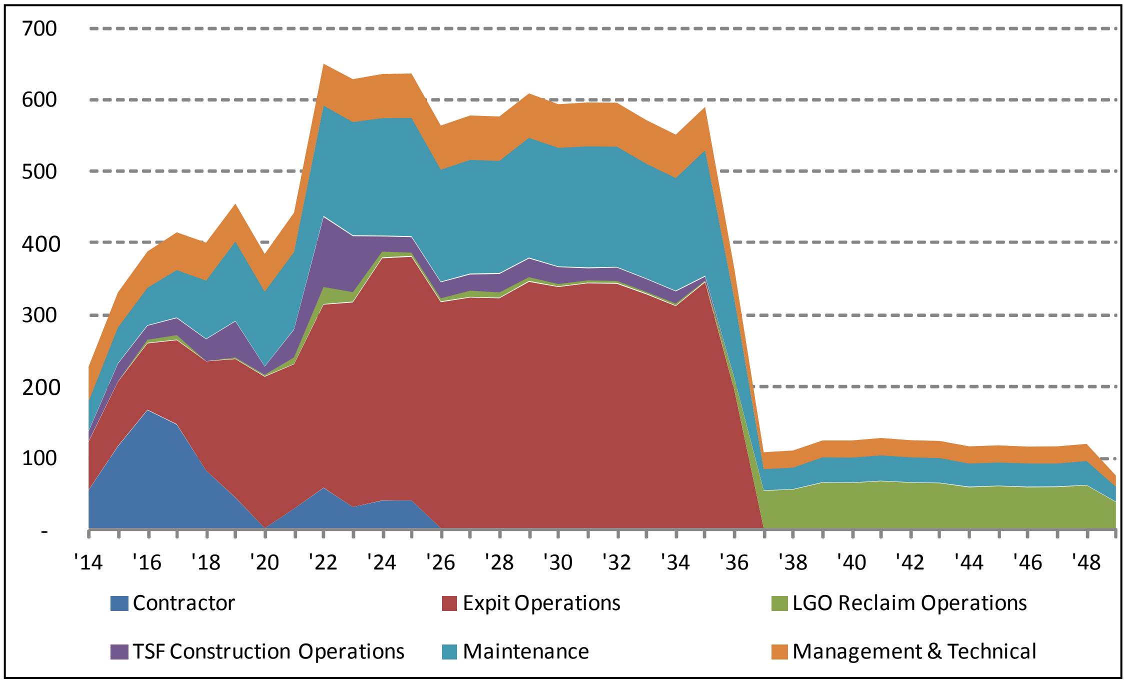
16.4.4 Manpower

The mine will operate continuously, with 2 x 12 hour shifts daily and 365 days per year. This will be achieved by 4 crews each working an average of 42 hours per week and labour costs allow for two hours of planned overtime weekly (in addition to unplanned overtime). Staff personnel will work on a conventional 5-day week schedule.

The life of mine labour complement illustrated in Figure 16.39 was calculated from first principles based on the number of units of equipment required to achieve the planned production schedule. During the initial 4.5 years of commercial production, while the concentrator is operating at 52.5 kt/d, the complement would average 407 persons, including 87 mining contractors.

After the concentrator is expanded to 105 kt/d and the mining rates increase, the complement increases to a maximum of 650 and average 587. The Owner’s labour complement averages 331 persons over the life of project, including 463 while the pit is active and 116 during reclaim of the low grade stockpile. These totals do not include personnel allocated to TSF construction, who would average a further 34 during the 20 years that construction of the TSF is active. The Contractor’s complement will average 95 persons during the 8 years that the Contractor is active.

Figure 16.39: Labour Complement



Source: RNC.

17 RECOVERY METHODS

17.1 General

The process plant and associated service facilities will process ore delivered to primary crushers to produce nickel concentrate and tailings. The proposed process encompasses crushing and grinding of the ore, desliming via hydrocyclone circuit, slimes rougher and cleaning flotation, nickel sulphide rougher and cleaning flotation, magnetic recovery of sulphide rougher tails and sulphide cleaner tailings, regrinding of magnetic concentrate and an awaruiter recovery circuit (consisting of rougher and cleaner flotation stages).

Concentrate will be thickened, filtered and stockpiled on site prior to being loaded onto to railcars for transport to third-party smelters. The slimes flotation tailings, magnetic separation tailings and awaruite rougher tailings will be combined and thickened before placement in the TSF.

The process plant will be built in two phases. Initially, the plant will be designed to process 52.5 kt/d. The expansion will be designed as a duplicate processing plant to increase plant capacity to 105 kt/d. The initial phase will include an allowance for common concentrate thickening facilities.

17.2 Design Criteria Summary

The overall engineering approach was to design robust process plants that could handle a wide range of ore variability and operating conditions. The key project and ore-specific criteria for the plant design and operating costs are provided in Table 17-1.

17.3 Plant Design Basis

The key criteria selected for the base plant (52.5 kt/d) and expansion plant (105 kt/d) designs are:

- nominal base plant treatment rate of 52.5 kt/d and a nominal expansion plant treatment rate 52.5 kt/d for a combined 105 kt/d treatment rate;
- design availability of 92% (after ramp-up), which equates to 8,059 operating hours per year, with standby equipment in critical areas; and
- sufficient plant design flexibility for treatment of all ore types at design throughput.

The selection of these parameters is discussed in detail below.

Table 17-1: Summary of Process Plant Design Criteria

Criteria		Units	Design 52.5 kt/d	Design 105 kt/d
Crusher Feed		kt/d	52.5	105
		Mt/a	19.2	38.3
Crusher Availability		%	75	75
Crusher Throughput		t/h	2,917	5,833
Crusher Selection	Size		60” x 89”	60” x 89”
	No		1	2
Mill Throughput		Mt/a	19.2	38.3
Mill/Flotation Availability		%	92	92
Mill Throughput		t/h	2,378	4,755
Physical Characteristics (Design Values)	BWi	kWh/t	21.0	21.3
	RWi	kWh/t	15.6	15.6
	CWi	kWh/t	15.3	15.3
	SMC	kWh/m^3	5.33	5.33
	JK Axb	-	54.2	50.4
	Specific Gravity	t/m^3	2.57	2.57
Grind Size	P_{80}	µm	180	180
Head Grade (Design)		% Ni	0.37	0.37
		% S	0.10	0.10
		% Magnetite	4.20	5.10
Metal Recovery (Design Values)	Sulphide Nickel	%	56.0	56.0
	Awaruite Nickel	%	4.47	4.47
	Overall Nickel	%	60.5	60.5
Flotation Circuit Residence Times	Slimes	min.	33	33
	Sulphide Roughers	min.	90	90
	Sulphide 1st Cleaners	min.	45	45
	Sulphide 2nd Cleaners	min.	14	14
	Sulphide 3rd Cleaners	min.	11	11
	Magnetic Sulphide Scavenger	min.	60	60
	Awaruite Roughers	min.	45	45
	Awaruite Cleaners	min.	21	21
Ni Concentrate Filtration Rate		kg/m^2/h	450	450
Concentrates Thickening Flux		t/m^2/h	0.25	0.25
Tailings Thickening Flux		kg/m^2/h	0.45	0.45
Tailings Thickener Underflow Density		% w/w	40	40
KAX51 Consumption		g/t	80	80
MIBC Consumption		g/t	89	89
Cytec 65 (Frothing Agent) Consumption		g/t	2	2
Calgon Consumption		g/t	254	254
CMC Consumption		g/t	6	6
Sulphuric Acid Consumption (H_2SO_4)		g/t	3,888	3,888
Flocculant Consumption	Concentrate	g/t	10	10
	Tailings	g/t	20	20
SAG Mill Media Consumption		t/a	999	1,999
Ball Mill Media Consumption		t/a	1,808	3,615
Regrind Mill Media Consumption		t/a	621	1,242

17.4 Throughput & Availability

Ausenco selected one 11.6 m (38 ft) diameter SAG mill and two 7.9 m (26 ft) diameter ball mills for each 52.5 kt/d plant. Ausenco believes that this circuit is suitable to achieve the design throughput for design competency ore, with potential for increased throughputs for softer ores. Ausenco has nominated an overall plant availability of 92% or 8,059 h/a. This is an industry standard for a large, multi-train flotation plant with moderately abrasive ore. Benchmarking indicates that operating plants have consistently achieved this level. Given the low abrasion index of the Dumont ore, this availability is likely conservative.

17.5 Processing Strategy

Selection and sizing of the crushing and grinding circuits was determined through variability comminution testwork performed at SGS-Lakefield. Testwork provided a crusher work index, Bond ball and rod mill indices, Bond abrasion index, SMC and JK Axb values for the selected samples. Ausenco elected to use the 75^{th} percentile of each of these values in the design.

17.6 Head Grade

Each plant is designed to treat ore with a head grade of 0.37% Ni. The annual feed grades for the LOM range between 0.22-0.37%. The average plant feed grade for phase 1 (first 5 years of operations) is 0.34% nickel, with a maximum peak of 0.37%. The average feed grade for phase 2 is 0.26% nickel, with a peak of 0.31% nickel. The phase 2 plant design will need to be revisited once the phase one in operational to determine the optimum design point.

17.7 Flowsheet Development & Equipment Sizing

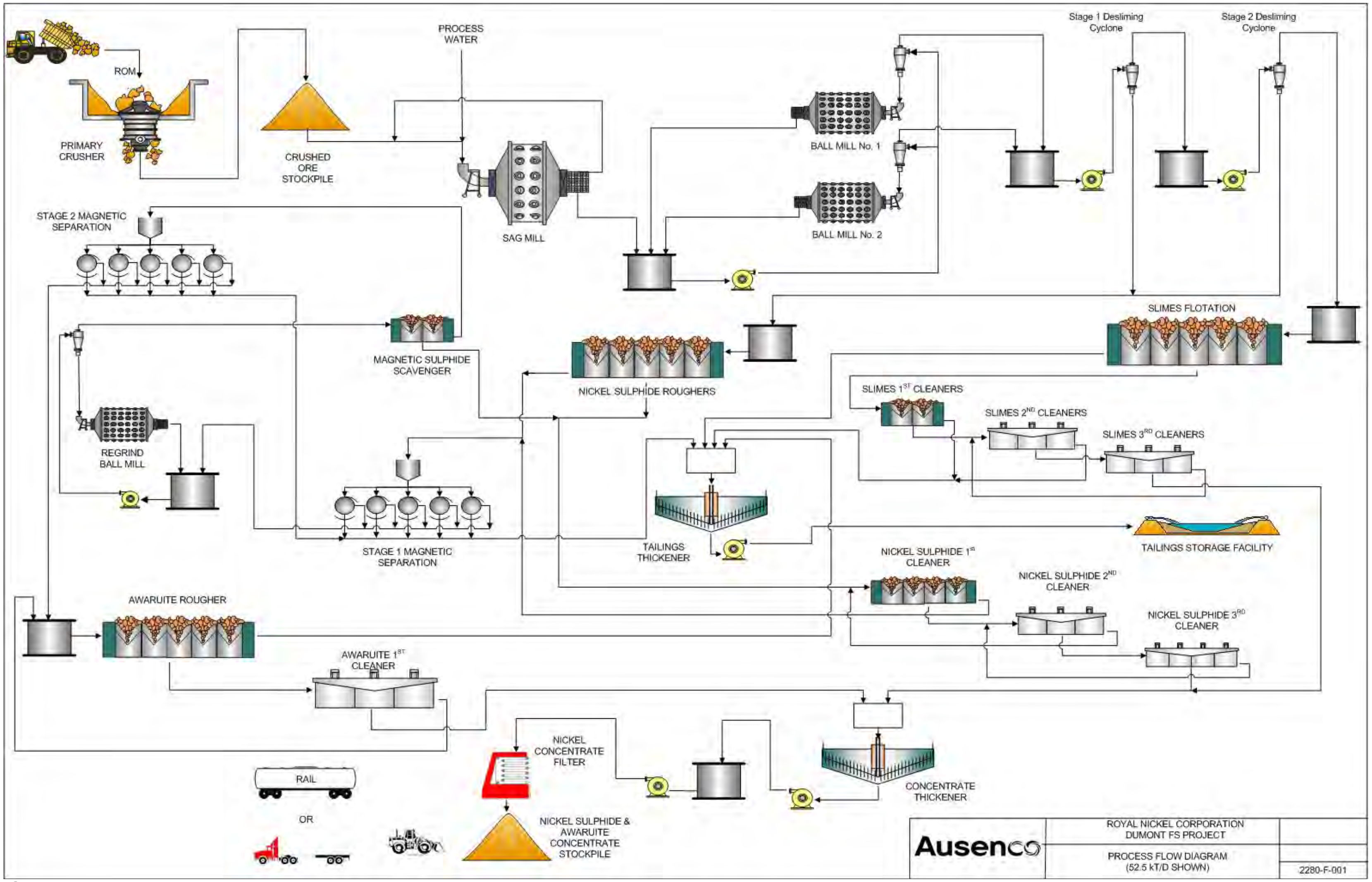
The process plant flowsheet design for the Dumont circuit was conceptually based on those of comparable large flotation plants and then confirmed or altered based on trade-off studies and metallurgical testwork. Figure 17.1 shows a process schematic for the Dumont plant (only 52.5 kt/d plant shown). Details of the flowsheet design and the selection of major equipment for the process plants are discussed in the sections below.

17.8 Unit Process Selection

The process plant design is based on a flowsheet with unit process operations that are well proven in the minerals processing industry. The Dumont flowsheet incorporates the following unit process operations (52.5 kt/d plant discussed below):

- Ore from the open pit is crushed using a primary gyratory crusher (assisted with a rock breaker) to a crushed product size of nominally 80% passing (P_{80}) 90 mm. Crushed ore is fed onto the covered stockpile feed conveyor.
- A covered conical stockpile of crushed ore with a live capacity of 12 h, with three apron feeders, each capable of feeding 60% of the full mill throughput.
- A 21 MW SAG mill, 11.6 m diameter (38 ft) with 6.7 m effective grinding length (EGL) (22 ft), utilizing a trommel screen for classification and oversize recirculation.
- Two 16 MW ball mills, 7.9 m diameter (26 ft) with 12.2 m EGL (40 ft), in closed circuit with hydrocyclones, grinding to a product size of nominally 80% passing (P_{80}) 180 µm.

Figure 17.1: Dumont Process Plant Schematic



Source: Ausenco.

- Two-stage desliming circuit via hydrocyclones. First stage to split mass with a cut size (D50c) of 50 µm. Second stage to split mass with a cut size (D50c) of 1 to 15 µm. Hydrocyclone sizes for each stage are 400 and 100 mm, respectively.
- Slimes rougher flotation consisting of one train of eleven 300 m^3 forced air tank flotation cells to provide 33 minutes of retention time.
- Slimes 1st cleaner, 2nd cleaner and 3rd cleaner flotation consisting of four 50 m^3, three 5 m^3 and three 1.5 m^3 forced air tank flotation cells to provide 30 minutes, 14 minutes and 10.5 minutes of retention time, respectively.
- Nickel sulphide rougher flotation consisting of three trains of nine (27 total cells) 300 m^3 forced air tank flotation cells per train to provide 90 minutes of retention time.
- Nickel sulphide 1st cleaner, 2nd cleaner, and 3rd cleaner flotation consisting of seven 200 m^3, six 20 m^3 and five 5 m^3 forced air tank flotation cells to provide 45 minutes, 14 minutes, and 9 minutes of retention time, respectively.
- Magnetic separation on nickel sulphide rougher and sulphide cleaner flotation tailings, consisting of two trains of seven 3.6 m long low intensity magnetic separators (LIMS) for a nominal mass recovery of approximately 12-15% of sulphide rougher and cleaner flotation feed.
- Magnetic concentrate regrind stage in a 8 MW ball mill, 6.7 m diameter (22.0 ft) with 10.8 m EGL (35.4 ft), operating in closed circuit with hydrocyclones, grinding to a product size of nominally 80% passing (P_{80}) of 46 µm.
- Magnetic sulphide scavenger flotation consisting of seven 200 m^3 forced air tank flotation cells to provide 66 minutes of retention time.
- Magnetic separation on magnetic sulphide flotation tailings, consisting of five 3.6 m long LIMS magnetic separators for a nominal stage mass recovery of approximately 50%.
- Awaruite rougher flotation consisting of six 70 m^3 forced air tank flotation cells per train to provide 70 minutes of retention time.
- Awaruite cleaner flotation consisting of five 1.5 m^3 forced air tank flotation cells to provide 21 minutes of retention time.
- Nickel concentrate thickening in a 14 m diameter high-rate thickener followed by dewatering in a vertical pressure filter.
- Thickening of deslime tailings, combined magnetic separation tailings and awaruite rougher tailings in an 88 m diameter high-rate thickener to an underflow density of 40% solids.
- TSF for process tailings deposition in a conventional dam.
- Reagent mixing facilities for KAX51 (collector), Calgon (depressant), CMC (depressant) and both concentrate and tailings flocculant.
- Reagent off-loading facilities for MIBC and Cytec 65 (frothers) and sulphuric acid.
- Process water and distribution system for reticulation of process water throughout the plant as required. Process water is collected in a process water pond that is predominantly supplied from the tailings thickener overflow and tailings storage facility. Other sources include concentrate thickener overflow and pit de-watering operations.
- Potable water is generated by treatment water from the freshwater tank in a reverse osmosis (RO) unit at the site. Potable water is distributed to the plant and for miscellaneous purposes around the site.
- Raw water distribution services to supply cooling water, gland water, a portion of the reagent mixing water, firewater, etc.
- Plant, instrument and flotation air services and associated infrastructure.

Layouts of the process plant area and process plant are shown in Figure 17.2 and Figure 17.3, respectively.

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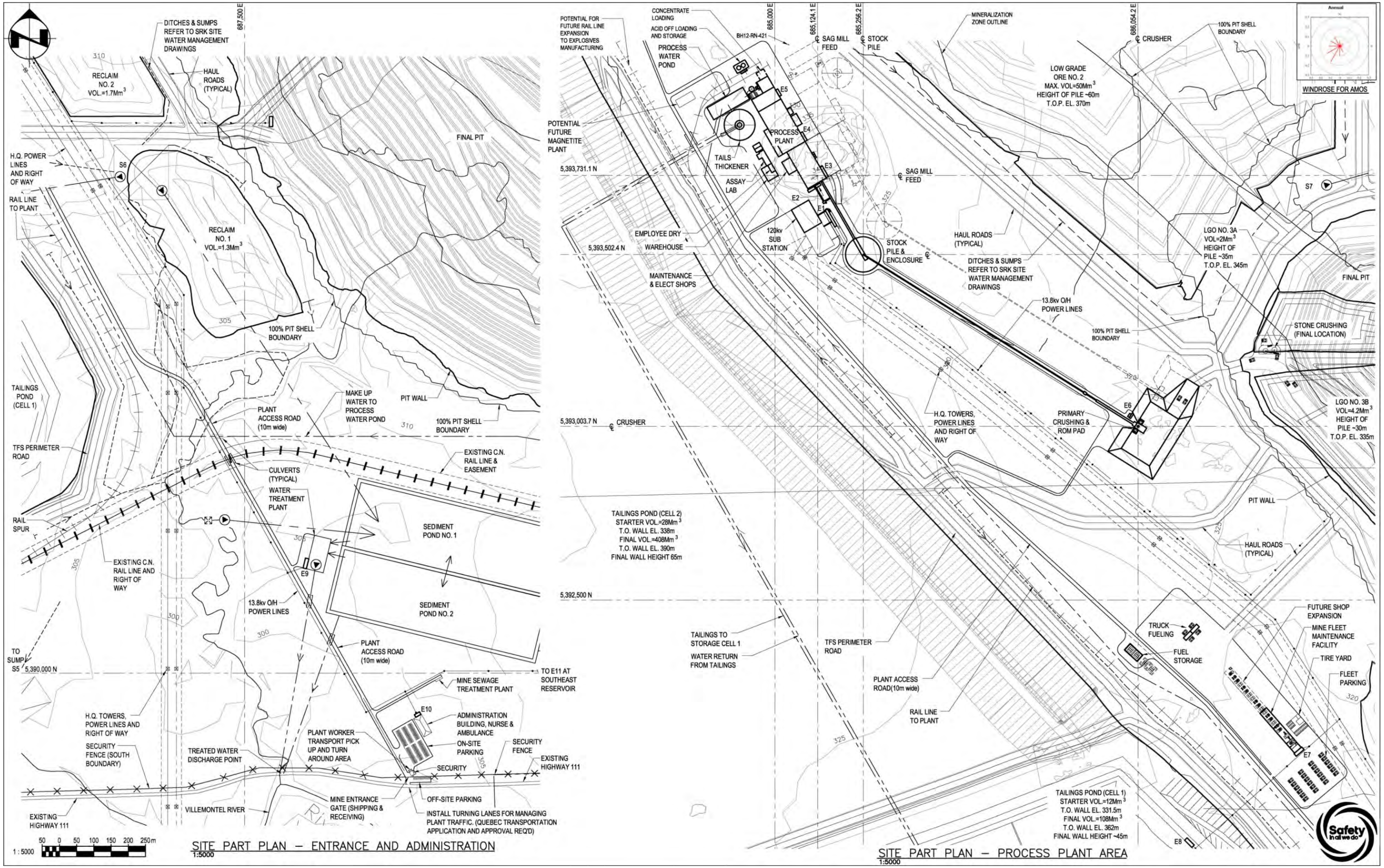
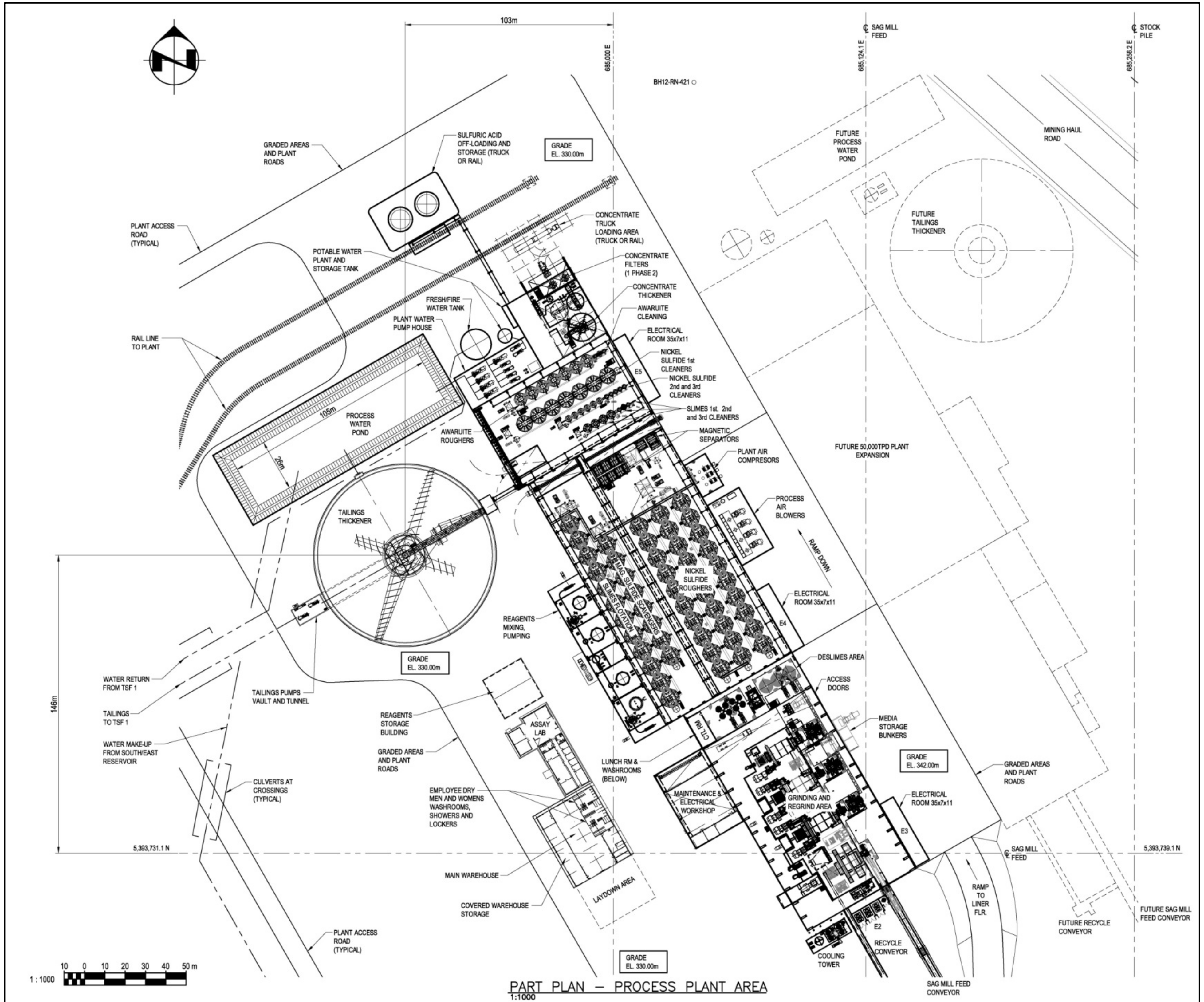
Figure 17.2: Layout of Process Plant Area



Source: Ausenco.

Ausenco

Figure 17.3: Layout of Process Plant



Source: Ausenco.

17.9 Comminution Circuit

17.9.1 Primary Crushing

Based on the design throughput and moderate competency ore characteristics, a 60” x 89” gyratory crusher is considered the most suitable primary crusher for the duty.

The primary crusher will be located at the edge of the ROM pad. A partially buried crusher design has been selected to reduce ROM pad elevation (reduce mine haulage costs) without major excavation being needed.

Trucks will dump from both sides of a 243 m^3 capacity live hopper above the crusher; alternatively, ore can be rehandled and fed with a front-end loader (FEL).

The primary crusher will be located in an enclosed building. This will help minimize dust emissions and reduce noise. Additionally, an overhead crane will be installed in the building for maintenance of the crusher.

The gyratory crusher will crush ore to a product size of 80% passing 90 mm. The maximum feed size to the crusher will be 1.2 m; oversize material will be broken with a rock breaker.

17.9.2 Crushed Ore Stockpile

Crusher product will be conveyed from the crusher discharge vault by the variable-speed primary crusher discharge belt feeder, and discharged onto the sacrificial conveyor. Ore is transferred via the stockpile feed conveyor to the crushed ore stockpile. A weightometer will be installed on the sacrificial conveyor to provide production rate data for the crushing circuit. Space for a self-cleaning electromagnet has been made over the sacrificial conveyor head pulley to remove steel tramp prior to discharge onto the stockpile feed conveyor should it be needed.

The stockpile will provide a minimum of 12 h live capacity at the design SAG mill new feed rate of 2,378 t/h; higher throughputs will reduce this capacity. The total capacity of the stockpile is approximately 60 hours of SAG mill new feed capacity or approximately 143 kt. In event of the crushing circuit being out of operation for extended periods, a bulldozer can be used to reclaim the dead material in the stockpile to provide emergency feed to the milling circuit. Three apron feeders have been selected to reclaim ore from the stockpile, each able to deliver 60% of the design mill feed rate.

The stockpile will be enclosed to minimize fugitive dust emissions. The cover will be a dome of structural steel construction and cladding. The stockpile will be approximately 98 m diameter x 37 m high.

17.9.3 Comminution Design Criteria

The major comminution design parameters used for this study are**:**

- crusher work index (CWi) of 15.3 kWh/t based on the 75th percentile of the samples tested at SGS;
- bond rod mill work index (RWi) of 15.6 kWh/t based on the 75th percentile of the samples tested at SGS;
- bond ball mill work index (BWi) of 21.0 kWh/t based on the 75th percentile of the samples tested at SGS;

- bond abrasion index (AI) of 0.007 g based on the 75^{th} percentile of the samples tested at SGS;
- drop weight index (DWI) of 4.8 kWh/t as measured from the SMC test (equivalent to an Axb of 54.2 at a specific gravity of 2.6); and
- target grind size P_{80} of 180 µm, based on various flotation testwork programs (directed by RNC).

The grinding circuit was designed to be capable of processing the required tonnage of 52.5 kt/d. To account for variations in ore competency, Ausenco uses the 75^{th} percentile for all design data. This would indicate that the current circuit could achieve higher throughputs when processing less competent ore.

Flotation testwork and mineralogy have indicated that Dumont ores are relatively insensitive to grind sizes (P_{80}) up to about 150 µm in the laboratory. In order to achieve the specified design recovery, RNC has nominated a primary grind size target of P_{80} of 180 µm. A P_{80} of 180 µm has been selected because it is typical for sulphides to be found in finer size fractions under plant conditions when compared to laboratory testwork (as screens are used for sizing in the laboratory compared to hydrocyclones in the process plant).

Ausenco uses a power-based approach to determine grinding circuit power requirements. The approach is based on empirically derived models developed from a database of actual plant operations data and associated bench-scale testwork. Critical input parameters to the model are ore competency (measured by either JK drop weight Axb or SMC DWi values) and Bond work indices (CWi, RWi and BWi). Ausenco's power-based model predicts the milling efficiency of the various circuits based on the JK drop weight/SMC data, which is a measure of ore competency. The approach also considers the impact of ultramafic ores on the Bond BWi test results.

The specific energy and mill sizing determined using Ausenco's in-house method for the major ore types is shown in Table 17-2.

The installed ball mill power of 16,000 kW incorporates allowances for drive train losses as well as a design contingency to account for the accuracy of the models, calculations and testwork used to determine the expected average pinion power.

The installed motor power for the SAG mill incorporates similar allowances, as well as an additional contingency to allow adjustment in the mill operating conditions to handle ore variability. These allowances and contingencies require the installation of 21,000 kW.

Table 17-2: Mill Design Criteria

Criteria		Units	Design 52.5 kt/d	Design 105 kt/d
Throughput		t/h	2,378	4,755
Mill Type			SAG	SAG
Shell Power Required		kW	17,716	35,432
No. of Mills			1	2
Mill Speed		% Nc	75	75
Ball Charge Volume	Nominal	% vol	15	15
	Maximum (design)	% vol	20	20
Total Charge Volume	Nominal	% vol	27	27
	Maximum (design)	% vol	35	35
Mill Diameter	Inside shell	M	11.6	11.6
Mill Length	EGL	M	6.7	6.71
Installed Motor Power		kW	21,000	21,000
Mill Type			Ball	Ball
Grind Size	P_{80}	µm	180	180
Pinion Power Required		kW	24,922	49,843
Number of Mills			2	4
Pinion Power Required per mill		kW	12,461	12,461
Mill Speed		% Nc	75	75
Ball Charge Volume	Nominal	% vol	26	26
	Maximum (design)	% vol	30	30
Mill Diameter	Inside shell	m	7.9	7.9
Mill Length	EGL	m	12.2	12.2
Installed Motor Power		kW	16,000	16,000

17.9.4 Reclaim, SAG & Ball Mill Circuit

The crushed ore will be reclaimed from the ore stockpile by three apron feeders onto the SAG mill feed conveyor. The feeders will be equipped with variable speed drives.

A SAG mill feed weightometer will be installed on each SAG mill feed conveyor to provide feed rate data for control of the reclaim feeders. The reclaimed crushed ore will be fed at a controlled rate to the SAG mill.

Discharge from the SAG mill will gravitate through a trommel screen. Oversize pebbles from the trommel screen (scats) will be recycled back onto the mill feed conveyor after magnetic separation of scats. Pebbles will be reintroduced onto the mill feed conveyor via the recycle pebble conveyors. Undersize from the SAG trommel screen will gravity flow into the cyclone feed hopper.

The SAG mills discharge slurry will be pumped via dedicated cyclone feed pumps to the two ball mill cyclone clusters, each operating in a closed-circuit configuration with a single ball mill. Water is added to the cyclone feed hopper as needed to achieve the required cyclone feed pulp density.

Hydrocyclone underflow from each cluster will gravity flow to a dedicated 16 MW twin pinion ball mill (two 8 MW motors operating in parallel). Discharge from each ball mill will gravity flow through a trommel screen, into the cyclone feed hopper for reclassification. Cyclone overflow will gravity flow to the first stage deslime cyclone feed hopper.

The milling circuit will require the installation of two clusters of seven 800 mm hydrocyclones per cluster, of which six will be in operation with one on standby. A pneumatically actuated valve will be provided with each hydrocyclone for isolation requirements. Rubber-lined steel pipes, hoppers, and chutes will be installed throughout the grinding circuit to handle coarse slurry.

Two vertical sump pumps will be provided in the grinding area, and one in the stockpile area, to facilitate clean-up.

17.9.5 Mill Circuit Classification

The classification circuit has been designed for a nominal circulating load of 250%; this is a typical design value for material of similar characteristics and target grind size and is widely used in the industry for SAB circuits. To avoid damage to the cyclone clusters, the SAG mill discharge slurry first passes through a trommel screen with 12 mm x 55 mm slotted apertures to remove pebbles; the undersize flows into the hydrocyclone feed hopper. The pebbles will be fed back to the SAG mill feed via pebble recycle conveyors for further grinding. An electromagnet on the pebble recycle conveyors will remove any tamp metal.

A pebble circulating load of 15% of the new feed rate has been assumed in the design, based on typical industry experience with ores of similar competency. The conveyors are designed to handle peak loads of up to 25% of new feed.

SAG and ball mill discharge will be combined in the ball mill hydrocyclone feed hopper and then pumped to two clusters of 800 mm diameter hydrocyclones to a target overflow P_{80} of 180 µm. Each hydrocyclone cluster will be fed with a dedicated hydrocyclone feed pump.

Hydrocyclone overflow will report as feed to a desliming circuit prior to slimes flotation, while the coarse hydrocyclone underflow from each of the two clusters will combine and report to a dedicated ball mill (No. 1 or No. 2) for further grinding.

17.9.6 Deslime Circuit

A two-stage desliming circuit deslimes ball mill No. 1 and No. 2 hydrocyclone overflow to remove slimes. This is critical to achieve optimal flotation kinetics. The two-stage circuit accomplishes this with hydrocyclone clusters. The first stage will split mass with a cut size (D50c) of 50 µm utilizing 400 mm hydrocyclones. Overflow from the first stage of desliming passes through a horizontal trash screen to remove large particles that could potentially block the smaller cyclone in the second stage. Trash screen underflow is feed for the second stage.

The second stage will split mass with a cut size (D50c) of 10-15 µm utilizing 100 mm hydrocyclones. The underflows from both the first and second stage are combined and fed to the nickel sulphide rougher flotation circuit. The stage 2 hydrocyclone overflow flows by gravity to the slimes flotation circuit.

17.10 Flotation Circuit Design

Mineralogical examination and lab scale testwork has revealed that a majority of the nickel sulphide in the ore is recoverable with adequate concentrate grades through flotation at a P_{80} of 150 µm. However, the use of magnetic recovery stages and regrind has been shown that additional nickel sulphide recovery is achievable.

The aforementioned magnetic recovery stage has the main purpose of recovering nickel that is in various alloy forms, predominately awaruite. A subsequent regrind stage and magnetic recovery is required to liberate additional nickel sulphides and allow for higher rates of gangue rejection.

17.10.1 Circuit Type & Size

The flotation circuit selected to concentrate Dumont ore consists of slimes rougher and three-stage cleaner flotation, sulphide rougher and three-stage cleaner flotation, magnetic separation, regrind, secondary magnetic separation and awaruite rougher and cleaner flotation. Slime rougher and cleaner flotation tailings, combined magnetic tailings and awaruite rougher tailings report to a common tailings thickener. Slimes, sulphide and awaruite concentrates are combined after flotation. The residence times for the nickel flotation circuit have been based on the testwork performed on various Dumont ore types and composite ore type samples.

The testwork flotation and design residence times are summarized in Table 17-3.

Table 17-3: Summary of Nickel Flotation Residence Times

Flotation Stage	Locked-Cycle Test Time (min)	Scale Factor	Specified Design Time (min)
Slimes Roughers	10	3.3	33
Slimes 1st Cleaners	10	3.0	30
Slimes 2nd Cleaners	4	3.5	14
Slimes 3rd Cleaners	3	3.5	10.5
Sulphide Roughers	30	3.0	90
Sulphide 1st Cleaners	15	3.0	45
Sulphide 2nd Cleaners	4	3.5	14
Sulphide 3rd Cleaners	3	3.0	9
Magnetic Sulphide	20	3.3	66
Awaruite Roughers	20	3.5	70
Awaruite Cleaners	6	3.5	21

17.10.2 Flotation Circuit Configuration

Stage 2 deslime hydrocyclone overflow will be fed by gravity to the slimes distribution box, where flotation reagents will be added. The slimes flotation circuit consists of eleven 300 m^3 tank flotation cells. The cells will be in a paired configuration arrangement with an elevation change between pairs of cells. Additional dosing points for the flotation reagents will be located along the slimes flotation banks.

Concentrate from the slimes flotation cells will flow by gravity to a slimes concentrate hopper and is pumped to the slimes cleaning circuit. Slimes rougher tailings will be pumped to the tailings thickener.

The slimes 1st cleaner stage consists of four 50 m^3 tank flotation cells, operating in an open circuit configuration. Slimes 1st cleaner flotation tailings are combined with the second cleaner tailings and pumped to tailings thickener and the concentrate is pumped to the nickel sulphide 2nd cleaner flotation stage.

The slimes 2nd cleaner stage consists of three 5 m^3 tank flotation cells, operating in open circuit (with the option to send 2nd cleaner tails to 1st cleaner feed). The slimes 3rd cleaner stage consists of three 1.5 m^3 tank flotation cells. The 2nd and 3rd stages are configured such that 3rd cleaner flotation tailings flow by gravity back to the head of the 2nd cleaner stage. The slimes 2nd cleaner concentrate is pumped to the slimes 3rd cleaner and the 3rd cleaner concentrate is pumped to the concentrate thickener.

The reagents added will consist of a combination of KAX51 (PAX, collector), MIBC and/or Cytec 65 (frother) and Calgon (depressant). The distribution box will gravity flow to the slimes flotation cells, which are connected in series.

Stage 1 and 2 deslime hydrocyclone underflow is combined, and fed via gravity to three parallel rougher conditioning tanks, where flotation reagents will be added. The conditioning tanks will gravity flow to the nickel rougher flotation cells, which are connected in series. Three trains of nine 300 m^3 forced-air tank flotation cells have been selected to provide the required residence time for the rougher flotation. The cells will be in a paired configuration with the exception of one cell, with an elevation change between pairs. Additional dosing points for flotation reagents will be located along the rougher banks.

Concentrate from each train of the rougher cells will flow by gravity to a common nickel rougher concentrate hopper and then be pumped to the nickel sulphide cleaning circuit. Rougher tailings from each train will flow by gravity to the magnetic plant feed hopper.

The reagents added will consist of a combination of KAX51 (PAX, collector), MIBC and/or Cytec 65 (frother) and Calgon (depressant).

Nickel rougher tails will gravity flow to the stage 1 magnetic separation feed hopper were it is combined with sulphide cleaner tailings (see Section 17.11). Magnetic concentrate will be pumped to a regrind stage. Regrind cyclone overflow flows by gravity to a magnetic sulphide scavenger flotation stage.

The magnetic sulphide scavenger flotation stage consists of seven 200 m^3 tank flotation cells (single train). The cells will be in a paired configuration arrangement with an elevation change between pairs of cells. Magnetic sulphide scavenger tailings will be pumped to stage 2 magnetic separation (see Section 17.11), while recovered nickel sulphide concentrate will be pumped to the nickel sulphide cleaner flotation circuit.

Three streams feed the sulphide 1st cleaner flotation circuit; sulphide rougher flotation concentrate, magnetic sulphide scavenger flotation concentrate and nickel sulphide 2nd cleaner flotation tailings. The sulphide 1st cleaner stage consists of seven 200 m^3 tank flotation cells. Nickel sulphide 1st cleaner flotation tailings are pumped to the aforementioned regrind circuit and concentrate is pumped to the nickel sulphide 2nd cleaner flotation stage.

The nickel sulphide 2nd cleaner stage consists of six 20 m^3 tank flotation cells. The sulphide 3rd cleaner stage and the awaruite cleaner consist of five and six 5 m^3 tank flotation cells, respectively. The 2nd and 3rd stages are configured such that 3rd cleaner flotation tailings flow by gravity back to the head of the 2nd cleaner stage.

Stage 2 magnetic separation concentrate will be pumped to the awaruite flotation circuit.

The awaruite flotation circuit consists of a rougher flotation stage (tailings report to the tailings thickener) and a single cleaner flotation stage (tailings recirculated to head of the rougher stage). A conditioning tank is used before the rougher flotation stage, where reagent addition takes place. The rougher flotation stage consists of six 70 m^3 tank flotation cells. The awaruite cleaner flotation stage consists of five 1.5 m^3 flotation cells.

17.11 Magnetic Separation

A low-intensity magnetic separation circuit is used to treat the tailings of the nickel sulphide rougher and cleaner flotation stages. The function of this circuit is to recover nickel containing magnetic awaruite (primarily awaruite) that can be found in the sulphide rougher and cleaner tails.

Testwork performed using magnetic separation established mass recovery and approximate concentrate nickel grade design criterion. Other parameters are based on benchmarking and vendor recommendation.

The selected design criteria are summarized in Table 17-4.

Table 17-4: Summary of Magnetic Concentrate Recovery Circuit Design Loadings

Flotation Stage	Magnet Strength Gauss	Magnet Linear Loading t/h/(m drum)	Magnet Volumetric Loading m^3/h/(m drum)	Magnet Configuration
Nickel Sulphide Rougher Tailings (Train 1)	1,000	30	110	Counter-current
Nickel Sulphide Rougher Tailings (Train 2)	1,000	30	110	Counter-current
Magnetic Sulphide Scavenger Tailings	1,000	30	110	Counter-current

Nickel sulphide rougher and cleaner flotation tailings will be pumped to two trains of 7 single drum magnetic separators (3.6 m long and 1.2 m diameter) per train via a feed distributor. The number of separators selected will allow for variations in throughput and magnetic recoveries.

Magnetic concentrate will be pumped to the regrind circuit, specifically a regrind mill cyclone feed hopper.

A second low-intensity magnetic separation circuit is used to treat the tailings of the magnetic sulphide scavenger flotation stage. This material is the 1st stage magnetic concentrate which is first passed through the regrind mill and a sulphide flotation stage. The function of this circuit is to recover nickel containing magnetic awaruite (primarily awaruite) and further reject gangue material.

The only design criterion that has been established at this stage is approximate mass recovery and an approximate concentrate nickel grade. Other parameters are based on benchmarking and vendor recommendation.

Magnetic sulphide scavenger flotation tailings will be pumped to one train of five single-drum magnetic separators (3.6 m long and 1.2 m diameter) per train via a feed distributor. The number of separators selected will allow for variations in throughput and magnetic recoveries.

Magnetic concentrate will be pumped to the awaruite flotation circuit, specifically an awaruite rougher conditioning tank.

Magnetic separation tailings (non-mags) from both stages will be combined and pumped to the tailings thickener.

17.12 Magnetic Concentrate Regrind

A closed regrind circuit is used to grind the magnetic concentrate stream. The stream is fed to the regrind mill cyclone feed hopper and then pumped to a cluster of 508 mm hydrocyclones to achieve a product size of nominally 80% passing (P_{80}) of 46 µm. Hydrocyclone overflow will report as flotation feed to magnetic sulphide flotation, while the underflow will report to the regrind ball mill for further grinding.

The design stream to the regrind mill is 650 t/h; with an estimated maximum BWI of 18 kWh/t based on data from similar applications. This indicates a specific power requirement of below 11.8 kWh/t to achieve the nominated regrind size of P_{80} of 45 µm. On this basis, a single 8,000 kW regrind ball mill has been selected for this study.

17.13 Nickel Concentrate Thickening, Storage & Filtration

The thickening of the nickel concentrate will be common to both the 52.5 kt/d and 105 kt/d plants. A larger thickener has been selected to accommodate the additional concentrate for a plant throughput of 105 kt/d.

Nickel flotation concentrate will be thickened to approximately 60% w/w solids in a 14 m diameter above-ground high-rate thickener. Concentrate thickener design is based on a typical (estimated) settling rate of 0.25 t/m^2/h for nickel concentrate.

The concentrate storage tank prior to filtration will have a live volume of 450 m^3, which has been sized based on 24-hours of residence time for the 52.5 kt/d plant. The concentrate storage allows for routine maintenance of the concentrate filter.

The concentrate filter will be a vertical plate and frame “Larox style” filter press. No concentrate handling testwork has been undertaken as there has been insufficient concentrate production to do filtration testwork. Ausenco has based the design on experience with similar ultra mafic nickel concentrators to derive a conceptual design. These assumptions will require testwork and confirmation during subsequent study.

This filter was selected as a benchmark against other similar operations as there is an absence of testwork to determine appropriate filter selection at this point in time.

Filter cake is stockpiled and loaded onto either rail cars or trucks via a front-end loader.

Expansion to a plant throughput of 105 kt/d will require an additional concentrate storage tank and vertical Larox style filter press.

17.14 Tailings Disposal

The design basis chosen for this level of study includes a tailings thickener, with disposal of thickened tailings in a nearby TSF and recovery of water from the TSF surface.

The tailings thickener design has been based on a settling rate of 0.45 t/m^2/h and results in the selection of an 88 m high rate thickener.

Plant tailings slurry, consisting of slimes rougher and cleaner flotation tailings, combined magnetic separation tailings and awaruite rougher tailings, will be pumped to the high-rate thickener. The slurry will be thickened to a target density of 40% w/w solids. The thickened tailings slurry will be pumped to the TSF via a 4 km HDPE pipeline. Process water from the thickener overflow will flow by gravity to the plant process water storage pond. Additional process water recovery from the TSF will be pumped back to the process water pond via barge pumps, and a 4 km HDPE water pipeline.

17.15 On-Stream Analysis

The on-stream analysis (OSA) system will provide online nickel and other supporting assay analysis on the following 11 streams:

- ball mill hydrocyclone overflow;
- slimes flotation tailings (one stream);
- nickel sulphide rougher flotation tailings (three streams);
- combined magnetic separation tailings (non-mags);
- awaruite rougher flotation tailings;
- awaruite cleaner flotation concentrate;
- nickel sulphide 3rd cleaner flotation concentrate;
- final combined concentrate; and
- final combined tailings.

Analyzed samples, exiting the OSA system, will be discharged into sample return hoppers and will either be pumped back to the 1st stage deslime cyclone feed hopper, concentrate filter feed tank, or the combined magnetic tailings hopper.

The OSA system will be located at an optimal position to permit the maximum use of gravity flow of sample feed and sample reject slurries and minimize requirements for sample pumps and hoppers.

17.16 Reagents

Reagents for the project are listed below.

Collector – Potassium Amyl Xanthate (KAX51 (PAX**)** – Potassium Amyl Xanthate (KAX51, PAX) is a sulphide mineral collector and will be supplied in 1000 kg bulk bags as a dry reagent. KAX51 will be shipped by road to site and offloaded by forklift. KAX51 will be stored in the reagents storage area of the warehouse facility and delivered to the KAX51 mixing area. KAX51 bulk bags will be lifted by overhead hoist and loaded into the mixing tank by way of a bag splitter. Water is added to the agitated tank to produce a solution concentration of 20% w/w. The diluted mix is transferred to the KAX51 storage tank by way of pump. The KAX51 solution is stored in a day tank, where it is reticulated around the plant in a ring main system using the lime ring main pumps (duty/standby arrangement).

Frother 1 – Methyl Isobutyl Carbinol – Methyl Isobutyl Carbinol (MIBC) will be supplied by bulk tankers and off-loaded via pump into a storage tank. The storage tank will have capacity for several days of consumption at design flow rates. The frother will be distributed to the flotation circuit dosing points by multiple dosing pumps.

Frother 2 – Cytec 65 – Cytec 65 is a trademarked frother that will be supplied in bulk boxes and off-loaded into a storage tank. The storage tank will have capacity for several days of consumption at design flow rates. The frother will be distributed to the flotation circuit dosing points by multiple dosing pumps.

Depressant 1 – Calgon – Calgon (sodium hexametaphosphate) is used as a gangue depressant in this flotation circuit and will be supplied in 1000 kg bulk bags as a dry reagent. Calgon will be shipped by road to site and offloaded by forklift. Calgon will be stored in the reagents storage area of the warehouse facility and delivered to the Calgon mixing area. Calgon bulk bags will be lifted by overhead hoist and loaded into the mixing tank by way of a bag splitter. Water is added to the agitated tank to produce a solution concentration of 5% w/w. The diluted mix is transferred to the Calgon storage tank by way of pump. The Calgon is stored in a day tank, where it is reticulated around the plant in a ring main system using the lime ring main pumps (duty/standby arrangement).

Depressant 2 – Carboxy Methyl Cellulose (CMC) – Carboxy Methyl Cellulose (CMC) is used as a gangue depressant in this flotation circuit and will be supplied in 1000 kg bulk bags as a dry reagent. CMC will be shipped by road to site and offloaded by forklift. CMC will be stored in the reagents storage area of the warehouse facility and delivered to the CMC mixing area. CMC bulk bags will be lifted by overhead hoist and loaded into the storage hopper by way of a bag splitter. Loose CMC is transported via screw feeder to the CMC mixing tank. Water is added to the agitated tank to produce a solution concentration of 1% w/w. The diluted mix is transferred to the CMC storage tank by way of pump. The depressant will be distributed to the flotation circuit dosing points by multiple dosing pumps.

pH Modifier – Sulphuric Acid (H_2SO_4) – Sulphuric acid will be supplied by bulk tankers and off-loaded into a storage tank; expansion will require an additional storage tank. One unloading pump is required for pre and post expansion conditions. The storage tank will have capacity for 84 hours of consumption at design flow rates. The sulphuric acid will be distributed to the flotation circuit dosing points by multiple dosing pumps.

Flocculant – Magnafloc 333 – Two flocculant mixing, storage and dosing systems will be provided, located in the reagent preparation area. The tailings thickener and the concentrate thickener will have dedicated flocculant dilution systems. Magnafloc 333 will be supplied in 1000 kg bulk bags for the tailings flocculant mixing system and as 25 kg bulk bags for the concentrate mixing system. Magnafloc will be shipped as a dry reagent.

The flocculant for the concentrate thickener will be manually loaded into the flocculant storage hopper and fed via screw feeder to the flocculant mixing tank where it is diluted to 0.25% w/w. The flocculant will be pumped via dosing pumps to an inline mixer where the solution is further diluted to 0.025% w/w and fed to the concentrate thickener.

For the tailings thickener, flocculant 1000 kg bulk bags will be lifted by overhead hoist and loaded into the storage hopper by way of a bag splitter. Loose flocculant is transported via screw feeder to the flocculant mixing tank. Water is added to the agitated tank to produce a solution concentration of 0.25% w/w. The diluted mix is transferred to the flocculant storage tank by way of pump. The Calgon is stored in a day tank. The flocculant will be pumped via dosing pumps to an inline mixer where the solution is further diluted to 0.025% w/w and fed to the tailings thickener.

Grinding Media – Forged carbon steel grinding media will be delivered to site in 20 tonne containers. The balls will be unloaded into a storage bin via a vendor-supplied, hydraulically-operated container unloader. Overhead cranes in the primary milling and regrind areas will be used to load steel balls into the SAG mill, ball mill and regrind mill. Steel balls will be transported from the SAG, ball, or regrind mill grinding media storage bunker in the storage yard by FEL to the grinding media hoppers located near the mill feed end. From the hoppers, balls will be added to a bottom discharging kibble, which will be hoisted by overhead crane to a position above the mill feed chute, and emptied into the mill.

17.17 Air Services

17.17.1 Process Air

The flotation blowers will supply low pressure process air to the flotation cells at the required supply pressure (two different pressures). There will be four blowers (all four duty) installed to meet flotation air requirements for the initial 52.5 kt/d process plant. In order to meet the process air requirements for the plant expansion to 105 kt/d, four additional air blowers will be added. Pressure control valves will be installed in the air distribution lines to meet the different air pressure requirements of different flotation cells. Multiple-stage, centrifugal type blowers will be used with a “blow-off” arrangement to adapt to fluctuations in flotation air demand.

The blowers will be housed inside their own room to reduce plant noise to an acceptable level. The room will have ventilation for cooling.

17.17.2 Plant & Instrument Air

Three rotary screw air compressors will provide intermediate pressure compressed air for plant and instrument air requirements. There will be two duty and one standby compressor operating in lead-lag mode. Plant air will be stored in the plant air receivers to account for variations in demand prior to being distributed throughout the plant.

During the expansion, two plant air compressors and one instrument air compressor will be added.

17.18 Process Control Philosophy

The control philosophy to be implemented for the Dumont Nickel project is typical of those used in modern mineral processing operations.

Field instruments provide inputs to a set of programmable logic controllers (PLCs). Process control cubicles are located in the motor control centres (MCCs), and contain the PLC hardware, power supplies, and input/output (I/O) cards for instrument monitoring and loop control.

The PLCs perform the control functions by:

- collecting status information of drives, instruments, and packaged equipment;
- providing drive control and process interlocking; and
- providing proportional-integral-derivative (PID) control for process control loops.

Standard personal computers (PCs) will be located in the main control room (MCR) and the crusher control room (CCR). The PCs are networked to the PLCs and operate a supervisory control and data acquisition (SCADA) system that provides an interface to the PLCs for control and monitoring of the plant.

The SCADA system is configured to provide outputs to alarms, control the function of process equipment, and provide logging and trending facilities to assist in analysis of plant operations.

The control rooms are purpose-built structures. The majority of the plant is controlled from the MCR, which is located between the comminution and flotation circuits. The MCR houses two control room operator stations, one engineering station and a printer.

Operator control stations are fully redundant, such that the failure of one station would not affect the operability of the other station or control of the plant. Control stations are supplied from an uninterruptible power supply unit (UPS) with 20 minutes of standby capability.

Drives that form part of a vendor package are controlled from the vendor's control panel. At a minimum, "Run" and "Fault" signals from each vendor control panel are made available to the SCADA system via the PLC.

The general control strategy adopted for the Dumont Nickel project is as follows:

- integrated control via the process control system (PCS) for areas where equipment requires sequencing and process interlocking;
- hardwired interlocks for safety of personnel;
- motor controls for starting and stopping of drives at local control stations, via the PCS or hardwired depending on the drive classification (all drives can be stopped from the local control station at all times; local and remote starting is dependent on the drive class and control mode);
- control loops via the PCS except where exceptional circumstances apply;
- monitoring of all relevant operating conditions on the PCS and recording selected information for data logging or trending.

Trip and alarm inputs to the PCS will be failsafe in operation (i.e., the signal reverts to the de-energized state when a fault occurs).

18 PROJECT INFRASTRUCTURE

18.1 Introduction

The project consists of an open pit mine, crushing, stockpile conveyor, coarse ore stockpile and enclosure, SAG and ball mill grinding circuit, nickel flotation circuit including regrind, nickel concentrate thickening, filtration and storage, rail car/truck loadout, tailings thickening facility, reagents, and ancillary services (refer to Figure 18.1 for an illustration of the overall site layout).

The layout of the plant and all associated facilities was designed to restrict impact to only the St. Lawrence watershed. The boundary between the St. Lawrence and Arctic watershed is shown on Figure 18.1. Any waste dumps are located at least 1 km from the Launay Esker.

The site layout takes into account site topography and limits imposed by the locations of the pit, stockpiles and waste dumps subject to the above constraints. The grinding area is founded on bedrock to reduce civil costs and take advantage of gravity flow where possible.

18.2 Site Power Supply

Hydro Quebec (HQ) will provide electrical power to the mine site via a 10.5 km long, 120 kV overhead powerline to be constructed, that would be connected as a tee-off to an existing line. The line enters the property from the south near the security entrance gate, and runs up to the process plant main 120 kV substation.

Both the initial and expansion phases of the Dumont project will require three 120:13.8 kV 60/80 MVA ONAN/ONAF main transformers. The new 120 kV substation and six main transformers will be installed near the SAG mill feed conveyor. The 13.8 kV medium voltage network will be used for the primary electrical distribution and for feeding large loads such as the SAG mill and ball mills.

The 13.8 kV distribution circuits run from the main electrical room E1 (located adjacent to 120 kV outdoor substation) to secondary electrical rooms located close to the areas served. In these secondary electrical rooms, the 13.8 kV distribution voltage will be converted to 4.16 kV and 600 V using 13.8-4.16 kV and 13.8-0.6 kV indoor dry-type transformers. For the mine circuit, an isolation transformer 13.8-13.8 kV will be used to separate the neutral grounding circuit of the portable substations from the main plant ground system.

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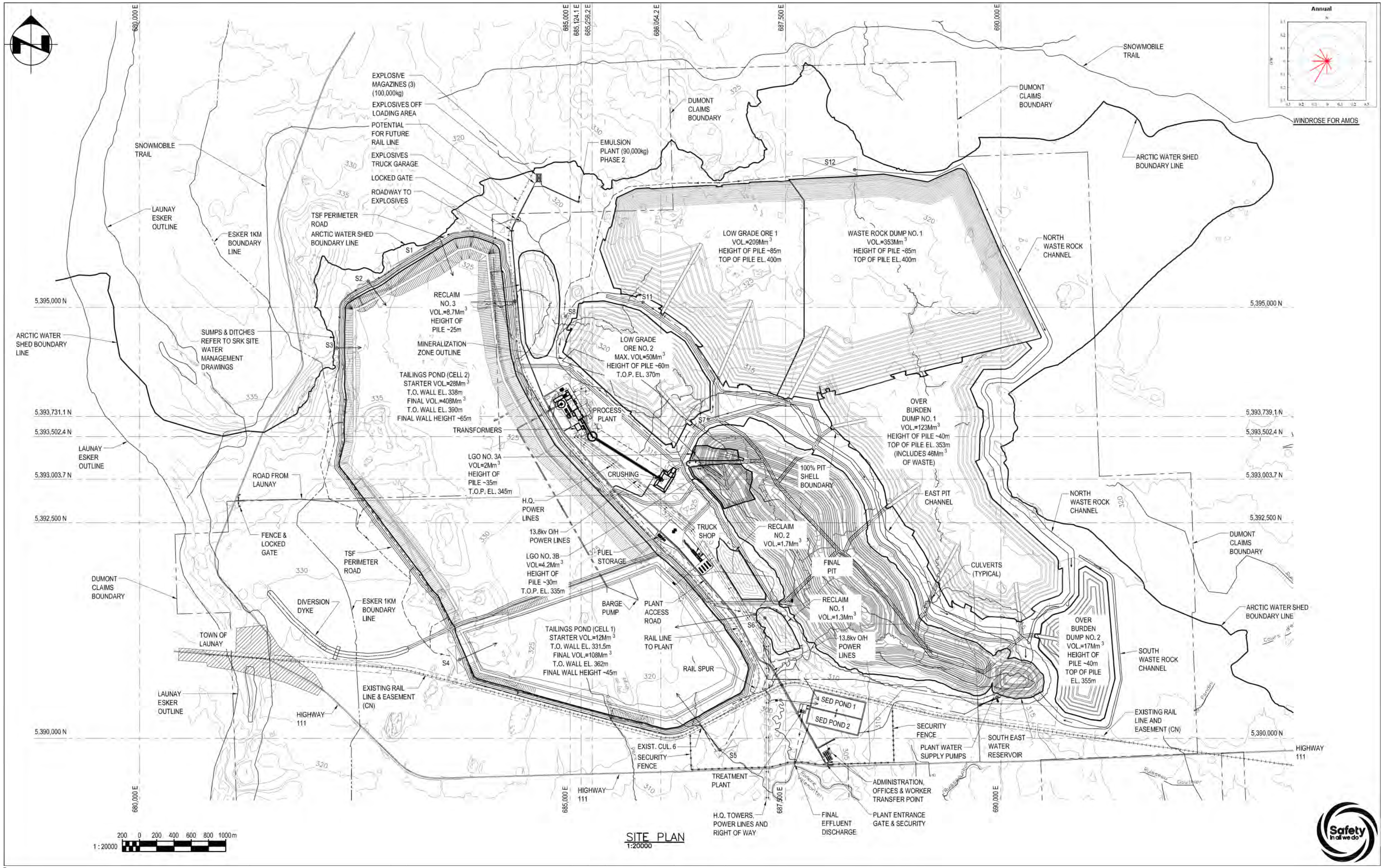
Figure 18.1: Overall Site Layout



Source: Ausenco.

In case of power failure, two 13.8 kV emergency diesel generators will automatically start and supply power for all essential plant loads. The generators will be located at the main electrical room No. E1 conveniently located for distribution of power throughout the plant using the 13.8 kV network. Uninterruptable power supplies (UPS) and DC battery systems will be provided in the various electrical rooms for essential protection and control equipment.

Power factor correction equipment and harmonic filters will be located near the main electrical room and connected to the main 13.8 kV switchgear to ensure that the electrical load, as seen by HQ, meet their requirements.

HQ will provide power during construction at 25 kV and will enter the site from the south near the security gate house. A temporary 25:13.8 kV substation will be located at the point where the line enters the site and power will be distributed to the rest of the site via 13.8 kV overhead lines which will be re-used for the permanent installation.

18.3 Rail Spur

A rail spur that services the process plant is proposed for the project. The total length of the rail spur is approximately 5 km. Fuel unloading is located near the mining truckshop, and the track extends north of the process plant to load concentrate.

18.4 Roadways

The Dumont on-site roads will be constructed of crushed waste rock available from site and naturally available materials. A dedicated mobile aggregate crushing plant will be utilized for the entire life of project (including the period post ex-pit operation, when stockpiles are being reclaimed) to provide aggregate for continually resurfacing haul roads.

18.5 Process Plant

The process plant area consists of the crushing facility, covered stockpile and process plant building. The overall process plant enclosed structure is approximately 350 m long, and consists of four connected buildings: grinding, flotation and magnetic separation, cleaning and scavenging, and concentrate thickening. These are described below.

The primary crushing facility is closely located to the open pit, to the east. The crushed ore is conveyed to a covered stockpile, which is approximately 40 m high x 96 m diameter. The crushed ore conveyor from the crushed ore tunnel to the crushed ore transfer station is approximately 200 m long. The stockpile feed conveyor feed conveyor extends a further 800 m from the transfer station to the stockpile.

From the covered stockpile, the ore is conveyed via apron feeders, through a reclaim tunnel, and a 280 m long SAG mill feed conveyor into the grinding area. The feed to the SAG mill is at 90°, which helps reduce the size of the grinding building. The grinding building consists of a SAG mill, two ball mills, a regrind mill, desliming cyclones and an overhead crane. It is 121 m long x 81 m wide x 47 m high. The grinding area electrical room E3 is connected at the east side. The plant control room will be located at an elevated position adjacent to the hydrocyclone cluster, and will be have aluminum-framed windows for viewing into the process plant. In particular, the grinding and flotation areas will be easily viewable from the control room. The lunch room is located below the control room.

The slimes and nickel flotation building is north of the grinding circuit. It contains the slimes flotation, nickel roughers cells, magnetic separators, two overhead cranes, and is 138 m long x 74 m wide x 29 m high. The reagents mixing area is connected to the west of the nickel flotation building. The process air blowers, plant air compressors, and electrical room E4 are connected to the east side of the building.

To the north of the flotation building is the cleaning and roughers building. It contains the nickel sulphide cleaner cells, awaruite roughers and cleaners cells, and one overhead crane that services the entire area. This building is 46 m long x 77 m wide x 22 m high. The electrical room E5 is located on the east side of the building.

To the north of the cleaning and roughers building is the concentrate thickening building, which is 42 m long x 35 m wide x 19 m high. This building also contains the steam boiler. The water pumphouse and process water pond are west of the concentrate thickening building. The 88 m diameter tailings thickener is adjacent to the process water pond, on the south side.

During the plant expansion in Year 5 of operations, a second train of the crushing facility, stockpile, grinding building, slimes and nickel flotation building, cleaning and roughers building, process water pond, and tailings thickener will be duplicated and built to the east of the original process plant. The concentrate thickener area is in between the two process buildings and will not need to be expanded.

18.6 Waste Rock & Overburden Dumps, Low-Grade Ore & Reclaim Stockpiles

The open pit mining operation will generate 1,338 Mt of overburden and waste rock and 606 Mt of low-grade ore that will be temporarily impounded in stockpiles. Waste rock will be utilized in the construction of various site facilities including, for example, the tailings dams, mine roads and rail lines. The balance will be stored in three waste rock dumps. Overburden from the open pit stripping will be used for reclamation, where applicable, with the balance stored in two overburden dumps (OB-1 and OB-2). Low-grade ore will be intermittently processed or stored in four low-grade ore stockpiles (LGO1, LGO2, LGO3a and LGO3b). Low-grade ore that is still stockpiled when mining ceases will be processed over the remaining project life.

In addition, three reclaim stockpiles will store select overburden for subsequent use as cover material at the tailings storage facility. It is expected that the reclaim stockpiles will be depleted, reloaded and depleted multiple times during the project life.

Figure 18.1 shows the location of the various dumps and stockpiles.

18.6.1 Waste Rock Dumps

Of the total waste rock volume, expected to be 540 Mm^3, approximately 83 Mm^3 of waste rock will be used in dam construction at the TSF cells, approximately 8 Mm^3 will be used in the TSF key trenches, and approximately 18 Mm^3 will be used elsewhere on site. The total waste rock storage requirement is, therefore, about 431 Mm^3. The combined waste rock storage capacity can be met by the three dumps shown on Figure 18.1.

WR-1, located in the northeastern part of the site, will have a capacity of 353 Mm^3 (including ten clay cells) and is expected to have a final height of approximately 85 m. Approximately 46 Mm3 of waste rock will be co-disposed with sand and gravel and clay within the main overburden dump (OB-1), which reaches an approximate height of 40 m. The in-pit waste dump is located within the southeast extension of the open pit and will store approximately 53 Mm^3.

18.6.2 Overburden Dumps

Two overburden dumps, separated by one of the main drainages, will be developed immediately east of the proposed open pit (Figure 18.1). The northernmost overburden pile (OB-1) will be

38 m high and will store up to 123 Mm^3 of material which includes 46 Mm^3 of waste rock. The southeastern pile (OB-2) will be 40 m high and will store up to 17 Mm^3 of material.

Some of the overburden from these two stockpiles will be used as cover material during site reclamation, but most of the overburden will remain in place post-closure.

18.6.3 Low-Grade Ore Stockpile

Four stockpiles, LGO1, LGO2, LGO3a and LGO3b will be developed to store and periodically extract low-grade ore on a regular basis during the period of active mining. The maximum volume of low-grade ore that will require storage is estimated to be 263 Mm^3.

LGO1 is located north of the open pit and west adjacent to WR1 (Figure 18.1) and is expected to have an ultimate height of approximately 85 m at its peak storage volume of 209 Mm^3. LGO2 is located north west of the open pit and adjacent east to the mill plant (Figure 18.1), and is expected to have an ultimate height of approximately 60 m at peak storage volume of 50 Mm^3. LGO3a and LGO3b is located within the northwest region of the open pit. LGO3a and b will be used for short-term ore storage during early mine development.

18.6.4 Reclaim Stockpiles

Three reclaim stockpiles (Reclaim 1, 2 and 3) are located between the TSF and open pit (Figure 18.1). They are designed to temporarily store overburden, topsoil and organics for reclamation purposes. A portion of reclaim 3 will be used for clay and sand and gravel storage for TSF construction. All three stockpiles have an ultimate height of around 25 m.

18.7 Tailings Storage Facility

The TSF is located approximately 400 m west of the process plant and consists of two cells (see Figure 18.1). Cell 1 will be constructed first and will provide storage through approximately Year 6 of operations. Cell 2, situated immediately north of Cell 1, will provide storage from approximately the end of Year 6 through Year 22 of operations.

The TSF is designed to store approximately 680 Mt of tailings; 142 Mt in Cell 1 and 538 Mt in Cell 2. Each of the two TSF cells will be developed in stages while the open pit is being mined. Once mining at the open pit has ceased, stockpiled ore will be processed for approximately 13 years and those tailings, approximately 498 Mt, will be deposited in the open pit.

18.7.1 General Description

The two TSF cells will be constructed as ring dikes due to the subdued regional relief.

The starter dam at TSF Cell 1 will provide storage for tailings produced during the first year of mineral processing, approximately 12 Mm^3. Dam raises will be completed annually based on the centreline construction method and will provide storage for tailings produced over the next five years. In total, TSF Cell 1 will store approximately 108 Mm^3 of tailings.

TSF Cell 2 will be constructed using the same general methods as Cell 1. The Cell 2 starter dam will store approximately 29 Mm^3 of tailings and the final dam will store a total of 408 Mm^3 of tailings.

18.7.2 Design Criteria

The design criteria for the TSF are listed in Table 18-1.

Table 18-1: Tailings Storage Facility Design Criteria

Design Item	Criterion	Reference
Mine Life Open pit mining period Mill only period	33 years 20 years (first of 33 years) 13 years (remainder of 33 years)	RNC
Tailings production Year 1 Year 2, 3, 4 Year 5 Year 6, and onward	 46 kt/d 52.5 kt/d 77. kt/d 105 kt/d	RNC
Required total TSF tailings storage Mass Volume	 680 Mt 516 Mm^3	RNC
Required in-pit storage Mass Volume	 498 Mt 371 Mm^3	RNC
Dam Classification(3)	Variable between Very High and Significant(1)	SRK
Maximum Design Earthquake	1:5000 year, PGA = 0.085g	SRK
Freeboard above supernatant water pond	2.5 m(2)	SRK
Design Flood	1:1000 year, 24-hour duration	SRK
Stability Factor of Safety (FOS) (3) Static, short term (end of construction) Static, long term Dynamic	 1.3 1.5 1.0	SRK
Setback limits CN Rail Plant Rail Arctic watershed boundary Esker 1 km buffer	 100 m 30 m 100 m 100 m	SRK/RNC

Note: 1. Dam classification follows Canadian Dam Safety Guidelines 2007 Edition (CDA 2007). 2. The freeboard is assumed to be the "dry" freeboard between the dam crest and the maximum water level in the pond (the exposed tailings beach is assumed to have a slope of 2% and to extend from a point below the dam crest to a line a minimum of at least 2.5 m below the dam crest). 3. Dams at the TSF were designated with two classifications based on their corresponding failure consequences.

18.7.3 Site Selection

The selection of the FS TSF site was influenced by:

- Potential impacts to the township of Launay as a result of dust and noise associated with mining, and particularly the deposition of waste rock and low-grade ore at their respective dump locations. Modelling indicated that these dumps should be sited as far as practical from Launay (i.e., north and northeast of the open pit, which led to siting the TSF to the west and northwest of the open pit).
- Arctic watershed boundaries, wetlands, public infrastructure, foundation conditions and topographic relief constrain the location and footprint of the TSF. In order to keep the mine facilities in a single watershed, the TSF was sited on the St. Lawrence side of the boundary that separates the St. Lawrence and Arctic watersheds. To the west of the TSF, there are wetlands, which have been avoided to the maximum practical extent, due to their high environmental value.

- The CN rail line that bounds the southern limit of the TSF.
- The TSF competes for space with other mine features, including the plant site, waste rock and overburden dumps, low-grade ore stockpiles, reclaimed soil stockpiles, project transportation corridors and water management facilities.
- To the extent possible, the TSF utilizes bedrock outcrop and topographic highs for siting the TSF dams.

18.7.4 Foundation Preparation beneath the Perimeter Dams

The geotechnical database for the TSF area indicates that grey clay with a very soft to firm consistency is present along some sections of the proposed perimeter dams. In the event this material was to be left in place, stability analyses indicate the typical downstream slope of the perimeter dam would likely have to be in the order of 8H:1V to satisfy stability criteria. At this slope, and assuming that about 40% of the TSF perimeter is underlain by clay, the storage capacity of the TSF would be significantly impacted.

It was concluded that a key trench developed by excavating the grey clay and replacing it with waste rock from mining operations would facilitate steepening of the downstream slope of the perimeter dams. Therefore, where clay is present beneath the dams, the clay within a specified portion of the dam footprint will be removed to expose firm bearing material, either the competent, dense sand and gravel stratum or, close to outcrops, bedrock. The area of the key trench is shown in Figure 18.2.

The typical cross-section through the key trench will be between 3 m and 10 m deep, and 60 m to 120 m wide across the base, with cut slopes at 2H:1V. Since the key trench will be constructed in stages as the dam is raised, the dimensions of the key trench at any given stage will depend on the dam height and the clay thickness.

The total excavation and replacement quantity is estimated to be 1.7 Mm^3 for TSF Cell 1 and 6 Mm^3 for TSF Cell 2. The brown clay material excavated from the key trench will be used in construction of the TSF core or for reclamation, grey clay will be deposited with the overburden from the open pit stripping. The replacement material will be waste rock from the open pit, although it is possible that some sand and gravel from the overburden stripping at the open pit could be used.

Foundation preparation outside the key trench, but beneath the dam footprint, will consist of logging, where necessary, grubbing of roots and stripping of the organic soils. The organic soils will be stockpiled for later use in TSF reclamation activities.

18.7.5 Starter Dam Design

18.7.5.1 TSF Cell 1

A plan view of the TSF Cell 1 starter dam is shown in Figure 18.3. A typical Cell 1 cross-section is shown in Figure 18.4. Key trenches will be constructed in identified areas shown in Figure 18.2. The Cell 1 starter dam will be constructed to a crest elevation of 331.5 m and a length of approximately 7.1 km. Where clay is present, the starter dam upstream slope will be 8H:1V with no key trench, while the downstream slope will be 2H:1V with a key trench. Where clay is absent, both slopes will be 2H:1V. A 6 m wide, vertical clay core will tie into the clay stratum or bedrock at the upstream limit of the key trench and will be extended to the top of the starter dam. A filter zone, also 6 m wide, will be constructed downstream of the clay core.

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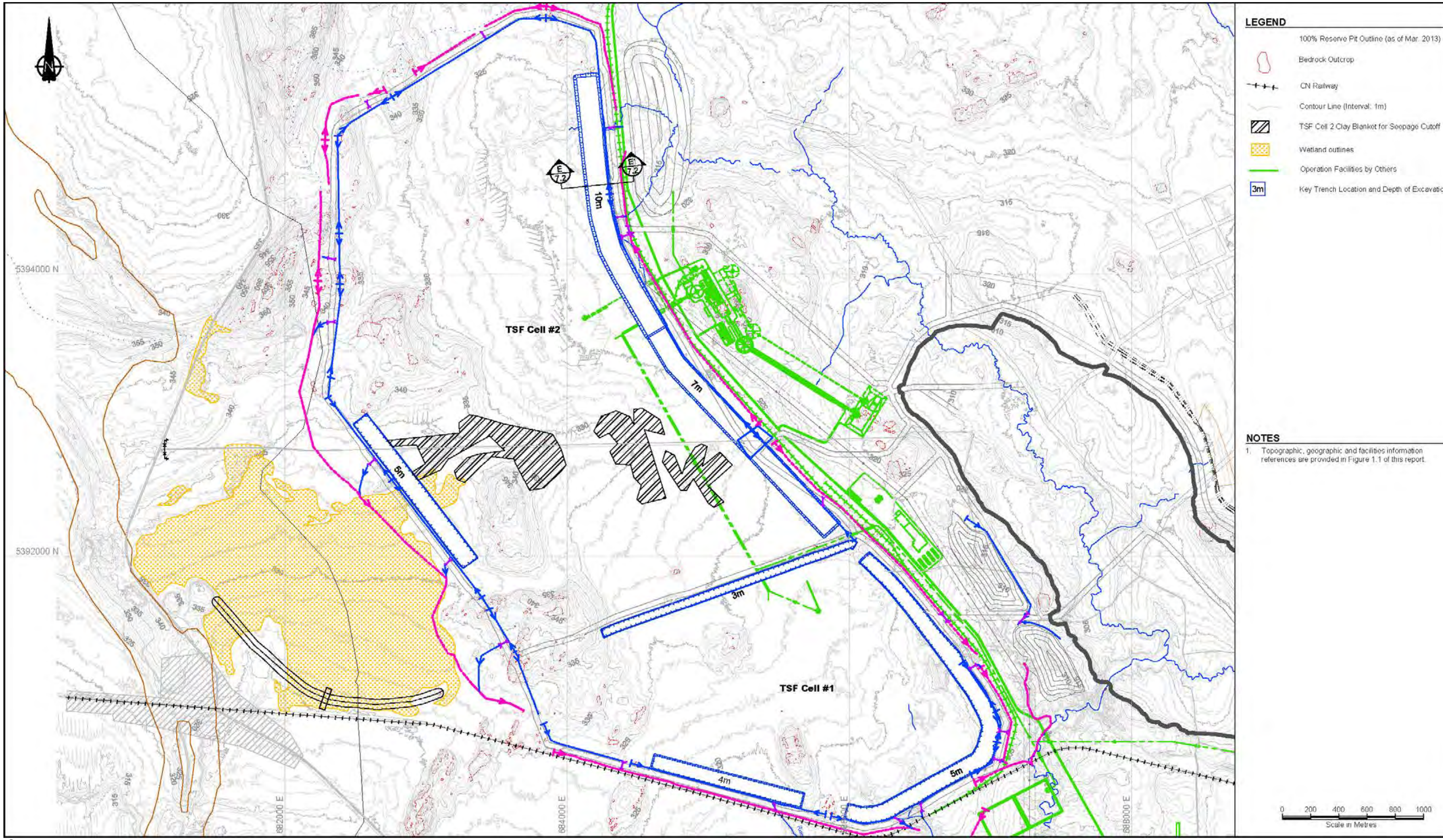
Figure 18.2: Location of the Key Trench



Source: SRK.

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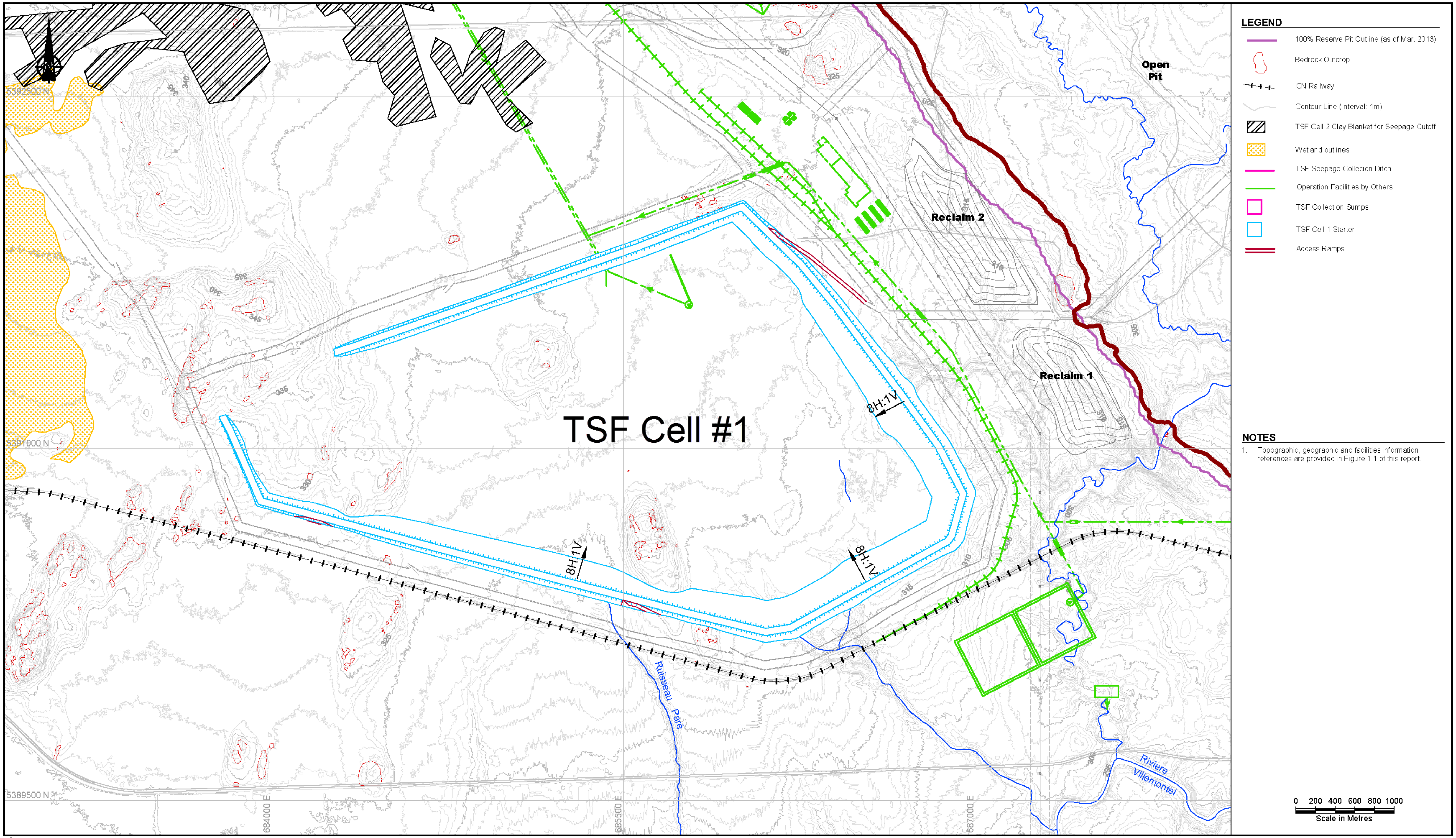
Figure 18.3: TSF Cell 1 Starter Dam Plan



Source: SRK.

The total volume of the starter dam at TSF Cell 1 will include 5.2 Mm^3 of waste rock and a combined clay core and gravelly silty sand filter volume of 1 Mm^3. In addition, a 0.6 Mm^3 of waste rock is required for key trench construction.

18.7.5.2 TSF Cell 2

Cell 2 starter dam will be constructed to a crest elevation of 338 m approximately 9.7 km in length, which will provide storage for Year 7 of the operation. The configuration of the starter dam will be similar to the Cell 1 starter, with a clay core, filter, and areas of key trench.

The total volume of the starter dam at TSF Cell 2 will include 6.6 Mm^3 of waste rock and a combined clay core and gravelly silty sand filter volume of 1.2 Mm^3. The Cell 2 starter dam will include 6.6 Mm^3 of waste rock, a combined core and filter volume of 1.2 Mm^3, and 1.5 Mm^3 of waste rock for key trench construction.

18.7.6 Dam Raises & Final Dam

The dams will be raised using the centreline construction method. Tailings will be spigotted from the perimeter dams in order to create a tailings beach between the dam and the interior pond. For each dam raise, waste rock will be placed over the existing dam and onto the upstream edge of the tailings beach, and a trench will then be excavated to install the clay core and filter zone (Figure 18.4 overleaf). Both of these zones will be carried up the dam as the dam is raised in 3 to 5 m lifts.

The downstream slope of the TSF cells will be constructed at 2H:1V.

It is expected that waste rock will be hauled to the TSF on a nearly continuous basis throughout the year. The clay core and filter zone will only be constructed when temperatures are above freezing.

Construction scheduling details related to the dam raise will depend on the final mine plan.

A number of access ramps will be built into the strategic locations in the downstream face of the dams for construction access. These ramps will be approximately 30 m wide, with a grade of 12H:1V. Four ramps will be developed around Cell 1 and five at Cell 2.

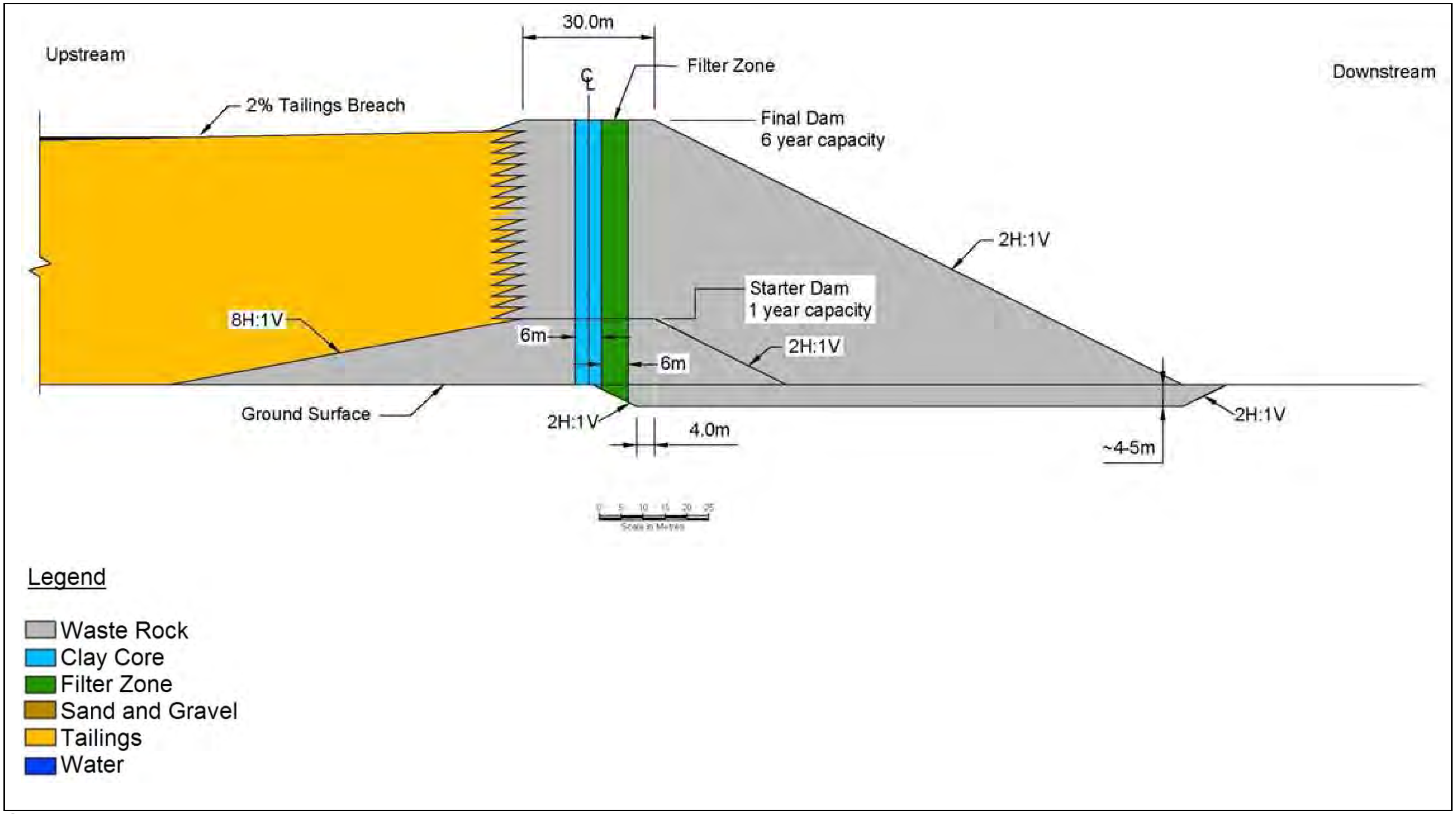
The final crest elevations will be 362 m and 390 m for TSF Cells 1 and 2, respectively. The total volume of material to construct Cell 1, including perimeter dams, core, filter, key trench, access ramps will be 31 Mm^3. Cell 2 construction will require 79 Mm^3 of material.

18.7.7 In-Pit Tailings Disposal

18.7.7.1 General

When the mining operation is completed, the mill will continue to process the stockpiled ore for another 13 years. Approximately 498 Mt of tailings will be deposited into the open pit for permanent storage.

Figure 18.4: Typical Cross-section through TSF Dam



Source: SRK

18.7.7.2 Operations

Tailings will be discharged from one or more spigots at the northwest portion of the open pit. Perimeter discharge is considered unnecessary due to the fact that the available storage volume within the pit greatly exceeds the required tailings storage volume.

At the end of mine life, the tailings water in the pit will be pumped out and treated prior to discharge to the Villemontel River. The open pit will then be allowed to fill with direct precipitation and runoff. Figure 18.5 shows a typical section at end of milling (approximately Year 33).

18.7.8 Water Management

18.7.8.1 General

The water management plan at the TSF is largely controlled by the following factors:

- The supernatant pool within the TSF will be separated from the perimeter dam by a tailings beach, and will be used to provide recycle water for the plant site.
- Seepage through the TSF dam will be limited by the clay core, and seepage through the TSF foundation will be limited by the existing (or constructed) basal layer of clay.

18.7.8.2 Water Pool & Water Return

Under non-winter conditions, tailings will be deposited around the perimeter of the dam using a conventional spigot method to develop a tailings beach which separates the supernatant pool from the perimeter dams.

Winter tailings deposition will utilize single point discharge into the supernatant pool (subaqueous) to fill the depression that is expected to form in the middle of each TSF cell. This will also mitigate the potential formation of ice within the tailings beach.

Water for recycle will be obtained using a floating barge and pump and pipeline system.

18.7.8.3 Seepage Collection

Within the footprint of TSF cell 2, there are a number of relatively small areas where sand and gravel are exposed with no natural clay cover. A 0.5 m layer of clay will be placed over these areas to prevent tailings pore water from seeping into the natural groundwater system (Figure 18.2).

A series of ditches leading to seven seepage collection sumps will be established around the external perimeter of the TSF. Pumps will be setup at each pond to convey this water back into the TSF for recycling, and thereby prevent contaminated water from potentially leaving the project area. Typically the sumps are rectangular, with length to width ratios of approximately 3:1, dimensions ranging from 50 m to 500 m, and depths of 2 to 4 m. The seepage collection ditches will be 1.5 to 2.5 m deep, with a ditch bottom width between 0.5 m to 1.5 m and side slopes of 6H:1V (in soil) and 0.75H:1V (in bedrock). The layout of the seepage collection facilities is shown on Figure 18.1.

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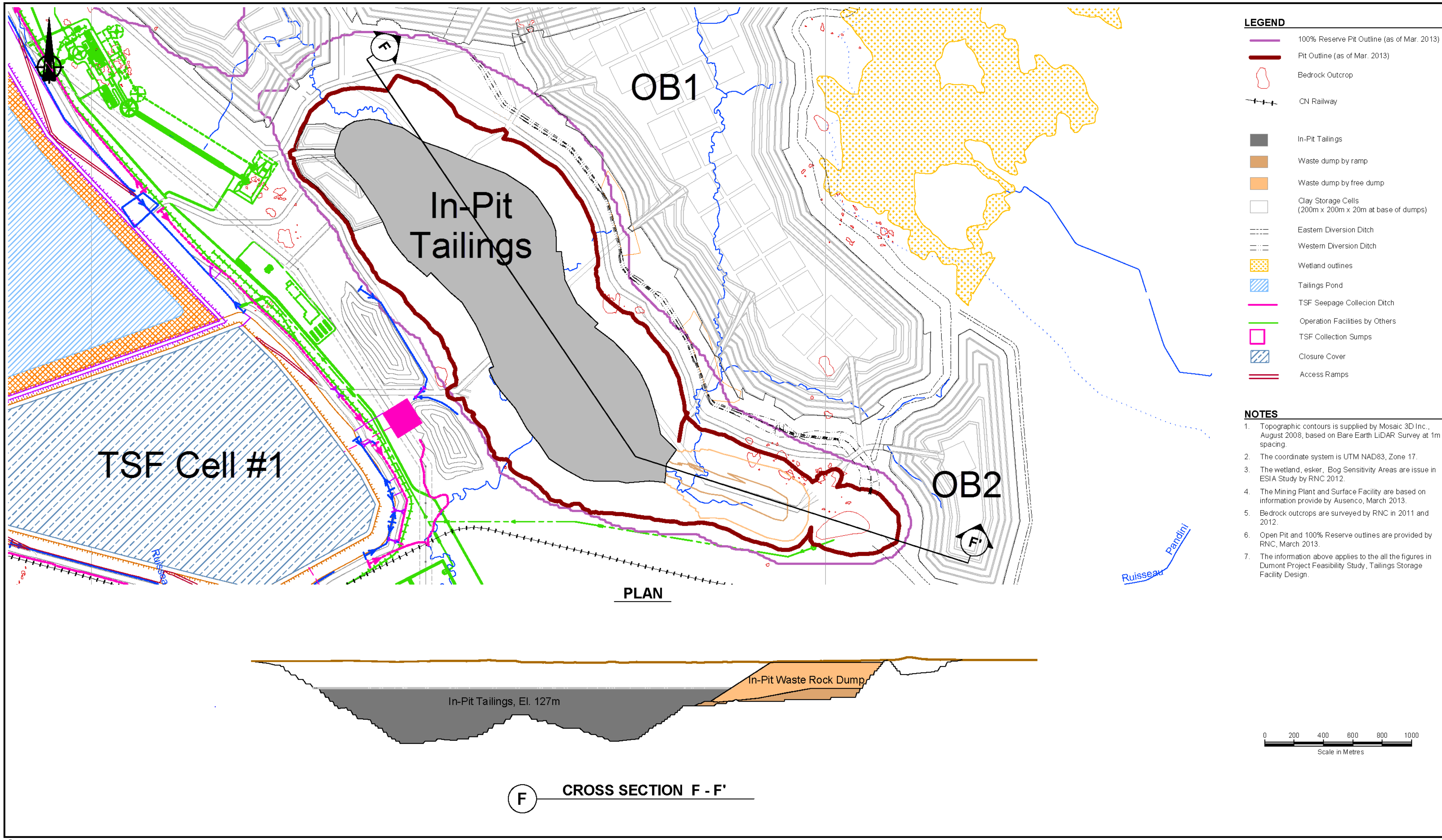
Figure 18.5: Typical Section at End of Milling



Source: SRK.

18.7.9 Tailings Delivery System

The tailings delivery system will transport slurried tailings from processing plant to the TSF. The delivery system will be sized initially on the basis of a 52.5 kt/d operation and increase to 105 kt/d. This will consist of 40 inch diameter HDPE (high density polyethylene) pipeline, approximately 4 km long. This pipeline will initially transport 4,407 m^3/h of tailings to the TSF. A second line of the same size and length will be installed adjacent to this pipeline in order to meet the expansion to 105 kt/d during the sixth year of operation. The two pipelines will transport a combined total of 8,826 m^3/h of tailings to the TSF. The tailings thickener underflow pumps will be on emergency power to prevent the lines from freezing in case of a power loss.

18.7.10 Return Water Delivery System

The return water delivery system for recycle water from the TSF has been sized on the basis of 2,353 m^3/h of water being pumped from the TSF to the Process Water Pond, for the initial 52.5 kt/d operation. This system will consist of barge pumps and a 24 inch diameter HDPE pipeline, approximately 4 km long, adjacent to the tailings pipeline. A second line of the same size and length will be installed adjacent to this pipeline to meet the expansion to 105 kt/d during the sixth year of operation. The two water return lines will transport a combined total of 4,628 m^3/h. The pipelines will be heat traced at low points to prevent freezing.

18.7.11 Closure

TSF Cells 1 and 2 are to be reclaimed progressively and at the end of their respective operational lives.

Soil covers at least 0.5 m thick will be established on the sides of the TSF perimeter dams. In the case of Cell 1, most of this cover will be placed at or near the end of its operational life, at the end of Year 6. At Cell 2, this will correspond to the end of Year 22, though it is anticipated the final downstream face of the perimeter dams will be in place well before then, making it possible to progressively cover most of the downstream face before Year 22.

Soil covers approximately 0.5 m thick will also be established over the tailings surface of both TSF cells. The ponded water over the central part of the cell will be pumped to the open pit to facilitate consolidation and improve bearing capacity for equipment.

The soil required for Cell 1 reclamation will be stored in two nearby stockpiles from active open pit stripping. When Cell 1 reclamation is completed, these stockpiles will be reloaded with material from stripping operations in preparation for Cell 2 reclamation.

Engineered water management structures will be built in the cells to manage surface water runoff passively, and in a way that prevents erosion of the soil covers.

Post closure, the quality of the water collected in the sumps will be monitored, and until quality requirements are met, water in the sumps will continue to be pumped to the water treatment plant. Once the water quality is deemed acceptable, the channels will be re-graded towards the open pit where possible. An outlet from the open pit reservoir will be constructed to direct excess water to the Villemontel River. The in-pit tailings storage will permanently maintain passive water cover. An engineered outflow channel will be constructed to control the water level within the pit.

18.8 Truckshop & Warehouse Facilities

A truck maintenance facility that will service the mining fleet is located west of the open pit and southwest of the process plant. For the initial 52.5 kt/d operation, only six truck bays will be required. During expansion, four additional truck bays will be added to meet the increase to the mining fleet. The fleet continues to expand as the length of hauls increases due to deepening of the pit and the facility will be progressively expanded to a total of 20 bays. The building type will be structural steel and covered in architectural cladding. The tire yard is located beside the truckshop.

The warehouse will house mechanical, electrical, instrumentation, and general items. The warehouse structure will be contiguous to the plant maintenance workshop. Internal offices will be supplied adjacent to the warehouse for warehouse and maintenance staff.

18.9 Assay Laboratory

An assay laboratory and metallurgical lab, as well as offices are located approximately 30 m west of the process plant building. The labs will process samples from the mining and exploration operations, as well as the process plant.

18.9.1 Administration Office Complex

A single-storey administration building is located near the main site entrance gate. The building will have a reception area, offices, meeting rooms, a main conference room, medical clinic, kitchenette and washrooms. The offices will be for managers, engineers, geologists, and clerks. A parking lot and transport and pick-up turnaround area are located adjacent to the administration building.

18.9.2 Sewage Treatment

The sewage treatment plant is located approximately 150 m northeast of the main administration building. The sewage sludge builds up at the bottom of the clarifier tank and is removed by a vacuum truck every six to nine months when full. The sludge is then transported and deposited into the municipal garbage dump landfill.

Treated sewer effluent is pumped to the process water storage pond.

18.10 Water Supply & Distribution

The process water storage pond (Figure 18.1) lies north of the tailings thickener and supplies the process plant with the majority of its water requirement. The process water pond is fed from overflow from the tailings thickener and concentrate thickener, as well as from return water from the TSF1 (TSF2 for the expansion phase). The water return HDPE pipeline feeding the process water pond is 24 inches in diameter and approximately 4 km long.

The process water pond is designed for a volume of approximately 20,000 m^3 and a two-hour retention time for the 52.5 kt/d case. For expansion to 105 kt/d, a second process water pond of the same size is added.

18.10.1 Raw Water

Raw water is retrieved from the SER and pumped to the raw water storage tank located adjacent to the tailings thickener. From the raw water storage tank, the raw water is pumped to various users throughout the process plant, including the reagent area and all pump gland seals.

18.10.2 Potable Water

Fresh water will be supplied by local wells and will be treated with a reverse osmosis unit to produce potable water for drinking, cooking and showers. It will also be used for emergency shower and eyewash stations throughout the plant. The reverse osmosis concentrate (brine retentate) is pumped to a local area sump and periodically pumped back into the process circuit.

18.10.3 Fire Water

Fire water is contained in the raw water storage tank. The total volume of the tank is 2,500 m^3, of which 1,000 m^3 is designated for fire water and 1,500 m^3 for raw water distribution. Level controls will assure that the level of the tank does not fall below the 1,000 m^3 volume mark.

During expansion to 105 kt/d, a second, smaller raw water storage tank will be added, providing an additional 1,500 m^3 in volume.

18.11 Fuel Supply, Storage & Distribution

The initial 52.5 kt/d maximum diesel fuel consumption will be 50,000 L/d and increases steadily to 122,000 L/d at the time of expansion. The fuel farm has been sized to allow for surge. It is recommended that approximately one week storage be provided for a total of 854,000 L. The diesel fuel is required primarily for the mining fleet. A single diesel fuel tank volume is 150,000 L; therefore, six tanks will be required every week for periods of maximum consumption.

The diesel fuel tanks will be above-ground, horizontal, cylindrical tanks inside a rectangular secondary containment casing. The tankers can be unloaded and loaded three times each week, as per the rail schedule at the fuel delivery track. The fuel tanks and fuel dispensing pumps are located adjacent to the truck maintenance facility for easy access to the mining fleet.

In addition, there is one 35,000 L regular gasoline double-walled storage tank for cars, pickup trucks, and other site vehicles.

After the plant expansion to 105 kt/d, the fuel farm will be expanded to eleven tanks (1,650,000 L), providing over five days of storage for the year of peak consumption.

18.12 Transportation & Shipping

The concentrate loadout area is located at the north end of the process plant. Concentrate is stockpiled adjacent to the concentrate loadout rail spur. The nickel concentrate is loaded onto rail cars using a front-end loader (FEL). Fibreglass rail car covers are easily removed with a mobile crane and placed south of the rail spur during loading procedures, and quickly bolted back into place on completion. These will be loaded at the plant site or a transfer facility from a stockpile with FELs. They will be unloaded either in Sudbury with overhead mechanical scoops, or at the Port of Quebec.

Nickel concentrate initial peak throughput will be 142 kt/a (based on 16 t/h nominal plant capacity and 92% availability). Based on a minimum of three services per week, outgoing traffic will consist of ten 99 tonne wagons to be loaded over two days. A normal FEL (e.g., CAT 980) will have a productivity of 300 t/h. Therefore, only three to four hours of operation every two days will be required to load ten wagons.

A trade-off study was conducted to compare the costs of transporting nickel concentrate by truck and by rail. It was decided to include a rail spur, although the cheapest option was a truck-rail combination, where concentrate is trucked to an existing transfer facility in Rouyn-Noranda for furtherance by rail to Sudbury. The desire to have an option of sending concentrate to Quebec City necessitated the rail spur, since trucking that far is much more expensive. Other deciding factors were the fact that the rail spur can also utilized to deliver fuel, reagents and consumables, and explosives supplies. The emulsion plant is not installed until Year 2, and until then is trucked to site, likely from existing facilities in Malartic or Val d'Or, both of which are within 100 km.

Concentrate that is sent to the Port of Quebec by rail would be transferred via ship to a port in China or Finland.

18.13 Construction Camps

A permanent mining camp will not be required. All labour can be sourced or housed in Amos and within the Abitibi-Témiscamingue region.

18.14 Site Security

All entrants to the plant site must pass through the security guardhouse located at the front gate. The entrance to the site, separating the plant site from Highway 111, is fenced with approximately 5.5 km of chain-link security fencing. The explosives area and emulsion plant (built during the expansion phase) is also fenced for security. A locked gate blocks the road from Launay from the explosives area, as shown on Figure 18.1. The plant site is not fenced on the western, eastern and northern sides to allow access for snowmobiling.

18.15 Communications

18.15.1 Enterprise Ethernet Networking

The Enterprise Ethernet Network system will include all the necessary cabling, router, firewall and accessories required to transmit data within the plant, as well as provide communication with the external links.

IT rooms in the administrative building will contain equipment for off-site communication. Other equipment, such as a patch panel and repeater, will be located in remote electrical rooms or in local communication cabinets.

Restricted access to the IT room will be enforced by means of access control cards and video monitoring.

Firewalls and routers will allow communication within the different systems and users within the premises, while preventing intrusion to sensible data from outside. System servers will be used to collect and save data from the different systems.

The administrative network, by means of dedicated fibre optic and Cat6 cables, will service all major buildings to support telephone, intercom, process CCTV, and access systems, as well as providing a link from the process network to the external internet.

The process network, by means of redundant dedicated fibre optic cables and copper cabling, will service all the buildings where process control equipment is located.

18.15.2 Process Control System

The process control system will consist of a redundant operation station located in the main control room. Other non-redundant control stations will be located in each electrical room.

Process controllers, input/output (I/O) cabinets and human-machine interface (HMI) will be located in electrical rooms or control cabinets as part of the equipment package (e.g., crusher, blower and air compressor systems).

Communication between the processor and remote I/O cabinet will be redundant; communication with other equipment—such as the package controller, MCC and switchgear—will be non-redundant.

18.15.3 Telephone & Intercom System

The telephone and intercom system will allow direct communication between different areas and buildings throughout the plant.

The intercom or public announcement equipment will be installed in noisy areas or outside of buildings, where a telephone set is not practical.

The telephone and intercom systems will use IP addressing. The telephone management system will provide functions such as call directory, forwarding, messaging, usage statistics, call transferring, etc.

18.16 Surface Water Management System

18.16.1 Water Management Plan

The water management plan must facilitate the operation of the mine development through a wide range of climatic conditions, while at the same time protecting the environment. The prime objectives of the water management plan will be to:

- provide a reliable water supply to the concentrator;
- facilitate mining of the ore deposit by limiting inflows to the open pit and by timely removal of groundwater discharges and precipitation falling on the incremental catchment of the open pit;
- provide sediment control;
- collect and treat contact water that would otherwise impair water quality of receiving streams; and
- protect mine infrastructure during extreme flood events.

The water management plan revolves around changes throughout the mine life, which is broken into five main phases:

- Phase 1—Construction
- Phase 2—Low Ore Production
- Phase 3—High Ore Production
- Phase 4—Milling Low-grade Ore Stockpiles
- Phase 5—Closure.

Each phase incorporates diversion structures, sump and pump systems, sedimentation ponds, and reservoirs that manage contact and impacted contact water separately as the overall surface area or footprint of the mine expands.

18.16.2 Contact Water Diversions

All surface water runoff which comes in contact with disturbed areas, other than tailings, is considered to be contact water. The contact water will require removal of high suspended sediment as a treatment procedure. This water includes any runoff from the waste rock, overburden or low-grade ore stockpiles, and water pumped from the open pit.

Development of the ore deposit will require the diversions of both West Creek and East Creek around the ultimate footprint of the open pit. Because the drainage areas of these two streams are not extremely large, the diversions could be implemented using either a pump and pipeline system or an open channel.

The East and West Creek will be replaced by three major open channels, which will route surface water away from the open pit and towards the southeast reservoir. Two channels will be located east of the waste rock and overburden stockpiles, identified as the north and south waste rock channels, which will collect runoff from the stockpiles and prevent sediment-laden water from entering the Arctic watershed. The third channel will be located between the eastern edge of the open pit and the western edge of the waste rock and overburden piles, identified as the east pit channel.

A total of 14 sumps will be situated in low elevation areas throughout the East Creek and West Creek catchment area, where water flow by gravity conveyance is not feasible. Each sump collects a combination of surface water runoff from various site facilities, seepage from the tailings dams, and from direct precipitation, and will be implemented at various times throughout the development of the mine site. Seven of the sumps will collect non-impacted contact water, and will be pumped to one of the three major channels and ultimately end up in the southeast reservoir.

The contact water will be collected in the southeast reservoir, where it can be accessed by the concentrator for reclaim or as a raw water source. The open pit will also pump water to the southeast reservoir through an oil separator.

18.16.3 Impacted Contact Water Diversions

Surface water runoff that comes into contact with tailings is considered “impacted contact water.” Any impacted contact water excess will be treated in the water treatment plant, separate from the contact water. This water, which includes runoff and seepage from the tailings dams, will be collected by a network of channels and seven sumps situated around the TSF cells, and will be pumped back into the active TSF. The concentrator will reclaim as much of the TSF water as possible to minimize the requirement for treatment of the impacted contact water.

18.16.4 Sedimentation Pond

Sedimentation ponds 1 and 2 are located south of the TSF and are assumed for modelling purposes to have capacities of approximately 1 Mm^3 each. Excess water from these ponds reports either to the water treatment plant or Villemontel River. The sedimentation ponds will be in place early in the construction phase and will capture and treat disturbed area runoff for high suspended solids throughout the construction phase.

Sedimentation pond 1 is located east of the lower reach of the Unnamed Creek, and receives excess water from the TSF via sump 6, along with local runoff and direct precipitation. The local runoff may contain high sediment water during the construction phase from the disturbed areas. Water from sedimentation pond 1 is assumed to be impacted contact water upon the activation of TSF cell 1, and is the feed pond for the water treatment plant. Excess water in sedimentation pond 1 is pumped back to the active tailings facility for storage and potential reclaim.

Sedimentation pond 2 receives excess water from the open pit area during construction, excess water from the southeast reservoir prior to the start of LGO stockpile milling, water from sump 9 during construction, local runoff and direct precipitation. The pond will treat for high TSS (total suspended solids) concentrations. In addition, a CO_2 sparging system will be installed within the sedimentation pond to treat the water for high pH, before discharging to the Villemontel River.

The sedimentation ponds were sized for the 1:10-year return period flows and for a sediment threshold value of 0.01 mm. In order to minimize their footprint within the lower mine area, the depth of the sedimentation ponds has been set at 6 m, and the length to width ratio is 3 to 1, respectively. Both ponds are situated south of the railway and open pit, and north of the administration building.

18.16.5 Tailings Storage Facility

The TSF will serve two key roles in the management of water at the mine. Firstly, runoff generated within the catchment of the TSF, available water within the tailings slurry, and the associated TSF seepage collection system will provide an important source of water for the concentrator. Secondly, the pond within the TSF will serve as the single largest live storage for water at the mine during operations. A large live storage will be required to meet the demand of the concentrator (i.e., temporarily store water during wet periods for subsequent use in the concentrator during dry periods). If excess water exists in the system, then the pond can be relieved by discharging the excess for treatment.

Excess water from cells 1 and 2 will be pumped to the concentrator or to sump 6, depending upon concentrator's water demand. From Sump 6, water will be conveyed to sedimentation pond 1 and subsequently to the water treatment plant prior to discharge to the Villemontel River.

The largest inflow to the TSF will be the tailings slurry. The largest withdrawal will be outputs of reclaimed water for the concentrator. Water inflows to the TSF will also include local runoff, pumped flows from the surrounding sumps and direct precipitation. Other outflows include evaporation, seepage to groundwater and loss of water to tailings voids.

Freeboard for TSF cell 1 is 1.0 m and the freeboard for TSF cell 2 is 2.0 m. When the water level in the TSF exceeds the freeboard level, the excess water pumps are activated, and the water is drawn down until the water level has dropped below the allowable water limit, which is set to 1.0 m below the freeboard limit.

Most of the water which will seep through the base of the TSF will flow to the open pit. TSF seepage that may flow west towards the Launay esker, north towards the Chicobi River or south towards the Villemontel River has been modelled by Golder and results were presented in the report entitled Solute Transport Modelling of Tailings Storage Facility, RNC Dumont Project, Quebec (Golder, 2013b).

Three two-dimensional cross-sectional models were constructed to represent the groundwater flow paths between the TSF and the potential receptors: the Launay Esker to the west; the Villemontel River to the south; and, the Chicobi River to the north. Arsenic, chloride, and nitrite were identified as species of interest for the transport models, based on their anticipated concentrations in tailings pond water relative to the applicable groundwater criteria. Contaminant transport simulations were completed for both operations and post operations conditions. Various simulations were completed for each cross-section to evaluate the sensitivity of model results to various factors. The numerical modelling results demonstrate that the proposed design of the TSF will not affect compliance with the groundwater protection objectives at the potential receptors, as outlined in Directive 019 (Golder, 2013b).

Further details are provided in Section 18.7.8.

18.16.6 Collection System for Waste Rock Dump Runoff

Preliminary geochemical analyses indicate that the waste rock and the low-grade ore stockpiles will not be acid generating and will not require treatment in a water treatment plant before being discharged offsite. Channels will be constructed along the outer limits of the stockpiles to capture sediment-laden runoff water and route surface water flows to a network of sumps and reservoirs (Figure 18.1).

East of the waste rock and overburden piles, two channels will route runoff to the southeast reservoir, the north waste dump channel and the south waste dump channel. West of the waste rock and overburden piles, the east pit channel will also route runoff to the southeast reservoir.

The LGO stockpiles will evolve in size and shape over time across the ground surfaces that are north of the pit. Three sumps will collect runoff from around these areas and will discharge water to the east pit channel, and eventually to the southeast reservoir.

It is assumed that all of the infiltration into the stockpiles from rainfall will eventually report to the southeast reservoir.

18.16.7 Water Treatment

Impacted contact water is directed to the sedimentation pond 1. This includes excess water from the TSF, which is collected in Sump 6 prior to the sedimentation pond 1, TSF dam seepage collected in ditches along the southeast corner of the TSF, and local runoff.

A CO_2 sparging system is adjacent to sedimentation pond 2 to treat high pH values. The sparging system consists of a CO_2 pressurized tank, a pipe manifold and piping which extends to the sedimentation pond. If pH levels are high, CO_2 sparging system is activated (bubbling) to reduce it to meet environmental standards. Discharge of treated water is released to the Villemontel River.

After the expansion phase, in 2022, a water treatment plant will be installed due to possible high levels of arsenic and other elements. The water will only need to be treated when the water level in the TSF's cells exceeds the specified freeboard level. The withdrawal rate from the sedimentation pond 1 will be equivalent to the WTP capacity, which was optimized with the water balance model to prevent uncontrolled discharges of impacted contact water to the Villemontel River. The plant will operate at a capacity of 0.7 m^3/s and will discharge via pipeline to the Villemontel River as required.

19 MARKET STUDIES & CONTRACTS

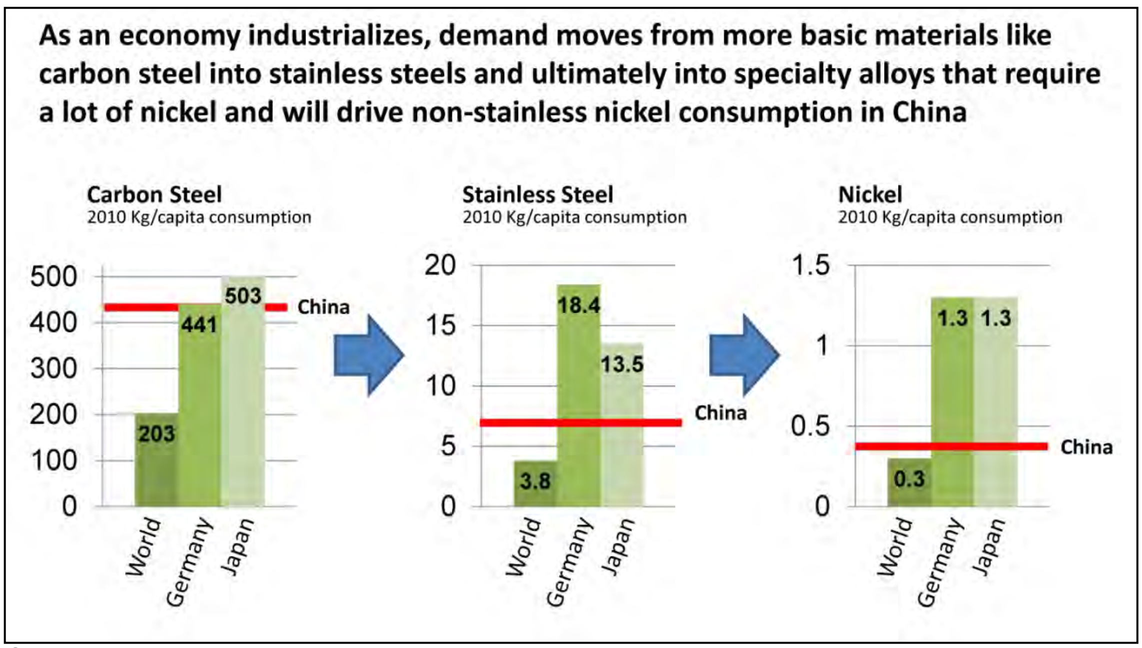
19.1 Nickel & Stainless Steel Market Outlook

According to the long-term outlook (March 2013) by the Metals Market Service, Wood Mackenzie forecasts that global nickel consumption will increase by 6.4% in 2013 to 1.83 Mt; and by 4.7% per year to 2.1 Mt in 2016; and 2.7% per year thereafter to 2.74 Mt in 2030. Both of the two main consumption sectors, stainless and non-stainless, are expected to grow strongly in the future.

Primary nickel demand in stainless steel is projected to increase to just under 2 Mt by 2030 driven by growth in global stainless melt output of 5% per annum to 44.6 Mt until 2017 with further growth of 3% per annum to 65.4 Mt until 2030. The bulk of this growth will be supported by the continued expansion of the Chinese stainless steel industry.

RNC management believes that potential for growth in nickel is even higher than forecasted by Wood Mackenzie driven by the Chinese economy as it continues to evolve towards value-added products. As shown in the chart below, if nickel demand in China is able to reach similar per capita consumption levels as industrial economies like Germany and Japan as China has been able to achieve in carbon steel, it would add more than 1 Mt of nickel demand per year to a total of 1.6 Mt per year by 2020. Given that carbon steel demand in China continues to grow and now exceeds Germany and Japan per capita consumption levels, the potential for growth in Chinese nickel demand could be even higher than the 1Mt currently forecast.

Figure 19.1: Beyond 2015 – Evolution of Nickel Demand per Capita



Source: World Steel Association, ICSG, World Stainless Steel Statistics, Brook Hunt – A Wood Mackenzie Company, RNC Analysis.

19.2 Price Assumptions

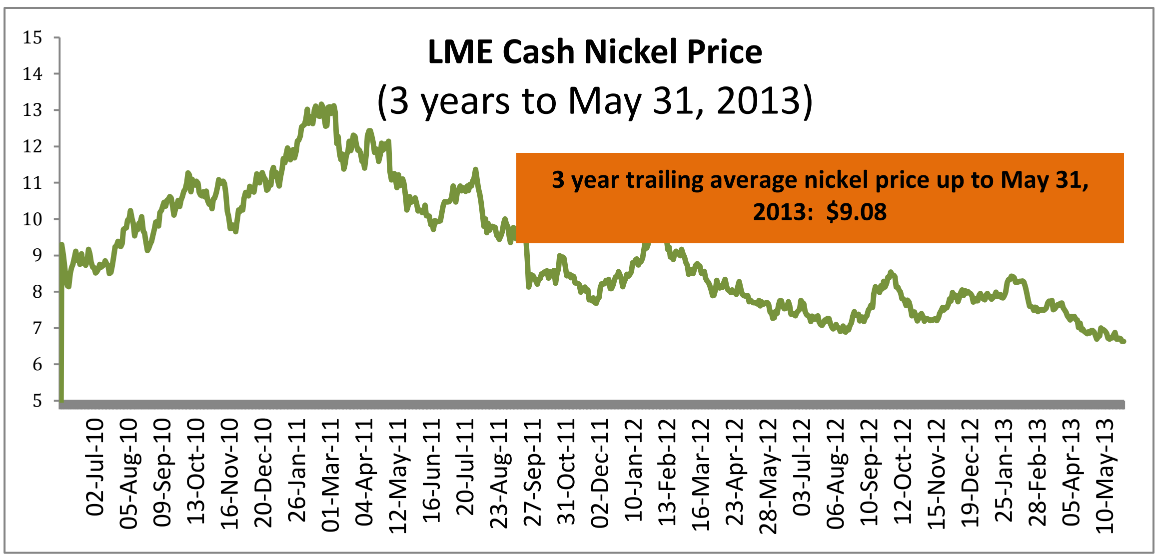
Pricing assumptions were developed for nickel and the cobalt, platinum, and palladium byproducts contained in the Dumont concentrate based on forecasts as of May 31, 2013 from the four analysts, of the five analysts who cover RNC, who publish commodity price forecasts. As the expected timing for the project, now falls within the short-term forecast of those analysts, annual price forecasts for nickel are used for 2015, 2016, and 2017 with a single long-term forecast for the remaining years. For the byproduct metals – cobalt, palladium, and platinum, a single price was used for each year. Table 19-1 summarizes the pricing assumptions.

Table 19-1: Pricing Assumptions in USD

		2015	2016	2017	Long-Term
Nickel	US$/lb	$10.00	$10.00	$10.50	$9.00
Cobalt	US$/lb	$14.00	$14.00	$14.00	$14.00
Platinum	US$/oz	$1,800	$1,800	$1,800	$1,800
Palladium	US$/oz	$700	$700	$700	$700

A long-term nickel price assumption of $9.00 per pound was utilized in the study which is consistent with the average long-term nickel price of $9.30 per pound forecast by the four analysts and the three-year trailing average nickel price to 31 May 2013 which averaged $9.08 per pound.

Figure 19.2: LME Quarterly Cash Nickel Price



The metal price assumption for platinum of $1,800 per ounce was consistent with the average RNC analyst forecasts for the long-term of $1,793 per ounce and a 2015-2017 range of $1,853 to $1.877 per ounce. The metal price assumption for palladium of $700 per ounce for palladium was consistent with the average RNC analyst forecast for the long-term of $667 per ounce and a 2015-2017 range of $712 to $775 per ounce. The metal price assumption for cobalt of $14 per pound was consistent with the average RNC analyst forecasts for the long-term of $13.88 per pound and a 2015-2017 range of $14.17 to $14.29 per pound. All sensitivities for these pricing assumptions are provided in Section 22.

19.3 Concentrate Marketing

The Dumont concentrate, which will have an average nickel content of 29% nickel over the life of project and recoverable quantities of cobalt, platinum, and palladium, is expected to be among the highest grade nickel concentrates in the world which should make it a desirable product to nickel smelters globally. The MgO content of this concentrate is expected to be between 7% and 10%, which is in line with the MgO content in concentrates produced by other ultramafic operations.

Assumptions regarding commercial terms for this concentrate have been based on benchmark rates and include:

- percentage payable of 93% nickel;
- base treatment charge of $150/t, with an additional penalty of $25/t of concentrate for the MgO content;
- base refining charge of $0.70/lb of nickel;
- price participation of 10% with a base price of $8.00/lb;
- payable percentage on contained cobalt of 50% and a refining charge of $3/lb; and
- payable percentage on contained platinum and palladium based on a 1 g/t deduction, and average 77% for the concentrate grade of 4.3 g/t PGE over the over the life of project with a refining charge of $50/oz.

The concentrate will be transported by existing road, rail and port facilities to the smelters. In the feasibility study, 50% of the concentrate is assumed to be processed by the Sudbury smelters at a transportation cost of $41/t. The remaining 50% of the concentrate will be transported to Quebec City at a cost of $36/t with half of the concentrate (25% of total) shipped to a smelter in Finland at a transportation cost of US$40/t, and the remaining half of the concentrate (25% of the total) shipped to smelters in China at a transportation cost of US$79/t. All sensitivities for these pricing assumptions are provided in Section 22.

Forward sales metal prices in terms of potential contracts are based on experience from actual similar operations and general knowledge. Unlike copper and other metals, there are no benchmark pricing for treatment and refining charges. No direct marketing has been done for the potential Dumont concentrates and therefore no off-take agreements exist. The final distribution and pricing terms on the Dumont concentrate will be determined only once the off-take agreements have been signed.

Based on current industry demands, it is envisioned that the nickel concentrates would be best suited for several smelters—two smelters in Canada owned by Vale and Xstrata, the Harjavlta smelter in Finland owned by Boliden, and the Jinchuan smelter in China. The company has also completed a separate study that demonstrates the viability of processing the concentrate through a combination of a fluid bed roaster and electric arc furnace and recently announced a memorandum of understanding (MOU) with Tsingshan Holding Group Co., Ltd. The MOU sets out the objectives of the two companies to work together in relation to downstream concentrate processing and the potential to enter into an offtake and/or partnership arrangement with respect to the Dumont project. After working in cooperation with RNC for more than a year, Tsingshan completed its own analysis and testwork on a sulphide nickel concentrate (utilizing a process similar to the one previously announced by RNC in the news release dated October 3, 2011) in its integrated NPI/stainless steel production facilities and plans to make the necessary investment in plant and equipment once concentrate feed is secured. This innovation represents the first time that nickel sulphide concentrate would be used directly to create stainless steel. This contemplated plant is also expected to be capable of handling nickel sulphide concentrate like that anticipated from Dumont.

Overviews of the various smelters that are believed to have the highest potential to process the Dumont concentrate are provided in the subsequent section.

19.4 Smelter Options

There are currently 11 nickel smelters globally, while a twelfth unit that will also treat sulphide concentrates is under construction (the Vale facility in Newfoundland). Brief profiles of the most likely smelters are provided in the subsections below.

19.4.1 Xstrata

The Xstrata smelter located in Falconbridge (a suburb of Sudbury) currently treats concentrates produced by Xstrata's operations located in the Sudbury basin (the bulk coming from the Nickel Rim South mine) and in Quebec (Raglan), as well as from third parties.

The smelter uses electric furnace technology, which is more suitable for treating concentrates containing elevated levels of MgO. The average MgO content of feed is understood to be higher than the MgO level of feed treated at Vale's Copper Cliff facility, but is primarily due to Raglan, and should thus decrease as production from Nickel Rim South ramps up. Due to the depletion of other Sudbury basin mines and limited additional third party concentrates, the smelter is believed to be currently operating less than full capacity of 76 kt/a. It is believed that Xstrata has no significant sulphide mine development projects underway to provide additional concentrate feed in the next 5 to 7 years.

Matte produced by the Falconbridge smelter is shipped to the Nikkelverk refinery in Norway. Overall cobalt recovery through the smelter and refinery is approximately 70%.

19.4.2 Vale

Vale's main smelter is located at Copper Cliff, which is another suburb of Sudbury. The smelter uses flash smelting technology, which is less suitable for treating concentrates containing elevated levels of MgO. However, the large capacity of the facility coupled with the high Ni grade of Dumont concentrate would result in concentrates from Dumont representing a small portion of the total feed tonnage. Furthermore, Vale's own Sudbury basin mines typically produce concentrates with low MgO. As a result, it should be possible to treat Dumont concentrate at Copper Cliff without exceeding MgO limits.

19.4.3 Boliden/Norilsk

Boliden currently operates the Harjavalta flash smelter in Finland. Harjavalta is part of a polymetallic complex that treats separate copper and nickel concentrates. Output from the smelter is refined at the adjacent Harjavalta Refinery, which is owned by Norilsk. The Harjavalta smelter has a capacity of ~40 kt/a of contained nickel and is understood to be operating at significantly less than design levels. It would thus have capacity for a significant percentage of Dumont concentrate. It is understood that the smelter can be expanded by converting the copper processing to nickel processing for a relatively minimal capital investment. The smelter can accommodate some quantity of MgO bearing concentrates. The Harjavalta refinery owned by Norilsk has a capacity of ~ 65 kt/a and is beginning to receive direct intermediate feeds from Talvivaara. The complex achieves high recoveries for cobalt.

19.4.4 Jinchuan

Jinchuan operates an integrated smelting and refining facility in Gansu Province, China.

The smelter currently has a capacity of ~120 kt/a contained nickel, while the refinery has a capacity of ~150 kt/a contained nickel. Over 40% of the concentrate feed to the Jinchuan smelter currently comes from third party sources. Given our understanding of their mine production profile, Jinchuan will have the capability to take MgO bearing feeds and will continue to need third-party concentrates to fill its smelting and refining capacity.

20 ENVIRONMENTAL STUDIES, PERMITTING & COMMUNITY IMPACT

The information presented in this section originates principally from the Environmental and Social Impact Assessment (ESIA) performed as part as the Dumont project permitting process and integrates a number of studies performed by RNC and its consultants over the past five years. Biophysical data came mainly from three distinct fieldwork programs performed from 2007 to 2009, with some complementary information extracted from the ongoing baseline studies designed to support the Environmental and Social Impact Assessment in 2011 and 2012. Table 20-1 summarizes the sources of information for the various biophysical and social components described in this report.

Table 20-1: Sources of Biophysical & Social Components included in the Pre-feasibility Study

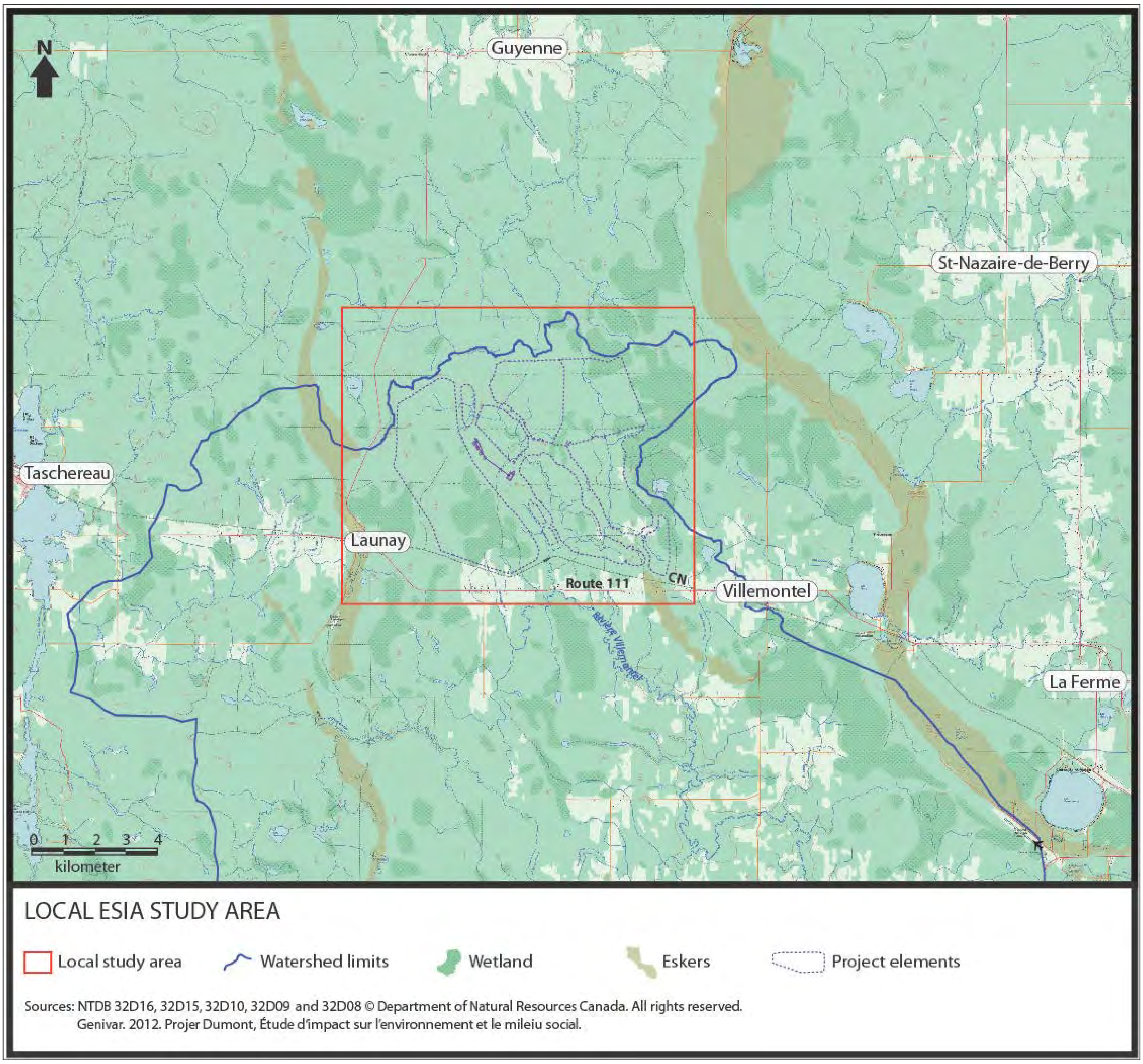
Type of Study	2007[1]	2008[2]	2009[3]	2011[4]	2012
Water and sediment quality	√	√	√	√	
Groundwater quality					√[6]
Vegetation and wetlands		√		√	
Wildlife	√	√	√		
Small mammals				√	
Fish	√	√	√	√	√[6]
Benthic invertebrates	√	√	√		
Birds		√		√	
Reptiles and amphibians				√	
Archaeology		√			
Stakeholders consultation				√[5]	√[7]

Notes: 1. Ménard et Coppola (2008). 2. GENIVAR (2009). 3. GENIVAR (2010). 4. Unpublished data. 5. Transfert Environnement (2011). 6. GENIVAR (2012) 7. Transfert Environnement (2013) **Source:** RNC.

The study zone for the ESIA encompasses an area larger than the footprint of the project, as shown in Figure 20.1.

Ausenco

Figure 20.1: ESIA Local Study Area



Source: RNC.

20.1 Description of Biophysical Components

20.1.1 Climate

The climate at the Dumont property is continental; mean temperatures ranging from -17.3°C in January to +17.2°C in July, with an annual mean temperature of 1.2°C. Annual precipitation totals about 918 mm: 670 mm of rain and 248 cm of snow. Mean calculated evaporation from lakes ranges from 2.0 to 4.2 mm for the months of June to September inclusively. A weather station installed onsite since June 2011 recorded wind speed ranging from 0 to 10 km/h, with gust speed peaking at 28 km/h. The average wind direction corresponds to a northwestern wind.

20.1.2 Drainage System & Hydrology

The local study zone is located in the St. Lawrence River watershed, which includes the Villemontel and Kinojévis Rivers. It is at the boundary with the James Bay watershed.

The vast majority of the study zone drains into the Villemontel River. This river connects with the Kinojévis River, which flows into the Ottawa River in the St. Lawrence watershed. The slope of the Villemontel River, between its confluence with unnamed stream 1 and the zone of influence of the Kinojévis River (27.9 km downstream), is 0.03%, representing an elevation drop of only 8.8 m between these two points. It flows in steps—a succession of waterbodies of constant elevation controlled by sills or beaver dams. During the month of August 2012, the streamflow measured in the Villemontel River ranged from 0.3 to 0.5 m^3/s (severe low water level).

Unnamed stream 1, a tributary of the Villemontel River, is the principal watercourse that will be affected by the project. At its mouth, where it empties into the Villemontel River, unnamed stream 1 drains a total area of 50 km^2. The average slope of this watercourse is 0.3%.

Two other watercourses, Ruisseau Paré and unnamed stream 2, are found in the study zone. These streams empty directly into the Villemontel River, just upstream from unnamed stream 1.

20.1.3 Hydrogeology

Four hydrostratigraphic units were identified in the study zone:

- glaciolacustrine deposits;
- fluvioglacial deposits;
- tills; and
- bedrock.

The fluvioglacial deposits are concentrated in the eskers, which form elongated sand deposits, all oriented in a northwest/southeast direction. With respect to the Dumont project, they are found to the west (Launay esker), at its centre (unnamed esker) and to the east (Saint-Mathieu-Berry esker).

Two major aquifer eskers, the Launay and Saint-Mathieu-Berry eskers, are exposed at surface in the study zone and in neighbouring areas. A third significantly smaller fluvioglacial deposit, the unnamed esker, borders the southern part of the study zone and is adjacent to the projected pit.

The water in the bedrock and the overburden of the study zone, other than the water in the eskers, is considered to come from Class II hydrogeological formations according to the MDDEFP's classification scheme (MDDEFP,1999), as they are used only locally to supply water to private properties along Highway 111. The Launay and unnamed eskers are Class I hydrogeological formations. These formations can supply a sufficient quantity of water of satisfactory quality and, in case of need, could constitute a source of supply for a community.

The groundwater in the overburden and the bedrock generally flows in the same directions: from northwest to southeast in the western part of the study zone and from north to south in the eastern part. Flow directions are consistent with local topography and surface waterflow.

The groundwater level is generally near the surface of the soil, at a depth of less than one metre, except in the areas of the unnamed and Launay eskers, where the piezometric level is deeper.

Groundwater velocities are around 0.6 m/a to 1.1 m/a in the overburden and 7.8 m/a to 15.3 m/a in the near-surface bedrock. The flow velocities do not exceed 0.06 m/a in the deep bedrock

20.1.4 Groundwater Quality

The groundwater quality in the study zone is generally good. Only a few of the analyzed parameters show exceedances, sometimes point-source exceedances, of the seepage in surface water and stormwater system criteria (RESIE) or of the criteria for drinking water (CESAFC), and then only in certain observation wells. These parameters are arsenic, copper, manganese, nickel, zinc and pH.

20.1.5 Surface Water Quality

In general, the surface water of the local study zone is slightly alkaline (pH most often slightly higher than 7.0) and moderately hard (total hardness most often between 17 and 57 mg/L). It is rich in organic carbon, which is mainly found in dissolved form, at concentrations ranging between 4 and 28 mg/L. The turbidity is highly variable from one station to another, and high values, reaching nearly 30 nephelometric turbidity units (NTU), were measured in some samples.

The sampled stations are separated into three distinct groups based on surface water characteristics. The Villemontel River differs from the other watercourses in several regards. Its water is harder and its major ion concentration is higher, which translates into a specific conductance about twice as high as other stations.

Among the measured nutrients, the total phosphorus concentrations are sometimes very high (up to 0.10 mg/L), frequently exceeding the criterion proposed by the Canadian Council of Ministers of the Environment and the MDDEFP, which is intended to prevent eutrophization of water bodies. Exceedances were observed at all stations, which is evidence of eutrophic aquatic environments.

Among the measured metals, the aluminium concentrations are especially high. They generally exceed the MDDEFP's chronic aquatic life toxicity criterion of 0.087 mg/L. In November 2009, they also exceeded the acute aquatic life toxicity criterion (0.75 mg/L) in five out of seven samples. The exceedance of the chronic aquatic life toxicity criterion is not rare in northern Québec, but the exceedance of the acute aquatic life toxicity criterion is less frequent.

The iron concentrations regularly exceed chronic aquatic life toxicity criterion.

20.1.6 Sediment Quality

The total chromium concentration in the sediments generally exceeds the rare effect level (REL) of the Québec criteria, for all sampling years. In addition, the threshold effect level (TEL) and the Canadian guideline were exceeded in nearly 50% of the samples. High chromium concentrations capable of producing harmful effects on organisms are frequently measured in the soils and sediments derived from serpentine, a family of minerals frequently found in the local study zone.

Other criteria exceedances were observed, albeit more rarely, for cadmium, copper and lead. These exceedances mainly come from Lac à la Savane.

20.1.7 Soils

As part of the ESIA, an environmental site assessment of past uses of the land covered by the Dumont property was performed in order to identify all the elements that could have posed a real or potential risk of contamination to soil and water. This study concluded that although the site was bordered by a sawmill and a railway there a no evidence that the site has been contaminated by past activities. A soil characterization program is planned for the latter part of 2013 to evaluate baseline conditions prior to project implementation. Geochemical characterization of the overburden that will be manipulated and stockpiled was performed in 2012 and results are presented in Section 1.7 of this report.

20.1.8 Vegetation & Wetlands

Throughout the local study zone, terrestrial environments cover 39% of the surface area (3,786 ha), while wetlands occupy 57% (5,540 ha). The remainder is composed of anthropogenic environments, such as agricultural fields and housing (399 ha; 4%). The terrestrial environments comprise 17 main types of vegetation, including deciduous (9%), mixed (15%), and coniferous (46%) stands, as well as other types of terrestrial environments (30%), such as uncultivated grassland. Recent cutting has fragmented several environments.

The majority of the terrestrial environments have medium ecological value. However, intolerant deciduous trees, uncultivated grassland, scrubland and recent cuttings have low ecological value. The anthropogenic environments have an ecological value ranging from low to very low.

Small areas of black spruce and jack pine stands have high ecological value. The black spruce stands are located in the bog east of Launay. They form thin forest strips, surrounded by open bog of high ecological value. Together with the bog, they form a diversity of interesting plant habitats. The jack pine stands contain Woolly Beachheather and Sand Jointweed, two special-status plants with high ecological value. This area is highly valued by the population and will not be disturbed by the Dumont project.

Open bogs and tree swamps represent 65% of all wetlands in the local study zone. Wooded bogs and shrub swamps account for 34%. Finally, associated ponds and marshes represent about 1% of the wetlands. The majority of the wetlands have medium ecological value. Two open bogs have high ecological value and one bog-pool system has very high ecological value.

All of the habitats within the studied area have been thoroughly characterized resulting in more than 150 descriptive listings.

20.1.9 Mammals

There is considerable wildlife diversity in the surroundings of the Dumont project, which is due to the Abitibi-Témiscamingue region’s sub-northern climate. This is a transition zone where species from the north and south can be found. Trapping and hunting data (2007-2008) from the MNRF indicated the presence of a broad range of animals in the greater Abitibi-Témiscamingue region that are likely to inhabit the study area. The list includes beaver, muskrat, red squirrel, white-tailed deer, moose, raccoon, striped skunk, Canada lynx, red fox, coyote, grey wolf, black bear, river otter, marten, weasel, fisher and mink. Surveys conducted onsite confirmed the presence of moose, wolf, black bears, beaver, groundhog, red squirrel and snowshoe hare.

According to the Quebec Ministry of Natural Resources and Wildlife (MRNF), Lac à la Savane Lake, slightly outside of the Dumont property, is considered a protected muskrat habitat.

20.1.10 Small Mammals

A field survey designed under the MNRF micro mammals protocol was performed in September 2011 in various habitats within the studied area. Preliminary data analysis indicated the presence of rock vole (*Microtus chrotorrhinus*), a species likely to be listed on Quebec's threatened or vulnerable species list. Only one specimen was captured in its preferred habitat, a mature mixed forest located on rock outcrops. This habitat, located west of Lac à la Savanne, will not be affected by the mine infrastructure. Habitat developments to promote the rock vole will be performed in the Lac à la Savane sector and/or west of the projected tailings storage facility, where individuals of this species have been captured. This measure was included in the ESIA as part of the compensation program.

20.1.11 Fish

The inventories conducted between 2007 and 2012 counted 24 fish species in the watercourses of the study zone (Lac à la Savane, Lac Doyon, Lac Gauthier, and Ruisseau Pandini). Among these species, White Sucker, Brook Stickleback and Trout-perch are the most widespread.

In the Villemontel River, a few cyprinid species and larger-sized species, such as Rock Bass, Northern Pike, Walleye and Yellow Perch, were captured.

In the watercourses of the study zone, the inventories conducted in the habitats most prospective for Brook Trout did not capture any specimen of this species. The Villemontel River and its tributaries offer low habitat potential for this salmonid because the water is generally very turbid, the bed is composed of clay and silt, and the flow is mainly lentic.

20.1.12 Benthic Invertebrates

Benthic invertebrate surveys were conducted in 2007, 2008 and 2009.A total of 66 taxa were identified in the inventory, with an overall density of 1,300 organisms per square metre. Of this total, 33% of the taxa and 23% of the organisms belonged to the Chironomidae family, which includes midges or small flies related to mosquitoes whose larval stages are aquatic. These larvae are an important food source for fish and other insects, and the adult forms are an important food source for birds and bats. These insects are quite resistant to organic pollution.

20.1.13 Birds

According to provincial birdwatchers database (ÉPOQ), 112 bird species were indexed in the Launay and Trécesson area. Surveys conducted in 2008 within the Dumont property allowed census of 44 species. Complete inventories, pursued in 2011, comprising listening stations, active research and 12 automatic songbirds recording devices allowed census of more than 90 species, including more than 20 species not surveyed in the EPOQ database. The most common species are the Nashville Warbler and the White-throated Sparrow.

The absence of waterbodies and watercourses of significant size in the perimeter covered by the Dumont project suggest low potential for their use by aquatic birds such as waterfowl. In fact, only four common species were identified during the field surveys (black ducks, mallards, teal and loons).

20.1.14 Reptiles & Amphibians

The local study zone shelters a good diversity of anurans, with six species detected. These are common and widespread species in Québec: the Northern Spring Peeper, the Wood Frog, the American Toad, the Mink Frog, the Green Frog and the Leopard Frog. A few Common Garter Snakes were observed during the fieldwork. Inventories of Wood Turtle were performed in 2011, but none were observed. An additional inventory of Blanding's Turtle was conducted in spring 2013 and none were observed.

20.2 Species at Risk

20.2.1 Plants

Consultation of Quebec government species at risk database (CDPNQ) revealed no occurrence of "at risk" species within the study area. Colonies of sand heather (*Hudsonia tomentosa*), however, were mentioned by CDPNQ east and northeast of the future mine site. Field surveys conducted in 2008 confirmed the presence of these colonies but at that time no plants were observed inside the Dumont property limits. This plant is likely to be designated threatened or vulnerable in Quebec.

In June, July and August 2011, three field campaigns focusing primarily on approximately forty "at risk" plant species were conducted within the Dumont project study area. These inventories allowed census of three precarious species: slenderleaf sundew (*Drosera linearis*) located in a wetland (bog) on the northeast corner of the study area, sand heather (*Hudsonia tomentosa)* and sand jointweed (*Polygonella articulata*) on the southwest corner of the Dumont property. Current project development plans would not impact the areas where these species were observed.

20.2.2 Reptiles & Amphibians

In May 2011, a field survey aiming specifically at locating wood turtle (*Clemmys insculpta*) along the watercourses potentially impacted by mining infrastructure was conducted. No wood turtles were observed. This species is likely to be designated threatened or vulnerable in Quebec.

Furthermore, real time recordings performed on site between May and July 2011 did not detect any audio evidence of the presence of the striped chorus frog (*Pseudacris triseriata*) also likely to be designated threatened or vulnerable in Quebec.

As part of the provincial environmental assessment process, RNC was requested to perform a spring survey of the Blanding's Turtle (*Emydoidea blandingii*) to confirm the absence of this species on the Dumont property. This species is considered threatened by both provincial and federal governments. Although intensive efforts were put during the 2013 spring survey (16 stations during 10 consecutive days), no specimen were captured.

20.2.3 Birds

During 2008 field surveys, a great grey owl (*Strix nebulosa*) was observed within the study area, but nesting was not confirmed. No observation of this species was made during subsequent surveys.

Field inventories in 2011 recorded the presence of three "at risk" species: olive-sided flycatcher (*Contopus cooperi*), rusty blackbird (*Euphagus carolinus*), and common nighthawk (*Chordeiles minor*). These three species are considered likely to be designated threatened or vulnerable in Quebec and are considered threatened by the Committee on the Status of Endangered Wildlife in Canada (COSEWIC).

Among the species identified in the Launay and Trécesson area in the birdwatchers database (EPOQ), the presence of the short-eared owl (*Asio flammeus*) was noted. This species is of special concern in Canada and is likely to be designated threatened or vulnerable in Quebec. Also noted in this list is the bald eagle (*Haliaeetus leucocephalus*) designated vulnerable in Quebec, although its presence within the study area is unlikely due to the absence of large water bodies demonstrating fish abundance.

20.3 Description of the Social Environment

The Dumont project is located in the regional municipality of Abitibi. This territory is composed of 17 municipalities and two unorganized territories. The First Nation reserve of Pikogan is also located within this geographical area. The population of the MRC is approximately 24,300. Socio-economic indicators for the surrounding municipalities are given in Table 20-2.

The proposed extent of the Dumont project is located principally in the municipalities of Launay, and Trécesson with a minor extension into the municipality of Berry to the northeast. The villages of Launay and Villemontel are located along the road and railway line linking Amos and the next regional municipality, Abitibi-Ouest, whose nearest town is Taschereau. These villages were established when the transcontinental railroad was built during the early stages of colonization of the Abitibi area at the beginning of the 20th century.

Table 20-2: Socio-economic Indicators for Nearby Municipalities

	Amos	Berry	Launay	Pikogan	Taschereau	TNO Lac Chicobi	Trécesson	Province of Québec
Total population in 2011	12,671	625	229	538	981	203	1,138	7,903,001
Total population of 15 years and over in 2011	84%	77%	80%	67%	82%	77%	84%	84%
Area in 2011	431 km²	577 km²	258 km²	1 km²	251 km²	722 km²	197 km²	1,356,547 km²
Population density per square kilometre in 2011	29.4	1.1	0.9	538	3.9	0.3	5.8	5.6
Median age of the population in 2006	41 yrs	33.4 yrs	45.3 yrs	24.5 yrs	38.6 yrs	39 yrs	39.9 yrs	41 yrs
Employment rate in 2006	58.30%	60.70%	41%	56.90%	43%	50%	56.60%	60.40%
Unemployment rate	12%	13.30%	21.10%	14%	19%	10%	10%	7%
Total population 15 years and over without certificate, diploma or degree in 2006	32.47%	65.90%	66.70%	60.60%	49.70%	53.30%	32.30%	25%
Median income in 2005 – All private households ($)	45,804	49,793	N.D.	34,432	35,408	N.D.	53,824	46,419
Dwellings requiring major repair - as a % of total occupied private dwellings in 2006	7.10%	21.40%	10%	23.10%	20.50%	30.80%	15.70%	7.7%

Sources: Statistics Canada, 2006 Census of population - Community profiles. Statistics Canada, 2011 Census of population, adapted by the Quebec Institute of Statistics.

20.3.1 Description of Surrounding Communities

20.3.1.1 Amos

The town of Amos, located 25 km east of the Dumont project, is the largest town in the regional municipality with a population of over 12,500. Amos is the commercial and administrative centre of the region. It provides public services such as health care, school board administration, and sport infrastructures to the surrounding municipalities.

20.3.1.2 Launay

Launay's economy relies mainly on agriculture and forestry. There are 229 inhabitants and 112 private dwellings in the municipality. The majority of its territory is located on public (Crown) lands. The Dumont project development limits are about 2 km from the urbanized area of the municipality, which is located on the Launay esker. The municipality is faced with population decrease and devitalization[1] that was exacerbated by the closure of a saw mill, its only industry, in 2006.

On September 26 2012, RNC and the municipality of Launay entered a provisional collaboration and partnership agreement. The main objective of this agreement is to formalize the collaboration between RNC and the municipality of Launay to the benefit of the community and the advancement of the Dumont project.

20.3.1.3 Trécesson (Villemontel)

Trécesson township, with 1,138 citizens and 562 private dwellings, contains two villages: Villemontel, located about 3 km from the project's limits; and an area called La Ferme, which is more distant. There are many agricultural, forestry, recreational and cultural activities within this municipality. This municipality, as a whole, enjoys a demographic increase.

20.3.1.4 Unorganized Territory of Lac Chicobi (Guyenne)

The town of Guyenne, with 203 inhabitants, is located 10 km north of the project site in one of the two unorganized territories managed by the regional municipality. Most economic activity is related to agriculture and forestry. A lake of significance, Lac Chicobi, is located in this area and hosts many cottages and a summer camps. Lac Chicobi is in the Arctic drainage basin.

20.3.1.5 Berry

The project touches the southwest corner of the municipality of Berry. This municipality of 625 citizens and 230 dwellings is composed of two villages, Saint-Gérard-de-Berry and Saint-Nazaire, and cottage sectors around lakes, including Lac Berry and Lac Du Centre. The main activities are agriculture and forestry. A slight residential growth was noticed last decade in rural parts of the municipality and around lakes.

20.3.1.6 Taschereau

The municipality of Taschereau, 981 inhabitants, adjoins Launay to the west. The town, located about 12 km away from the project site, was built around a sawmill 50 years ago, which closed

[1] Government Action Plan for devitalized municipalities, Ministry of Municipal Affairs, Regions and Land Occupancy

permanently in 2011. The economy is based on agricultural and forest activities and on a new tourist and recreational project. Taschereau hosts many lodgings and restaurants, benefits from its location along the lake, and is at the northern limit of Aiguebelle Provincial Park.

20.3.1.7 Pikogan (Abitibiwinni First Nation)

The First Nation reserve of Pikogan is located along the Harricana River and is enclosed within the Amos municipal boundaries. There are more than 150 dwellings on the reserve. The reserve exists since 1956 and was expanded in 2008 to meet residential, economic and community needs. There are 934 persons registered as Abitibiwinni band members, of which 538 live in Pikogan (Aboriginal Affairs and Northern Development Canada, Indian registry – 2010). Part of the population is Algonquin and part is Cree. The Abitibiwinni band council, principal employer in the community, offers many services including education, social activities and economic development.

On April 5 2013, RNC and the Abitibiwinni First Nation (AFN) signed a memorandum of understanding (MOU). The MOU will serve as a framework to govern the relationship between RNC and AFN in accordance with their intention to further build on a relationship characterized by cooperation and mutual respect, in connection with the development of the Dumont project.

20.3.2 Land Uses & Tenure

A map showing land tenure information for the Dumont project area is given in Figure 20.2.

20.3.2.1 Crown Lands

The Dumont Nickel project is largely located on public land. The principal activities performed on this land relate to forestry (lumbering and forest management) and are managed by the Ministry of Natural Resources since 2013 through supply contracts. Part of this territory is subject to a forest management convention with the regional county municipality of Abitibi.

According to the MNRF there are five leases for hunting camps and two registered traplines within the Dumont property boundaries. RNC is under discussions with registered hunting camp owners for purchase option agreements.

20.3.2.2 Private Lands

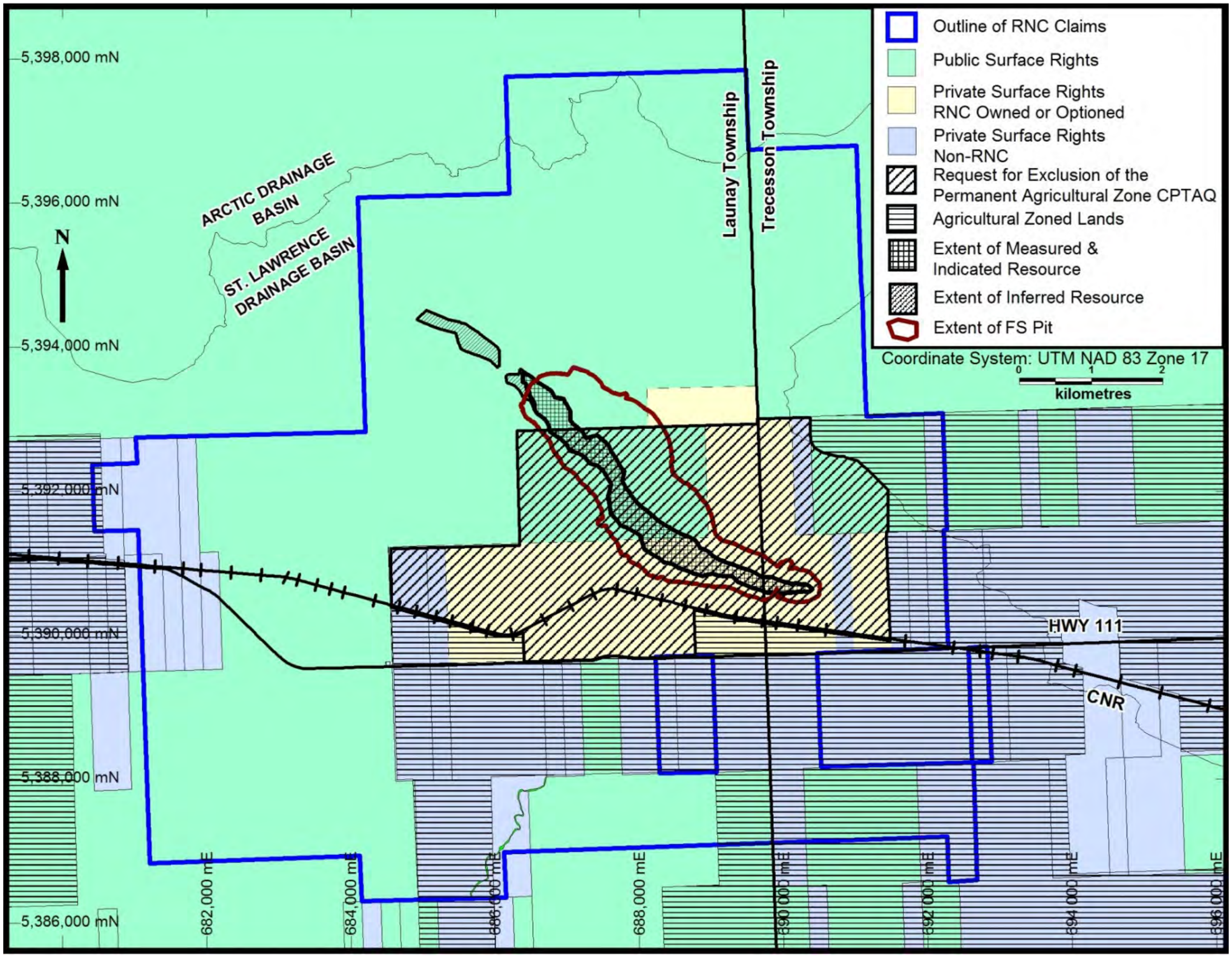
Part of the land proposed for project surface infrastructure development is privately held (Figure 20.2). These lands are assigned to agroforestry uses in the regional municipality land development plan. RNC has either purchased or concluded purchase options for the most strategic properties (see Section 5.5).

20.3.2.3 Agricultural Area

A portion of the private and public lands on the southern portion of the project is registered in the provincial agricultural zone (See Figure 20.2). Uses other than for agricultural purposes are subject to an authorization from the Quebec Agricultural Land Protection Commission (CPTAQ) under the Act respecting the preservation of agricultural land and agricultural activities (see Section 5.5).

Ausenco

Figure 20.2: Dumont Property Surface Considerations



Source: RNC.

20.3.3 Archaeology

A study of archaeological potential within the study area was conducted as part of the 2008 baseline study. No area of high archaeological potential was identified near the study area. Only a few areas of moderate to low potential have been noted on the banks of the Villemontel River and its tributaries. Since projected impacts are significant and permanent, a brief archaeological survey in the areas of moderate and low potential was recommended in case they are disturbed by the mining project. This survey will be performed in the summer of 2013.

20.4 Stakeholders Information & Consultation Process

Mindful of the interest shown by host communities following the announcement of the Dumont Nickel project, RNC voluntarily initiated a public information and consultation process during the exploration phase. The process aims to ensure effective communication and dissemination of information about the project, and to document the concerns, comments and suggestions of the host communities to refine the pre-feasibility study where possible and help define the content of the environmental impact study.

This approach comprises two main stages: (1) an information and consultation process associated with the pre-feasibility study; and (2) a consultation process associated with the ESIA.

To ensure a rigorous approach and to facilitate dialogue with the company, RNC retained the services of a social harmonization firm, Transfert Environnement. Acting as a third party during the consultation activities, its role was to support RNC in the coordination of the consultation activities and to produce the minutes and reports documenting the discussions and how RNC integrated them into the design of the Dumont project.

All information and consultation activities were documented and concerns expressed by the stakeholders were compiled. A report on the information and consultation process conducted during the pre-feasibility study was produced by Transfert Environnement in 2011. A second report on the consultation process associated with the ESIA will also be produced by Transfert Environnement and will be submitted to the relevant authorities, as well as being filed as a public document on the company's website.

The following types of communication were used during the consultation process:

- information sessions;
- open house events and site visits;
- feedback activities;
- establishment of advisory committees:
  - expanded advisory committee;
  - municipalities/company round-table; and
- information and consultation processes for the Abitibiwinni First Nation in Pikogan.

Tables 20-3 and 20-4 respectively present the main concerns and the location selection criteria discussed during the information and consultation activities.

20.4.1 Future Consultation Activities

The next steps in the consultation process for the Dumont project are part of the provincial and federal environmental impact assessment process. RNC intends to continue stakeholder consultation during the development and operating stages of the project to minimize and/or mitigate the impact of the project and foster acceptance. In the short term, consultation activities will consist of site visits and open house events.

Table 20-3: Main Issues of Concern raised during the Information & Consultation Processes

Category	Issues of concern
Information and consultation processes	Operation, composition, resources and role of the committees Access to information on the project Purpose of the consultation process
Methods and means of impact analysis	Credibility of the methods used to analyze the environmental and social impacts (e.g., questions regarding the methods selected to assess the project's social impacts) Accuracy of the data used (e.g., margin of error) Ongoing impact analysis Accounting for related projects
Economic development	Impacts on the local and regional economy Maximization of local and regional benefits Residential and industrial development Retention of newcomers and population growth
Water	Protection of groundwater (eskers, wells, etc.) Potential Contamination of surface water Chemical composition and management of effluent from the impoundments (waste rock piles and tailings storage facilities) Mitigation and compensation measures for impacts on water
Soil and location of components	Distance of components from the highway and residences Area of affected land
Fauna, flora and wetlands	Impacts on large fauna Compensation for destruction of wetlands
Visual impacts	Effect on the landscape Mitigation measures for visual impacts
Climate and air quality	Dust emission Dust control and mitigation measures
Human environment	Use of the railway Recreational tourism and agroforestry activities Purchase of nearby residences and negotiating process Real estate development Increase in the value of housing and its impact on the ability of residents to pay their taxes Benefits for the community in terms of infrastructure and community investment Social fabric and quality of life
Health and safety	Transport of chemicals Health risk to workers and residents related to the presence of chrysotile in dust Emergency response plan Site security
Nuisances	Noise Nuisances during the exploration and development phases Dust emissions Road congestion Heavy vehicle traffic
Restoration and post-closure	Plan for site restoration and future use Financial guarantees for site restoration Economic diversification fund
Project (various)	Possibility of gradually filling the pit Exploratory drilling and boreholes Profitability of the project Consequences of a possible sale of the project

Table 20-4: Location Selection Criteria Raised during the Consultations

Issues	Location criteria
Noise, visual and dust nuisances	Components positioned north of Highway 111 so that trucks do not have to cross it
	Truck traffic areas concentrated far from Highway 111 and residences
	Highest pile (waste rock pile) far from Highway 111 and residences
	Lower piles (tailings storage facilities and overburden storage area) near Launay and Highway 111
	Temporary piles (low-grade ore pile) near downtown Launay and Highway 111
	Rapid revegetation (overburden storage area and tailings impoundment dikes) near downtown Launay and Highway 111
	Tailings storage facility far from Highway 111 and residences
Water	Components located within a single watershed (Villemontel River)
	One-kilometre buffer zones around the Launay and St-Mathieu-de-Berry eskers
Sensitive environments	Protection of the wetland habitat of the Slender-leaf Sundew (special-status species)
	Protection of the wetland east of Launay
	Protection of the woods near the Launay esker
	Protection of the known territory of the Rock Vole (special-status species)

20.5 Preliminary Environmental & Social Impact Assessment

20.5.1 Preliminary Environmental & Social Impacts Identification

This section summarizes the main social and environmental impacts anticipated to be associated with the development of the Dumont project as identified in the ESIA. Although this list is not exhaustive, it underlines topics that will require specific consideration. The general approach retained complies with federal and provincial requirements for carrying out environmental assessments. The process used to identify and assess the importance of the impacts on the environment mainly relies on detailed descriptions of the project and the environment, consultations with stakeholders, and lessons learned from the performance of similar projects.

The importance of each impact was determined by experts focussing mainly on the effect of each impact on a component of the environment and integrates the criteria of intensity, extent, duration and probability of occurrence. The importance of an impact also integrates the effect of the proposed mitigation measures. The assessment is performed only once and describes the residual impact once mitigation measures are applied.

On the whole, the majority of the impacts are qualified as being of little or very little importance. It is worth noting the existence of several positive impacts, particularly for the components of the human environment. Medium residual importance levels are considered for the following impacts:

Physical Environment

- Greenhouse gas (GHG) emissions in the operating phase;
- Loss of arable land for other purposes during the operating phase;
- Changes to the water and sediment regimes during the construction/preproduction and operating phases; and
- Changes to the groundwater flow regime (lowering of the water table) during the operating phase.

Biological Environment

- Loss of forest habitats during the operating phase;
- Loss of bird habitats during the operating phase; and
- Loss of mammal habitats during the operating phase.

Human Environment

- Loss of jobs and reduced purchasing in the region during the closure phase;
- Possible deterioration of the economic security of households and reduction of community services during the closure phase;
- Encroachment on a portion of the land used by members of the Pikogan community for all phases of the project;
- Possible deterioration of the quality of life for part of the neighbouring population due to concerns about the potential effect of the project on the environment and health during the operating phase;
- Potential economic difficulties for low-income or fixed-income individuals and pressure on the existing services during the construction/preproduction phase; and
- Changes to the scenery as viewed by moving and stationary observers at some locations during the operating phase.

Only one impact is qualified as very important or important according to the *Canadian Environmental Assessment Act*, namely the risk of nitrogen dioxide formation at concentrations likely to affect health as this phenomenon has not yet been modelled and precise impacts could not be evaluated. This impact is considered to be a cause for concern due to the proximity of some residents of Launay and Villemontel and the scope of the blasting activities for ore extraction from the pit. Atmospheric dispersion modelling studies of airborne nitrogen dioxide concentrations during blasting will allow a more precise assessment of the health risks and whether specific preventive measures are required within the framework of the emergency response plan in order to ensure adequate protection of workers and the nearby population.

20.5.2 Mitigation Measures

Besides the commitment to implement standard mitigation measures normally formulated for similar industrial projects, RNC is considering the implementation of specific mitigation measures such as:

- remedial measure for private wells potentially affected by the water table drawdown associated to the pit dewatering;
- protection of the forested areas along Highway 111 to attenuate landscape modification issues;
- implementation of a 1 km buffer zone between the Launay esker and the closest mine infrastructure to avoid impacts on the aquifer;
- mitigation of the impact of the project development on the three identified “at risk” bird species by avoiding nest destruction related to wood harvesting during the nesting periods, from mid-May to August;
- construction of a berm between the tailings storage facility and the town of Launay to minimize the impact of a potential dyke failure;

- implementation of intensive dust control measures to reduce the project impact on air quality for surrounding populations; and
- implementation of a shuttle service to principal nearby towns to reduce employee traffic.

20.5.3 Compensation Program

20.5.3.1 Wetlands

According to the pre-feasibility study site layout, mining infrastructure encroaches on approximately 2,525 ha of wetlands. This will require that a compensation program be developed to protect, enhance or restore wetlands in the Abitibi-Témiscamingue region. This project will first be submitted to the Quebec Ministry of Environment, Sustainable Development, Wildlife, and Parks (MDDEFP) for acceptance and would be implemented during the construction phase.

20.5.3.2 Fish Habitat

According to the pre-feasibility study site layout, the development of the Dumont project is likely to negatively impact about 31 ha of fish habitat. However, concerned habitats are considered of low quality, do not include sensitive habitats (e.g., spawning grounds) and do not host any species of interest.

Under Section 27.1 of the *Fisheries Act* RNC will most probably be required to develop and implement a plan to compensate for damage, destruction and loss of fish habitat that will occur as a result of mine development. This compensation plan will have to satisfy both provincial and federal level of government. At this stage, the Department of Fisheries and Oceans (DFO) does not consider the sections of stream located in the footprint of the tailings storage facility to be fish habitat. RNC is presently in discussion with DFO and Environment Canada to determine if the watercourses located in the footprint of the waste rock dump, low-grade ore stockpiles, and overburden impoundments should be registered in Schedule 2 of the Metal Mining Effluent Regulations (MMER) under Article 36 (3) of the *Fisheries Act.*

20.6 Environmental Permitting & Applicable Regulations

20.6.1 Legal Context

Two levels of legislation control the environmental assessment and granting of operating licences for mining operations in Quebec. The following is a preliminary analysis of the environmental regulations in force that would be applicable to the Dumont nickel project. This analysis also includes other applicable law and regulations that could affect the permitting timeline, as well as proposed modifications of Quebec's mining law through Bill 43.

20.6.2 Provincial Permitting Process

In order to obtain the Certificate of Authorization allowing the construction and operation of the Dumont project, RNC is subject, under the *Provincial Environmental Quality Act* (Loi sur la qualité de l'environnement, L.R.Q., c. Q-2), to the assessment and review of environmental impacts procedure involving an environmental impact study eventually leading to public hearings. The provincial trigger to this process is the installation of a mill that processes 7 kt/d or more of ore. The actual mill design plans a 52.5 kt/d start-up, ramping up to 105 kt/d after expansion.

20.6.3 Federal Permitting Process

Given the processing capacity of 52.5 to 105 kt/d, the likely impact on fish habitat, and the storage and manufacture of explosive, the Dumont nickel project is subject to a comprehensive environmental study under the *Canadian Environmental Assessment Act* (CEAA, LRC, 1992, Ch. 37). In contrast with usual class screening environmental assessment, the comprehensive study process involves a greater implication of the federal government's experts from various departments such as Fisheries and Oceans (DFO) and Natural Resources (NRCan), as well as a formal public consultation process, including specific consultations of First Nations.

In addition to the comprehensive study, every mining project planning on using a fish habitat for storage of mining residue must be registered in Schedule 2 of the Metal Mining Effluent Regulations (MMER) under Article 36 (3) of the *Fisheries Act*. Consequently, RNC will have to evaluate various alternatives for mining residues storage and clearly demonstrate that the proposed scenario is the most appropriate under environmental, technical, economic and social considerations. In addition, under Section 27.1, RNC will have to develop and implement a plan to compensate for damage, destruction and loss of fish habitat that will occur as a result.

In 2010, the Governments of Canada and Quebec renewed the Canada-Quebec Agreement on Environmental Assessment Cooperation seeking to optimize the use of federal-provincial cooperation mechanisms within the framework of the *Canadian Environmental Assessment Act*. Although each government maintains authority in the areas under its jurisdiction and remains responsible for environmental assessment decisions required by its legislation, the agreement ensures a one-project-one-assessment delivery approach to projects requiring the application of federal and provincial environmental assessment requirements. The parties agree to work within the established timelines for environmental assessments set out in federal and provincial legislation. A one-window contact in each jurisdiction is accountable for ensuring that the legal requirements of the parties participating in the cooperative environmental assessment are met.

In addition to the collaboration agreement, the federal government established in 2007 the Major Project Management Office (MPMO) to support the Government of Canada's approach to ensure a more effective, accountable, transparent and timely review process. The MPMO's mandate is to provide overarching project coordination, management and accountability for major resource projects within the context of the existing federal regulatory review process; and to undertake research and identify options that drive further performance improvements to the federal regulatory system.

20.6.4 Other Applicable Law & Regulations

20.6.4.1 Quebec Mining Act

On May 29, 2013, the Quebec Minister of Natural Resources tabled the proposed new *Mining Act*, Bill 43. Bill 43 retains most of the current rules in the *Mining Act* pertaining to rights and ownership, but provides for several significant changes, including in regard to the rights of municipalities, environmental oversight, public interest considerations, economic benefit measures and First Nations consultations. The Bill is expected to be debated by the National Assembly during the fall 2013 session.

Previous to this, on February 13, 2013, the Minister of Natural Resources published several proposed amendments to the *Regulation respecting mineral substances other than petroleum, natural gas and brine.* All of these amendments pertain to the rules applicable to financial guarantee requirements for rehabilitation and restoration plans submitted and approved as a requirement for the granting of a Mining Lease. Under the modified regulation, the developer must provide a financial guarantee covering 100% of all anticipated costs related to site rehabilitation and restoration, including long-term water treatment and infrastructure dismantlement costs. The guarantee is payable in three instalments, 50% within 90 days of receipt of approval of the rehabilitation and restoration plan, 25% on the first anniversary of receipt of approval of the plan, and the final 25% on the second anniversary of approval of the plan. The economic model presented in this report accounts for this proposed regulatory change.

20.6.4.2 Act to Preserve Agricultural Land & Agricultural Activities

The purpose of the Act is to ensure the sustainability, on a territorial basis, of agricultural practices and to promote sustainable development of agricultural enterprises in established agricultural areas. In order to enforce this law, Quebec's government created the Quebec Agricultural Land Protection Commission (CPTAQ). Figure 20.2 shows the extent of the lands that are classified as an agricultural zone within the meaning of the Act respecting the preservation of agricultural land and agricultural activities. Mining activity on these lands would require rezoning or exclusion of these lands from the agricultural zone by the CPTAQ. This exclusion must be requested by the local municipality or by the regional county municipality (RCM). The application for exclusion must demonstrate that there are no suitable non-agricultural lands available for the stated purpose in the municipality. The majority of the agricultural lands located within the Dumont property are either non-arable or used for silvicultural purposes. The application for exclusion was submitted to the CPTAQ in February 2013 by the RCM supported by resolutions from the two municipalities directly concerned, Launay and Trécesson. Extensive consultation was performed by RNC with local farmers, the local and regional farmers union (UPA), as well as with the municipalities involved and the RCM in order to generate a strong consensus regarding the area targeted by the exclusion. Thus, RNC does not expect that exclusion for the purpose of developing the Dumont project would be unreasonably withheld.

20.6.4.3 Act Respecting the Acquisition of Farm Land by Non-Residents

Under the present Act, RNC is not considered a Quebec resident and must therefore request for a specific authorization to acquire parts of the private lots that are not included in the area targeted by the exclusion process described above. Both processes (exclusion and acquisition by non-residents) are being conducted simultaneously and will be analyzed together by the CPTAQ.

20.6.5 Permitting Timeline

20.6.5.1 Major Milestones

The proposed timeline for environmental permitting was developed under the assumptions that the two levels of governments, federal and provincial, will establish a good collaborative process under the Canada-Quebec Agreement on Environmental Assessment Cooperation.

The permitting process is initiated with the submission of a Project Notice to the Quebec Ministry of Environment and Sustainable Development (MDDEP). The project notice describes the scope of the project and provides a summary of potential environmental impact based on the PFS design. The Project Notice is assessed jointly at the federal and provincial levels and instructions on the scope and requirement for the environmental and social impact assessment (EISA) are forwarded to the developer.

Once the ESIA is completed and considered receivable by the authorities, the Quebec public hearing process is triggered by the Quebec public hearings bureau (BAPE). The BAPE then submits its recommendations to the MDDEP and eventually to other governmental authorities for decision concerning the issuance of a global Certificate of Authorization. Table 20-5 summarizes the main permitting milestones.

Table 20-5: Summary of Environmental Permitting Process Milestones

Major Milestones	Anticipated (Actual) Time frame
Project notice submission	December 2011 - Completed
Federal and provincial directive	February 2012 - Completed
Submission of the ESIA	November 2012 - Completed
Regulatory review of ESIA	In progress
Public hearing process kick-off	Q4 2013
BAPE recommendations to authorities	Q2 2014
C of A delivery	Q3 2014

Source: RNC.

20.6.5.2 Schedule II of the Metal Mining Effluent Regulations

Authorization of the placement of deleterious mining waste in a natural water body that is frequented by fish requires a regulatory amendment to list the water body on Schedule 2 of the Metal Mining Effluent Regulations (MMER). This process may take from 8 to 12 months. This process starts once the developer and the DFO come to an agreement with regards to a funded compensation plan for fish habitat loss. It is worth mentioning that the developer can start construction work upon receipt of the Certificate of Authorization prior to the MMER amendment, as long as the work carried on does not involve the use of fish habitat for storage of deleterious mining waste.

At this stage, RNC has been informed by DFO that the sections of creek potentially impacted by the two cells of the tailings storage facility are not considered to be fish habitat, and would therefore not trigger the MMER schedule 2 amendment process. RNC is presently seeking direction from DFO to better define the fish habitat status of sections of the unnamed creek potentially impacted by the waste rock, low-grade ore and overburden impoundments. RNC is also working to demonstrate to Environment Canada that these materials are not deleterious on the basis of extensive environmental geochemistry characterization and would therefore not trigger the amendment process.

20.7 Environmental Geochemistry Program

This section intends to give a broad overview of the environmental geochemistry work performed by RNC for the development of the Dumont Nickel project. It covers environmental geochemistry studies as well as studies designed to clearly define the potential of the mining waste to passively sequestrate carbon. The objectives of the environmental geochemical characterization program is to classify mine waste according to Québec *Directive 019 sur l'Industrie Minière* (Directive 019) for waste management planning and to identify elements of potential environmental interest in the framework of future mine site water quality, in order to assess possible water treatment requirements during mine operation.

20.7.1 Phase 1: Baseline Environmental Testing on Mineralized Rocks, Waste Rocks & Tailings

A preliminary environmental geochemistry study was completed in 2009 by GENIVAR LP (GENIVAR, 2010a). This study characterized mineralized rock, waste rock and metallurgical processing wastes expected to be equivalent to tailings at the time of testing. A total of 30 samples were subjected to acid base accounting and metal leaching tests (TCLP-1311, SPLP-

1312 and CTEU9, for each sample), one MWMP leaching test, and five samples subjected to kinetic humidity cell tests. The waste rock samples tested showed no potential for acid generation and were classified as non hazardous, but showed leachate concentrations of pH, aluminum, arsenic, fluoride, iron, mercury and zinc that exceed Quebec Effluent Criteria (Directive 019) and/or the criteria for groundwater quality. The MWMP test on the composite mineralized rocks showed no concentration in leachates above the criteria. The humidity cell test showed slight sulphide oxidation and neutralization by carbonates. Based on the kinetic test results, no acid generation was observed and the samples did not leach metals to a concentration elevated above the criteria used in the baseline study. The alkaline pH of the leach solutions did, however, exceed the upper range of the groundwater criteria. It was recommended that further testing be completed to meet permitting requirements.

20.7.2 Phase 2: Static Testing for Waste Rock, Low-grade Ore, Tailings & Overburden

A second, broader environmental geochemistry study was initiated in 2010. Static testing was completed in 2011 and kinetic weathering tests were completed in 2013 (Golder, 2013). The Golder 2013 report presents the results of the Phase 2 work completed on waste rock, low-grade ore, tailings, tailings process water samples and overburden. The report presents the chemical composition of the mine waste, its potential to generate acid rock drainage (ARD) and to leach metals to the surrounding environment upon exposure to ambient conditions. The static and kinetic test methods utilized on mine waste solids are consistent with those recommended under Quebec Directive 019. They include acid-base accounting (ABA), chemical composition (major and trace element) and static leaching tests (TCLP, SPLP, CTEU9) on all solid materials as well as standard humidity cell kinetic leaching tests on tailings and waste rock.

20.7.2.1 Waste Rock Geochemical Characteristics

All waste rock samples tested were classified as non-acid generating (Non PAG), but leachable per Directive 019. All but one sample of waste rock reported less than 0.3% sulphur content and high buffering capacity demonstrated by neutralization potential ratios (NPR) greater than 10 (compared to a minimum of 3 recommended in Directive 019). One sample of volcanic rock had a sulphur content (S(T)) of 0.32% but ample buffering capacity and thus, classified as Non PAG. Table 20-6 summarizes the results of the various static tests performed on waste rock and low-grade ore.

Table 20-6: Summary of Chemical Characteristics & Classification of Major Waste Rock Types & Low-grade Ore based on Static Testing Results (Golder, 2013)

Rock Type	Bulk Potential by Rock Type: No. of Samples	Avg S(T) (%)	Bulk NPR	Bulk ARD Designation	TCLP Leachate Exceedances to Groundwater Quality Criteria [1]	Waste Rock Lithology Classification (Directive 019)
Volcanic	27	0.10	29	Non-PAG	Cu (4), Mn (9), Ni (5)	Leachable
Volcanic (outcrop)	6	0.04	26	Non-PAG	Cu (2), Mn (1), Ni (1)	Leachable
Peridotite	32	0.05	72	Non-PAG	Cr (19), Mn (4), Ni (32)	Leachable
Dunite	28	0.04	119	Non-PAG	Cr (4), Cu (1), Ni (28)	Leachable
Dunite (Low-grade Ore)	11	0.04	165	Non-PAG	Mn (1), Ni (11)	Leachable
Gabbro	42	0.07	15	Non-PAG	Cr (4), Cu (17), Ni (3) Pb (1)	Leachable

1. For samples where the chemical composition also exceeds Quebec Soil Criteria A for the stated parameter.

Samples were classified as leachable based on the double criteria of TCLP static leaching test results and chemical composition. For many samples, chromium, copper, manganese and nickel occur in both in the solid phase at concentrations that exceed Quebec Soil Criteria A and in TCLP leachate at concentrations that exceed Quebec groundwater quality criteria. Chromium, copper and nickel also exceed groundwater criteria in the more representative acid-rain simulated SPLP test and in the CTEU9 water-leach test although less frequently and at lower levels (occur on fewer samples and generally at lower concentrations) than those measured in the more aggressive TCLP test. Nonetheless, the short-term leach test methods recommended under Directive 019 are limited in their ability to simulate site conditions and therefore to represent anticipated mine waste contact water quality.

Kinetic test methods provide a more representative assessment of probable future mine waste contact water quality over the long term. Standard humidity cell kinetic weathering tests were completed on 13 samples of waste rock from the different lithologies. Results are presented in Golder (2013). Apart from some exceedances to water quality criteria in the initial cycles of testing, the effluent and groundwater water quality criteria were met in the long-term, except for the alkaline pH that remained above the provincial effluent criteria range in all samples of peridotite and some samples of dunite. These results suggest that although waste rock carries a ‘leachable’ classification according to Quebec Directive 019 criteria, water quality contacting waste rock is likely to have low concentrations of the chemicals of environmental interest highlighted by static leaching tests.

20.7.2.2 Tailings & Process Water Geochemical Characteristics

The Golder 2013 study presents the static test results of the 15 tailing samples representing various types of processed ore (from different areas within the deposit) which will be deposited in the same tailings storage facility during mine operation. All tailings samples are classified as Non-PAG but leachable according to Directive 019. Ten of 15 samples released nickel at concentrations that exceeded Quebec groundwater quality criteria (Table 20-7). Water leaching tests (SPLP and CTEU9) on the tailings solids showed few additional parameter exceedances to groundwater criteria (mostly silver and copper).

Table 20-7: Summary of Environmental Characteristics for Tailings Samples (Golder, 2013)

Tailings Sample	ARD Potential			TCLP Based Leachability Classification [1]	Bulk Waste Classification (Directive 019)
	S(T) (%)	Bulk NPR	Bulk ARD Designation		
15 samples from various ore types	0.07	109	Non-PAG	Ni (10)	Leachable

1. For samples where the chemical composition also exceeds Quebec Soil Criteria A for the stated parameter

Standard humidity cell kinetic weathering tests were completed on 7 samples of tailings. Results are presented in Golder (2013). Most chemical concentrations met the effluent and groundwater water quality criteria during the testing except for the alkaline leachate pH that remained above the provincial effluent criteria range in all tailings samples. Some constituents including arsenic, chloride, copper and nitrate showed exceedances in the initial cycles of testing but decreased to below groundwater or effluent criteria subsequently. Nickel remained below the effluent and groundwater criteria in all samples for the duration of the kinetic tests.

Fifteen (15) samples of process water were analysed. Some samples showed exceedances to groundwater quality criteria for chloride, total chromium and total copper and fewer samples for dissolved chromium but no exceedances for dissolved copper. Total suspended solids concentrations were above Quebec effluent quality criteria in 5 samples but all other parameters including pH were below the effluent criteria. Six (6) of the 15 samples of process water that were subjected to toxicity testing on rainbow trout and daphnia magna showed no toxicity to both organisms.

20.7.2.3 Overburden

Samples of the different overburden types were subjected to the full suite of static tests including acid generation potential, chemical composition and the three leaching tests (TCLP, SLPL and CTEU9) per the Quebec recommended analytical methods. Results are summarized in Table 20-8.

Table 20-8: Summary of Chemical Characteristics & Classification of Overburden based on Static Testing Results (Golder, 2013)

Overburden Material	Number of Samples	Bulk Potential by Overburden Type			TCLP Leachate Exceedances to Groundwater Quality Criteria [1]	Bulk Overburden Classification (Directive 019)
		Avg S(T) (%)	Bulk NPR	Bulk ARD Designation		
Base Till	12	0.03	41	Non-PAG	Cr (1), Cu (1), Ni (5)	Leachable
Upper Till	2	0.06	50	Non-PAG	Cr (1), Ni (1)	Leachable
Silt Sand and Gravel	11	0.04	35	Non-PAG	Ni (1)	Low Risk
Clay	8	0.03	91	Non-PAG	none	

1. For samples where the chemical composition also exceeds Quebec Soil Criteria A for the stated parameter

All overburden materials are Non-PAG and some samples mostly of till leach metals at concentrations that exceed Quebec groundwater quality criteria and soil criteria. The sand-silt-gravel and the clay are considered low risk given the small number of exceedances, the low level of exceedances in the one sample and that the average TCLP concentrations for all parameters meet the comparative criteria.

20.7.2.4 Waste Rock Classification for Construction Use

Re-use of waste rock based on static leaching tests classifies Dumont waste rock as Category III, re-usable outside the mine footprint only if encapsulated without direct contact with natural soils.

Notwithstanding this, kinetic tests suggest that contact water is likely to contain low concentrations of metals. Thus, the use of waste rock as fill or for infrastructure construction within the mine property may require measures to protect soil or groundwater during mine operation or at closure. As such, their use on the mine site should be discussed with Quebec authorities.

20.7.3 Large Scale Kinetic Weathering Tests

20.7.3.1 Leaching Columns

Large scale kinetic weathering tests (leaching columns) were initiated in March 2012 and are largely complete on each of the major lithologies and low-grade ore (6 cells) and on tailings (1 cell) to evaluate test scale-up effects on leachate water quality. These tests were conducted at the Unité de Recherche et Services en Technologie Minérale (URSTM) of the Université du

Québec en Abitibi Temiscaming. The results of this study are included in appendix to the Golder (2013) report.

Results corroborate those obtained from the standard size humidity test cells where exceedances to the effluent criteria are noted for pH from the waste peridotite, dunite and the low-grade ore dunite. Few isolated exceedances to Quebec groundwater quality criteria are noted mostly in the early leaching cycles. Late cycles show no exceedances to these criteria.

20.7.3.2 Field Scale Experimental Cells

Two larger field scale leaching tests (in-situ experimental cells) were built at the project site in 2011 and continue to be run by RNC (Figure 20.3). One of the cells contains a mixture of waste and low-grade dunite and the other contains tailings. These tests were meant to evaluate the carbonation potential and the geochemical behaviour of the waste rock and tailings under conditions that are similar to those expected in the actual waste rock piles and in the tailings management facility, particularly for the lithologies containing sulphides and/or alloy.

Figure 20.3: In-Situ Cells – Tailings cell in foreground, waste rock (serpentinized dunite) in background diameter of tailings cells is 5 m

Source: RNC.

The tailings cell is instrumented with sensors measuring volumetric water content, temperature and water potential. This provides information on the geotechnical behaviour of the tailings exposed to natural conditions. A meteorological station was installed onsite to monitor atmospheric conditions (precipitation, atmospheric pressure, wind speed and direction, solar radiation).

Leachate water quality from both experimental cells meets Quebec effluent criteria. Leachates also generally meet the groundwater criteria with few isolated exceptions for silver, arsenic and manganese (few cycles and marginal exceedances).

Results obtained to date corroborate those obtained from the smaller scale standard humidity test cells and larger leaching columns; they suggest that leachate water quality contacting tailings and waste rock is likely to be substantially better than those on which are based the leachable classification for these wastes.

20.7.4 Carbon Sequestration

Sequestration of CO_2 by reaction with magnesium-rich natural minerals, such as the serpentine contained in the Dumont deposit, and its long-term storage in the form of magnesium carbonates has been identified as one of the only permanent carbon sequestration processes. This is considered to offer a significant potential for the reduction of the environmental footprint of the project through reduction of net greenhouse gas emissions (GHG). This spontaneous reaction is known as spontaneous mineral carbonation. Spontaneous mineral carbonation is a process that occurs naturally at ambient conditions whereby magnesium silicate serpentine minerals (including chrysotile) are transformed into magnesium carbonate minerals, such as magnesite, in the presence of water and carbon dioxide.

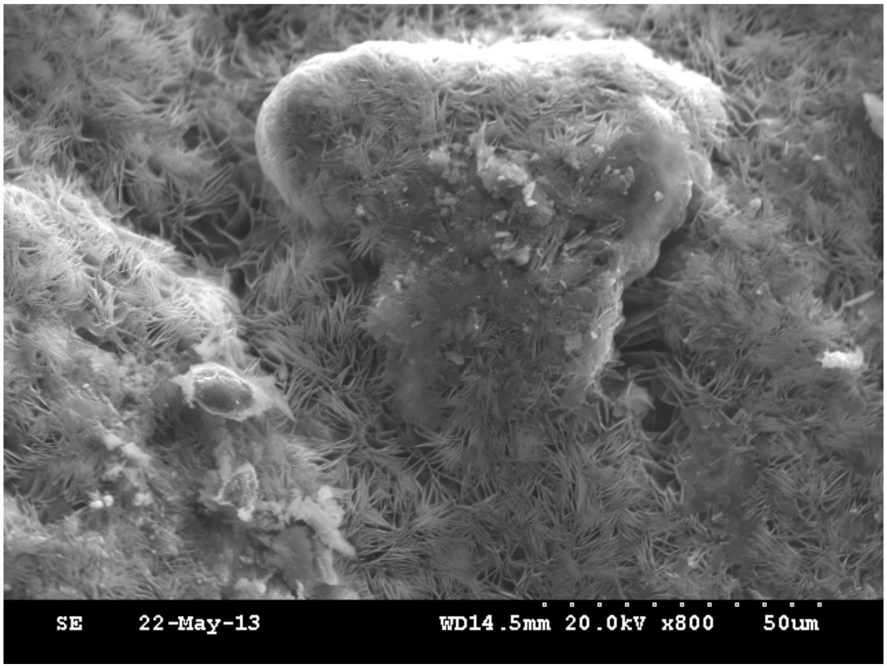
In 2010, a team from Laval University conducted a study aiming to determine the potential for carbon sequestration on various Dumont project mine wastes including: air-classified fibres, desliming tails (slimes) and final flotation tailings (Pronost et al., 2010). The study clearly demonstrated that the materials are able to sequestrate carbon by binding atmospheric carbon dioxide (CO_2) in the form of various secondary carbonate minerals. Samples carbonated under ambient air sequestered approximately 0.8% to 1.0% of their mass of CO_2. Their CO_2 concentrations increased from an initial value of 0.3% to 0.9% CO_2 to 1.5% to1.9% CO_2 after carbonation. Samples carbonated in eudiometers which reached their total carbonation potential have a final CO_2 concentration varying from 5.2% to 9.5%.

The experimental tailings and waste rock cells constructed at the Dumont site were instrumented to determine CO_2 sequestration under natural conditions. This ongoing study, involving researchers from Laval University and Université du Québec en Abitibi-Témiscamingue (UQAT), aims to better understand carbonation mechanisms to allow RNC to quantify and optimize the carbon sequestration reactions in the Dumont waste rock and tailings and thus potentially offset the GHG emissions from the project.

The large leaching columns completed at UQAT were dismantled after 1 year of operation. The particles in the ultramafic rock columns were found to have agglomerated together into clumps. The cemented clumps were mounted as whole grains/clumps and imaged via SEM (scanning electron microscopy). The images showed extensive growths of various carbonate minerals (identified by EDS, Energy-Dispersive X-Ray Spectroscopy) across ultramafic (peridotite) grains which were cemented together by carbonate matrix (Figure 20.4). Fibrous serpentine was also found to show evidence of carbonate growth and cementation (Figure 20.5). SEM characterization for the remaining lab weathered samples is ongoing and will include further SEM imaging and XRD,

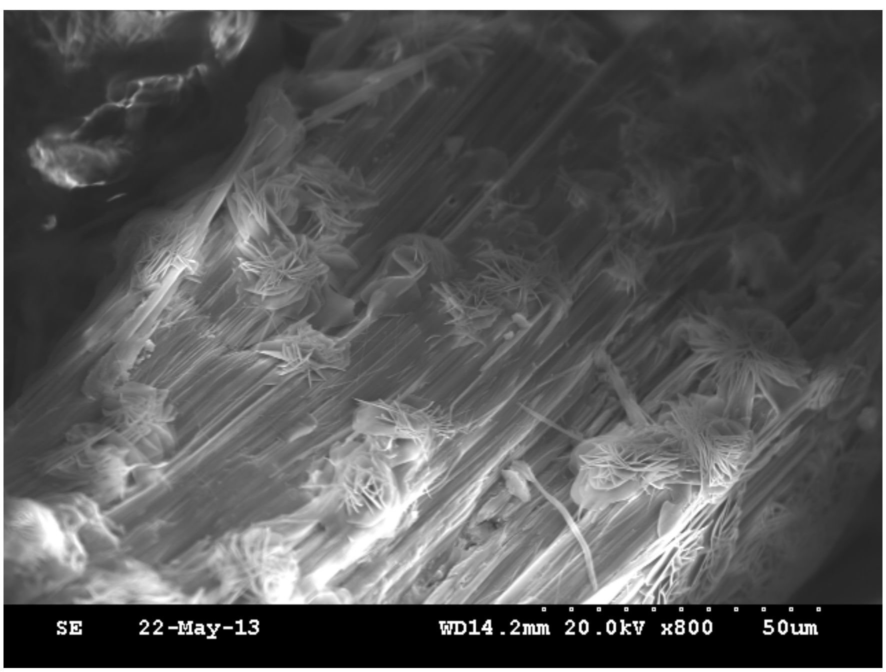
In May 2013, an onsite experiments was set up to characterize short-term weathering of ultramafic waste peridotite, dunite and tailings. Previous studies have shown the production of secondary carbonate through prolonged weathering (~1 year). The purpose of the onsite experiment is to assess the rate at which the secondary carbonation reaction takes place. Samples will be taken after each week of weathering and analysed with SEM equipped with EDS. The weekly samples will also be assessed for chrysotile content though point counting to characterize chrysotile breakdown during secondary carbonation. This program is ongoing.

Figure 20.4: SEM image of lab weathered peridotite grains. A smaller peridotite grain is "cemented" to a larger grain through a network of platey carbonate minerals



Source: URSTM

Figure 20.5: SEM image of lab weathered fibrous serpentine from a peridotite: Fibrous Serpentine show secondary carbonate alteration



Source: URSTM

20.7.5 Subsequent Studies

Following on the results of the two phases of study and larger scale weathering studies, additional tests are being undertaken on tailings and leaching column waste rock materials as follows:

- UQAT Laboratory mineralogical analysis (SEM and EDS) on weathered rock extracted from the leaching columns. SEM and EDS will focus on characterizing secondary carbonate minerals from weathered ultramafic rock (as described in Section 20.7.4).
- Weekly SEM, EDS and point counting analysis for onsite experiment to assess the rate of carbonation (as described in Section 20.7.4).
- Static chemical tests on material from the waste rock columns (waste rock, low-grade ore and tailings). This aims to document chemical and mineralogical changes that might have occurred during testing, to assist in the interpretation of the long-term weathering behaviour of waste rock and tailings.

A submerged tailings column leach test is also underway at SGS Laboratories to examine the effect of the presence of tailings on the chemistry of the overlying water column. This test aims to replicate the geochemical processes that would occur between tailings that will be placed in the bottom of the open pit during the last years of mine operation and the open pit flood waters post closure. Results on water quality from the leaching column will be used to validate results of a predictive water quality model for the Site.

20.8 Health & Safety

20.8.1 General

Health and safety issues concerning communities and workers that are specific to the development and operation of the Dumont project are noted below.

- restricting access to the large industrial site through the use of efficient measures such as fencing;
- minimizing road traffic hazards related to trucking through optimal use of the railway;
- reducing psychosocial effect of the project on surrounding communities by implementing efficient communication channels such as a stakeholders monitoring committee and a complaint management system;
- limiting emissions of potential air contaminants, including chrysotile, through the implementation of efficient dust control measures;
- avoiding water contamination by minimizing the release of a mining effluent into the environment through an efficient site water management and maximizing recycling of process water, and by establishing an effective water treatment plant for water leaving the project site; and
- managing risks associated with the presence of chrysotile in the ore and waste rocks.

20.8.2 Chrysotile Management

The most specific health and safety hazard associated with the development of the Dumont project is the hazard associated with airborne chrysotile fibres. Chrysotile, a fibrous form of serpentine, is one of six minerals commonly referred to under the commercial identification of asbestos. Chrysotile is found in the Dumont ore body in the serpentinized dunites and peridotites in proportions ranging from 0% to 10%. The 95% confidence interval for the average bulk chrysotile content for these rock types lies between 1.6% and 1.9% (see Sections 9.5 and 11.1.7). Exposure to airborne chrysotile fibres must be minimized due to the carcinogenic potential associated with inhalation of airborne chrysotile fibres. Quebec occupational and health regulations set the exposure standard to chrysotile in air for workers at one fibre per cubic centimetre (1f/cm^3). There is no chrysotile in the gabbro and basalt rock types or in the clay and granular overburden.

Regulated standards for airborne chrysotile have been maintained at recently producing chrysotile mine and mill operations such as those in Thetford Mines, Quebec through effective engineered controls focussing on dust control and capture at source in dry process areas, air filtration in mobile equipment cabs, and humidification in open pit operations. Regulated standards are maintained by RNC in its exploration facilities through dust control and capture at source and wet core sawing. RNC has conducted an air quality testing program at its facilities since 2007. In 2013, sampling results measured chrysotile concentrations ranging from 0.0005 f/cm^3 to 0.11 f/cm^3 for 18 tests performed on workers and in fixed locations inside the facilities. The maximal value recorded since the beginning of this program was 0.48 f/cm^3 in 2007 and has been steadily reduced since then. Even though the measurements are significantly below Quebec standards, RNC requires employees working in sensitive occupations to wear chrysotile-rated respirators.

RNC has also implemented a chrysotile monitoring program at the in situ tailings and waste rock characterization cells (see Section 20.7.3) in collaboration with the local branch of Quebec's Health and safety commission. The objective of this program is to quantify the potential for airborne wind dispersal of chrysotile fibres into the surrounding environment and communities. The spontaneous mineral carbonation process described in Section 20.7.4 whereby chrysotile in tailings and waste rock is rapidly transformed spontaneously to magnesium carbonates is likely to play an important role in chrysotile dust control.

It is important to note that there is no regulation in Quebec or Canada regarding airborne chrysotile concentrations in the natural environment.

21 CAPITAL & OPERATING COSTS

21.1 Capital Cost Estimate Input

The capital cost of the project, including the 52.5 kt/d production rate, expansion to 105 kt/d, and sustaining expenditures over the 33 year life, has been estimated based on the scope of work defined in the sections below. The parties below have contributed to the preparation of the capital cost estimate in specific areas, as listed:

Ausenco

- Crushing;
- Process;
- Loadout;
- Tailings storage facility (excluding dam earthworks);
- Mine office, truckshop and washbay;
- Utilities;
- On-site infrastructure;
- Off-site infrastructure;
- Indirect costs; and
- Contingency.

SRK

- Waste dumps;
- Channel design; and
- Sumps.

RNC/Snowden

- Site preparation (clearing and grubbing);
- OP mine development (by both Owner and Contractor);
- OP mobile equipment;
- OP Ancillary equipment;
- Tailings dam earthworks; and
- Owner's costs.

All amounts expressed are in Canadian dollars unless otherwise indicated.

21.2 Capital Cost Estimate Summary

The estimate is classified as an Ausenco Class 3 Feasibility Study Estimate with ±15% accuracy.

Table 21-1 provides a summary of the capital cost estimate, including initial capital, expansion capital, and sustaining capital. The costs are expressed in real, Q2 2013 Canadian dollars.

Items that would be denominated in foreign currency take account of the forecast exchange rate at the time of purchase. Indirect costs include first fills of consumable items for the initial and expansion estimates, and the release of these under the sustaining estimate.

Table 21-1: Summary of Capital Costs ($M)

Description	Initial Capital ($M)	Expansion Capital ($M)	Sustaining Capital ($M)	LOM Total Capital ($M)
Mine	320	216	419	955
Process Plant	428	424	254	1,106
Tailings	34	61	172	267
Utilities	122	99	0	221
Infrastructure	87	27	0	114
Indirect Costs	125	80	(22)	183
Owners Costs	47	9	0	56
Contingency	105	81	0	186
Total	**1,268**	**997**	**823**	**3,088**

Notes: 1. Negative indirect costs for sustaining capital reflect the release of first fills.

Tables 21-2 to 21-4 show details of the initial, expansion and sustaining capital costs by WBS area and include the composition by currency. Bids all assumed a USD/CAD exchange rate of parity, which has been adjusted in the capital estimate to reflect the forecast exchange rate at the time expenditures would be made.

Table 21-2: Initial Capital Costs by Area

	Composition		C$M Total Cost	
WBS Area	C$ M	US $M	At Parity	At Forecast F/X
Area 1 - Mining	195	119	314	320
Area 2 - Crushing	50	5	55	55
Area 3 - Process	263	104	367	372
Area 4 - Concentrate Load Out	0.3	0.0	0.3	0.3
Area 5 - Tailings	32	1	34	34
Area 6 - Utilities	121	2	122	123
Area 7 - Onsite Infrastructure	80	0	80	80
Area 8 - Off-site Infrastructure	7	0	7	7
Sub-Total Directs	**748**	**231**	**979**	**991**
Area 9 - Indirect Costs	125	0	125	125
Area 10 - Owner's Costs	47	0	47	47
Sub-Total Indirects	**172**	**0**	**172**	**172**
Total Directs + Indirects	**920**	**231**	**1,151**	**1,163**
Area 11 - Escalation	Excluded			
Area 11 - Contingency	105	0	105	105
Total Project Costs	**1,025**	**231**	**1,256**	**1,268**

Table 21-3: Expansion Capital Costs by Area

	Composition		C $M Total Cost	
WBS Area	**C$ M**	**US $M**	**At Parity**	**At Forecast F/X**
Area 1 - Mining	32	166	198	216
Area 2 - Crushing	50	5	55	55
Area 3 - Process	253	104	357	369
Area 4 - Concentrate Load Out	0	0	0	0
Area 5 - Tailings	60	1	61	61
Area 6 - Utilities	97	2	99	99
Area 7 - Onsite Infrastructure	22	0	22	22
Area 8 - Off-site Infrastructure	5	0	5	5
Sub-Total Directs	**519**	**278**	**797**	**827**
Area 9 - Indirect Costs	80	0	80	80
Area 10 - Owner's Costs	9	0	9	9
Sub-Total Indirects	**89**	**0**	**89**	**89**
Total Directs + Indirects	**608**	**278**	**886**	**916**
Area 11 - Escalation	Excluded			
Area 11 - Contingency	81	0	81	81
Total Project Costs	**689**	**278**	**967**	**997**

Table 21-4: Sustaining Capital Costs by Area

	Composition		C $M Total Cost	
WBS Area	**C$ M**	**US $M**	**At Parity**	**At Forecast F/X**
Area 1 - Mining	63	320	384	419
Area 2 - Crushing	0	0	0	0
Area 3 - Process	254	0	253	254
Area 4 - Concentrate Load Out	0	0	0	0
Area 5 - Tailings	172	0	172	172
Area 6 - Utilities	0	0	0	0
Area 7 - Onsite Infrastructure	0	0	0	0
Area 8 - Off-site Infrastructure	0	0	0	0
Sub-Total Directs	**489**	**320**	**809**	**845**
Area 9 - Indirect Costs	(22)	0	(22)	(22)
Area 10 - Owner's Costs	0	0	0	0
Sub-Total In-Directs	**(22)**	**0**	**(22)**	**(22)**
Total Directs + Indirects	**467**	**320**	**787**	**823**
Area 11 - Escalation	Excluded			
Area 11 - Contingency	0	0	0	0
Total Project Costs	**467**	**320**	**787**	**823**

The estimate is based on an EPCM execution approach as outlined in Section 21.4.2.2.

The following parameters and qualifications are made:

- The estimate is based on Q2 2013 prices and costs.
- Financing related charges (e.g., fees, consultants, etc.) are excluded.
- There is no escalation added to the estimate, other than the contingency.

Data for these estimates have been obtained from numerous sources, including:

- feasibility level engineering design;
- mine schedules;
- topographical information obtained from site survey;
- geotechnical investigation;
- budgetary equipment quotes from multiple potential OEMs;
- budgetary unit costs from local contractors for civil, concrete, steel, electrical and mechanical works;
- data from recently completed similar studies and projects; and
- information provided by RNC, SRK, Snowden, and Norascon.

Major cost categories (permanent equipment, material purchase, installation, subcontracts, indirect costs and Owner's costs) were identified and analyzed. To each of these categories, a percentage of contingency was allocated based on the accuracy of the data, and an overall contingency amount was derived in this fashion.

21.3 Capital Estimate Scope

21.3.1 Mining

Mining costs have been estimated by RNC. Table 21-5 summarizes elements of the mining capital estimates for the initial, expansion and sustaining phases of expenditure. Note that costs presented in Table 21-5 are based on the forecast exchange rate at time of expenditure. Also note the time periods for the three stages of development:

- Initial Capital: to the end of Month 1 of mill production (22 months pre-strip + 1 month mill production = 23 months total), or September 2014 to July 2016;
- Expansion Capital: from Month 2 to the end of Month 54, at which time the expansion is commissioned (August 2016 to December 2020); and
- Sustaining Capital: from Month 55 (January 2021) to the end of project life (Month 398 or 33 years and two months after the start of production (August 2049).

Table 21-5: Summary of Mining Capital Costs ($ M)

WBS Area	Initial (52.5 kt/d mill)	Expansion (105 kt/d mill)	Sustaining (105 kt/d mill)	Total
01 - 100: Site Preparation	5	0.4	3	8
01 - 200: Contractor Stripping	50	0	0	50
01 - 300: Owner Stripping	101	0	0	101
01 - 400: Mining Equipment	133	199	385	717
01 - 500: Ancillary Equipment	18	5	7	30
01 - 600: Truck Shop	11	7	24	42
01 - 630: Magazines	0.2	0.2	0.0	0.4
01 - 640: Fuel Storage	1.0	0.7	0.0	2
01 - 700: First Fills	1	4	0	5
Total	**320**	**216**	**419**	**955**

Sources of the estimates presented in Table 21-5 are as follows:

Site Preparation – The estimate is based on the total area that would be cleared during each of the three stages of development and an estimated unit rate of $2,000/ha cleared that was provided by the pre-selected mining Contractor.

Contractor Stripping – The estimate is based on the quantity of mining that would be allocated to the Contractor and unit rates provided by the Contractor.

Owner Stripping – The estimate is based on the quantity of mining that would be performed by the Owner and a zero-based model of mining costs. This model has been reviewed and agreed to by Snowden.

Mining Equipment & Ancillary Fleet – The zero-based model includes a derivation of the mobile equipment that would be required to achieve the planned mining schedule. Unit costs for specific units of mining or ancillary equipment were based on budgetary estimates provided by dealers representing the major Original Equipment Manufacturers (OEMs). This includes Caterpillar, Komatsu, Hitachi, Joy and Sandvik. Estimates included not only the cost of machines, but also the associated cost of transport to site and assembly.

Truck Shop – The truck shop will be expanded over time in line with the fleet of production haul trucks. The number of bays required has been estimated using the empirical formulae of 1 workshop bay per five trucks and 1 auxiliary bay per 12 trucks. Six bays will be constructed for the initial phase, while the truck shop will ultimately reach 20 bays. The cost of bays is based on the requirement for 230t class haul trucks.

Magazine – Explosives facilities (including the magazines for storing explosives as well as the plant for manufacture of explosives that will be installed during Year 2 of mill production or Q2 2017) will be rented from the explosives supplier. The only costs included in the capital estimate are thus costs associated with site preparation.

Fuel Storage – The size of the diesel ‘fuel farm’ has been based on diesel consumption as estimated by the zero-based model.

First Fills – First fills for the mine have been calculated based on a stores holding of 1 month for all consumable items with the exception of tires (4 months), diesel (5 days) and electricity (no holding). No advance purchase of mine maintenance items would be required as these would be held on a consignment basis..

21.3.2 Process Plant

The nickel recovery plant and associated facility estimates have been prepared on a commodity basis (i.e., divided into earthworks, concrete, structural, etc.) and reported by area (i.e., crushing, milling, etc.). The estimate is based on the purchase of new mechanical equipment, and quantities have been assessed from first principles.

The estimate is based on the majority of the work being carried out under fixed price or unit price contracts under a normal development schedule. No allowance is included for contracts on a cost plus or fast-track accelerated schedule basis. The erection of tankage, structural, mechanical, piping, electrical, instrumentation, and civil works will be performed by experienced contractors, using local labour.

21.3.3 Tailings Storage Facility

The estimate makes provision for constructing the starter dam at the initial cell of the TSF (Cell 1) to a height of 10 m (compared to the ultimate height of approximately 45 m). This height is sufficient to store approximately the first year of tailings production. The dam would be constructed by first excavating any clay that underlies the downstream portion of the dam wall and backfilling this excavation (the key trench) with waste rock, though sand and gravel overburden stripped from the pit may be used depending on the availability of waste rock. The dam wall would then be constructed using waste rock from the pit. The wall would include a central clay core (constructed using brown clay excavated from either the open pit or the key trench) and an adjacent filter layer (constructed using sand and gravel stripped from the pit).

The dam wall will be constructed on a 12-month basis and sequenced so that all rock used would be provided by ROM ex-pit mining activities with no stockpiling or re-handling required. The cost of loading, dumping and hauling of this material to a pit exit is excluded from the capital estimate as it is provided for in the mining cost. Only the cost associated with the incremental hauling distance to the TSF is included (over the life of mine, the 1-way haulage distance for material used in TSF construction is approximately 5.8 km, in addition to the average 4.7 km one-way haul for all ex-pit material).

The clay core and filter layer would be constructed on a six-month basis during warm months, when the clay and filter material can be loaded, dumped and spread without material freezing, and peak requirements will exceed instantaneous ROM output. This results in the requirement of some stockpiling and re-handling of these materials. Material that does not require stockpiling is priced in the same manner as ROM rock used for construction, with only the incremental haulage distance included. For material that is stockpiled, the additional cost of re-handling, loading and dumping is also included in the capital cost.

The capacity of the TSF would be increased progressively through continual lifting of the dam walls. Cell 1 would be raised at a rate of approximately 6 m annually, to an ultimate height of approximately 45 m. The storage capacity of Cell 1 at this height will be approximately 142 Mt of tailings, sufficient for the initial six years of mill production. Cell 2 will be built to a starter height of approximately 15 m, then raised at a rate of approximately 4 m annually to its ultimate height of approximately 65 m. The additional 538 Mt capacity provided by Cell 2 will be sufficient for all ore processed while the pit is active. When feed to the mill switches from the pit to ore stockpiles, tailings will be deposited in the pit. All costs associated with the raising of dam walls will be classified as capital costs.

21.3.4 On-Site Infrastructure

The following administration buildings will be built:

- main administration building with medical centre and training room;
- mine dry;
- security office;
- security gatehouse; and
- assay lab.

In addition, the process plant buildings listed below will be built. The capital cost for these buildings is included in the process plant cost estimate.

- primary crushing facility;
- process building (includes grinding, flotation, magnetic separation, cleaning and scavenging);
- crushed ore stockpile cover;
- plant workshop (part of process plant building) and warehouse reagent storage (part of process plant building);
- explosives manufacturing facility; and
- mine truck maintenance facility.

The cost also includes the supply of the electrics, fittings, and furnishing for the buildings, but excludes earthworks. The cost to supply power and water services to the buildings form part of the process plant cost.

21.3.4.1 Rail Spur

A rail spur, approximately 5 km long, services the process plant. The rail spur consists of a fuel delivery track near the mining truckshop and a freight delivery track north of the process plant. During the expansion phase, the track may be extended to the explosives manufacturing plant, approximately 3 km northwest of the process plant.

21.4 Basis of Estimate

21.4.1 Direct Costs

Direct costs are quantity based and include all permanent equipment, bulk materials, freight (inland and ocean), subcontracts, labour, contractor indirects and growth associated with the physical construction of the facilities.

The same estimate build-up and philosophy was used for both the 52.5 kt/d and the 105 kt/d expansion case, taking into account that the scope of work was different in certain areas.

21.4.1.1 Commodity Take-offs

Bulk material take-offs to a feasibility level were developed from arrangement drawings. Rates were obtained from quotes obtained from local contractors. These rates include the appropriate gang rate for the commodity and the actual cost of the permanent materials. Local freight associated with contractor-supplied material is included in the unit rates.

No imported fill is required. Aggregate material is available via an on-site crushing plant, purchased by the Owner. The crushing plant will be operated by the mine, and aggregate will be provided "free–issued" at the site of the crusher (i.e., the cost associated with production of aggregate has been provided for in the mining cost). Excess cut material can be stockpiled on site or consumed for future construction.

21.4.1.2 Labour rates

Labour rates have been built-up from first principles for different trades (welders, boilermakers, roofers, pipefitters, millwrights, storeman, crane operator, etc.). These rates have been based on the Quebec labour collective agreement (industrial sector) which can be found on the website, http://www.ccq.org, and the Guide for Employers 2013 - source deductions and contributions on the website, http://www.revenuquebec.ca.

These labour rates include the following:

- Base hourly rate;
- Contribution rate from collective agreement – industrial sector:
  - vacation, holiday and sick leave pay;
  - premiums;
  - safety, health and welfare;
  - compensation for safety clothing and equipment; and
  - social benefits and funds.
- Contribution rate from Revenue Quebec:
  - Quebec Pension Plan;
  - Quebec Parental Insurance Plan;
  - Health Services Fund;
  - Labour Standards Commission;
  - Workforce Skills Development and Recognition Fund; and
  - Compensation Tax.

The work week is 50 hours, which consists of 40 regular hours and 10 overtime hours. The 10 overtime hours are calculated as 4 hours x 1.5 (the regular rate) and 6 hours x 2.0. This is based on working Monday to Friday at 8 h/d regular; Monday to Thursday at 4 hours at time-and-a-half and 6 hours on Saturday at double time.

A crew make-up for a typical structural, mechanical and piping (SMP) contractor was developed to achieve an average hourly crew rate of $72.85/h.

Contractor indirect costs for structural, mechanical, piping, electrical and instrumentation have been developed with the assistance of well-established local construction contractors within Quebec; earthworks and concrete has been based on unit rates from contractors within Quebec. Distributable costs have been allocated by percentage in the estimate on a manhour basis and are inclusive of the following:

- salaries, salary burden, allowances and benefits for the contractor’s indirect labour, supervisory and management staff;
- staff recruitment and travel expenses;
- living out allowances;
- mobilization and demobilization;
- temporary buildings and facilities at site specifically for and used by the contractor;
- workshop equipment and supplies;
- vehicles and equipment used by staff during construction;
- construction equipment including cranes up to 100 tonnes;
- temporary construction power (diesel gensets);
- small tools and consumables;
- site office overheads, such as stationery, communications, light and power, first aid, security, etc;
- head office costs/contribution;
- financing charges;
- insurances;
- advertising; and
- profit.

The total SMP all-in labour rate is $155.62/h which includes the SMP base crew rate of $72.85/h and the addition of costs associated with the items listed above. This rate was confirmed by local SMP contractors. The electrical and instrumentation (E&I) rate is slightly less, at $140/h, due to less costs associated with construction equipment.

21.4.1.3 Equipment costs

Multiple quotes were sourced for all the mechanical equipment, with the exception of small pumps, agitators, load out scale, and mobile equipment, which were sourced from Ausenco’s database. The budget quotes cover 95% of the mechanical equipment cost. The lowest technical accepted quotes were chosen for each equipment type.

21.4.1.4 Freight

All bulk materials, plant and equipment items within the direct costs are based on delivered to store on Site. Where possible, plant and equipment has been obtained from budget quotes inclusive of the freight component, if not percentage allowances have been included, where applicable. For mechanical equipment, 4% of the equipment supply cost has been included for inland freight and 12% for ocean freight for items not sourced in North America. These percentages are average for projects executed in Canada.

21.4.1.5 Duties & Taxes

Where duties are applicable on imported items of supply, the appropriate allocation of funds have been included in the estimate.

All taxes are excluded unless otherwise stated.

21.4.2 Indirect Costs & Owner's Costs

Indirect costs include items that are necessary for project completion, but not related to the direct construction cost. These items are summarized in the subsections below.

21.4.2.1 Temporary Facilities & Services

Temporary facilities and services are items which are not directly attributable to the construction of specific physical facilities of the plant or associated infrastructure, but which are required to be provided during the construction period to support the construction and have been estimated in detail.

These costs include:

- EPCM office complex, HS&E services, security services, site vehicles, refuelling, bus transportation, recurring project costs, maintenance services, provision of temporary roads, temporary power, water, effluent disposal and other facilities as required. For the expansion phase, power required by the construction work is to be provided by the Owner.
- Heavy lift construction cranes. These represent cranes over and above what the construction contractor provides. These are cranes greater than 100 tonne capacity.

21.4.2.2 EPCM

The engineering, procurement, project and construction management budget has been compiled by the identification of resources over a defined schedule. A detailed assessment of consultants and project general expenses are also included in the EPCM costs. The EPCM estimate includes the following:

- Corporate Services;
- Project Services;
- Engineering;
- Drafting;
- Construction;
- Travel Expenses;
- Home Office Expenses;
- Site Office Expenses; and
- Consultants (geotechnician, shipping logistics specialist, surveys, soils and compaction testing, concrete testing, fire and safety).

21.4.2.3 Vendor Reps

Allowances for vendor representatives, for both installation supervision and for the commissioning component, are included and are based on vendors recommended support that were provided in the quotations. These have been incorporated where applicable. Where these were not provided in the quotation but still required, a 3% of equipment supply cost was included.

21.4.2.4 Construction Camp

There is no requirement for a construction camp. All labour can be sourced from Amos and within the Abitibi-Témiscamingue region.

21.4.2.5 Spares

Spares include capital spares, one year operational spares, and commissioning/start-up spares.

A spares list was developed from the budget quotations provided. Where spares were not priced in the quotation, a percentage of the equipment cost was applied to each spares category.

No spare SAG motor has been included.

An increase in spares inventory is allowed for in the expansion phase.

21.4.2.6 Commissioning Support

The direct installation hours do not include construction labour to assist the EPCM commissioning team. Costs for these are based on two crews consisting each of one electrical technician, two fitters and one trade's assistant for the duration of four months. For the expansion phase, only two months are included.

21.4.2.7 First Fills

An estimate for first fills for the following reagents has been included: KAX, MIBC, Aerofroth 65, Calgon, CMC, H_2SO_4, $CuSO_4$, flocculent and sodium hypochlorite. A 100% charge for 38 mm, 65 mm and 100 mm grinding balls is also included.

An allowance has also been made for oils, lubricants, hydraulics, and greases.

21.4.2.8 Modification Squad (Mod Squad)

The direct installation hours do not include post construction modifications to facilitate handover and acceptance by the Owner. Costs for these are included in the form of a "mod squad" and are based on a crew of two fitters, three boilermakers, two trade assistants and one electrical technician, for four months duration, and a $500 k materials allowance. For the expansion phase, only 50% of the cost of the 52,500 t/d mod squad is included, as lessons learned from construction will be incorporated in the expansion.

21.4.2.9 Owner's Costs

Owner's costs have been provided by the Owner. They include the following:

- Capitalized general and administration costs (to the end of Month 1 of mill production, or July 2016);
- Capitalized process operating costs (also to the end of July 2016);
- Recruitment costs;
- Orientation costs;
- Training costs; and
- Construction insurance costs.

21.4.2.10 Escalation

Escalation is excluded from this estimate.

21.4.3 Estimate Growth, Estimate Contingency & Accuracy

21.4.3.1 Growth Allowance

From the time the estimate is prepared to the time the facility is completely constructed, a number of detail variations that are not scope changes are expected to occur. Allowances have been included in the direct cost section of the estimate and are specified against line items.

The growth categories assigned to each line item are dependent upon what level of definition was obtained. The categories are:

- A Engineered 2%
- B Preliminary Engineering.. 4%
- C Sketch 7%
- D Estimated 10%
- N Nil Growth 0%

In this case, the growth allowance for both the initial and expansion capital cost was calculated to be 4.1% for the process plant. Nil growth has been applied to the mining, winter works, and indirect costs.

21.4.3.2 Estimate Contingency

An estimate contingency allowance has been included and is money that is expected to be spent. It is meant to cover additional costs that will be incurred as a result of final detailed design and investigation to provide a holistic estimate of the defined scope. It is not intended to be a provision for changes in scope and standards.

The value of the construction cost and estimate contingency represent an estimated project scope value of 100%. In this case the estimate total contingency is assessed at 9.0% for the initial capital cost, and 8.8% for the expansion, based on an analytical method addressing the elements of the estimate and assessing the estimate for scope, cost and confidence.

The contingency categories assigned to each line item are dependent on the level of definition obtained scope wise and the level of costing pricing wise. Both categories are combined to determine the specific line items overall contingency.

The scope categories are:

- A Engineered 5%
- B Preliminary Engineering.. 12%
- C Sketch 17%
- N Nil Growth 0%

The pricing categories are:

- A Tendered 5%
- B Budget Quote 7%
- C Current Project - Escalated 10%
- D Estimated 15%
- N Nil Growth 0%

Direct contingency percentages were applied to the following items:

- Mining fleet.. 5%
- Ancillary Equipment.. 10%
- Site Preparation 8%
- Owners Pre-Strip 5%
- Contractors Pre-Strip... 8%
- TSF 10%
- G&A Capitalized Operating Costs 10%
- Owners Contingency.. 10%

No contingency has been applied on growth.

The estimate contingency does not allow for the following:

- the effect of abnormal weather conditions, over and above normal weather conditions;
- any changes to market conditions arising during the course of the project that could affect the cost of labour or materials;
- changes of scope within the general production and operating parameters outside the detailed scope of work defined by this feasibility study;
- special industry award allowances in addition to those included in the labour rates; and
- effects of industrial disputes.

The above items will be part of the Owner's contingency.

21.4.4 Exclusions

- project finance and interest charges;
- foreign exchange hedging;
- residual value of temporary equipment and facilities;
- residual value of any redundant equipment;
- cost to Owner of any downtime;
- currency fluctuations;
- escalation;
- impact caused by modifications directed by government authorities, including schedule;
- increased costs due to early works (e.g., concrete requirements before there is a batch plant on site);
- removal, remediation, or disposal of hazardous/contaminated materials encountered during construction;
- costs of any special requirement due to the participation of outside financing sources; and
- costs to identify, locate, remove or relocate existing underground obstructions or utilities.

21.4.5 Project Deferred & Sustaining Capital

Ongoing capital requirement for the mine production period totals $823 M over the mine life, which includes a credit of $22 M for the release of first fills at the end of project life. Items covered under sustaining capital include:

- Ongoing clearing of land prior to pushbacks of the pit or extension of waste dumps.
- Purchase of new production and auxiliary fleet for the mine (in response to longer hauls as the pit deepens) and replacement fleet (as the initial generation of equipment reaches the end of its economic life). In the case of haul trucks, which represent 60% of total expenditures on fleet over the life of mine, a life of 100,000 hours has been assumed – this equates to a life of approximately 16 years.
- Expansion of the workshop that will be required as the fleet expands. The initial workshop of 6 bays will be expanded to 10 bays during the expansion. Subsequent to the expansion, a further 10 bays will be added under sustaining capital.
- Ongoing expansion of the TSF.
- Construction of a water treatment plant in 2022. Note that costs associated with this plant are included under 'Area 3 Process' in Table 21-4.
- General plant and infrastructure replacements, that are expected to total $250 M over the life of project. These have also been included under 'Area 3 Process' in Table 21-4.

21.5 Operating Cost Estimate

21.5.1 Summary

This section details the estimated operating costs for mining, process plant and general and administration (G&A) for the Dumont project. Costs are presented in Q2 2013 Canadian dollars, unless stated otherwise. The estimate is considered feasibility study level with an accuracy of ±15%.

Operating costs were estimated in the following manner:

- Operating costs for the open pit were based on the production schedule, performance parameters for mining equipment as recommended by OEMs, the current cost of commodities and labour rates for the Abitibi region, as determined from two different salary surveys.
- Operating costs for the concentrator were based on rates of consumption for reagents and other consumables determined from metallurgical testwork and a labour structure that is appropriate for the current flowsheet.
- The operating cost estimate for the concentrator includes those costs associated with operating the TSF.
- G&A costs were based on the level of support required for the operation.
- Costs for treatment and refining of concentrate were based on the commercial terms discussed in Section 18, and the scheduled production of concentrate.
- Processing operating costs were calculated exclusive of variability from design throughputs (e.g., neglects ramp-up period, etc.).

A summary of life-of-mine (LOM) operating costs is provided in Table 21-6.

Table 21-6: Operating Cost Summary

	Units	52.5 kt/d 2016-2020	105 kt/d 2021–2036	Stockpile 2036–2049	LOM Average
Mine	$/t ore milled	$6.61	$6.15	$0.77	$3.89
Mine[1]	$/t ex-pit material mined	$1.63	$1.69	$0.00	$1.68
Process	$/t ore	$5.04	$4.76	$4.76	$4.78
G&A	$/t ore	$0.94	$0.56	$0.41	$0.52
Site Costs	$/t ore	$12.60	$11.46	$5.94	$9.18
Site Costs	$/lb	$3.45	$4.15	$3.59	$3.90
TC/RC	$/lb	$1.45	$1.40	$1.43	$1.42
Gross C1 Cash Cost	$/lb	$4.90	$5.55	$5.02	$5.32
Byproduct Credits	$/lb	($0.46)	($0.51)	($0.61)	($0.53)
Net C1 Cash Cost	$/lb	$4.44	$5.04	$4.41	$4.79

Note: **1.** To give a true reflection of expit mining costs, excludes $61 M for rehandle of 103 Mt stockpile ore during ex-pit mine life.

21.5.2 Key Assumptions

Key assumptions used in generating the operating cost estimates are given below.

- C$ prices for goods and services obtained prior to the cost basis date of Q2 2013 have been escalated to this date using average Canadian producer price index (PPI) for the period January 2010 to December 2012 of 2.57% per annum.
- US$ denominated prices for goods and services obtained prior to the cost basis date of Q2 2013 have been escalated to this date using average Canadian producer price index (PPI) for the period January 2010 to December 2012 of 2.85% per annum.
- Labour costs were estimated based on the organizational structure developed for each area and the rates of pay are based on wages and benefits at existing mining operations in the Abitibi region of Quebec and salary survey data collected by Coopers Consulting and PWC.
- Based on discussions with Hydro-Quebec, it has been assumed that the project would qualify for the “L Tariff.” The forecast price of $44.45/MWh based on Hydro-Quebec pricing effective April 2013.
- The forecast long-term diesel price of $0.94/litre is based on forecast long-term oil prices of US$90/bbl and a C$ F/X rate of US$0.90.

21.5.3 Mining Operating Costs

A summary of mining costs by function and category is provided in Tables 21-7 and 21-8, respectively.

Table 21-7: Mining Operating Cost Summary – By Function

Activity	units	Total	Capitalized Pre-Strip	Expensed	% of Total
Contractor	$M	166	50	116	2.5%
Owner by Process:					
Production Drilling	$M	234	7	227	5.0%
Production Blasting	$M	626	16	610	13.3%
Pre-Split Drilling & Blasting	$M	46	0	46	1.0%
Loading	$M	320	8	312	6.8%
Hauling	$M	2,153	28	2,125	46.4%
Low-Grade Ore Rehandle	$M	313	0	313	6.8%
Roadstone	$M	116	4	112	2.5%
Support Equipment	$M	180	10	170	3.7%
Auxiliary Equipment	$M	24	2	22	0.5%
Maintenance Labour	$M	330	13	317	6.9%
Management, Technical & Admin Total	$M	224	13	211	4.6%
Total	$ M	**4,732**	**151**	**4,581**	**100.0%**
$/t material		1.88	0.06	1.82	
$/t ore		4.01	0.13	3.89	

Table 21-8: Mining Operating Cost Summary – By Category

		Total	Capitalized Pre-Strip	Expensed	% of total
Contractor	$M	166	50	116	2.5%
Owner by Area:					
Labour cost	$M	1,127	40	1,087	23.7%
Consumables	$M	958	19	939	20.5%
Maintenance	$M	1,148	20	1,128	24.6%
Diesel	$M	1,240	20	1,220	26.6%
Power	$M	45	1	44	1.0%
Other	$M	48	1	47	1.1%
Total	**$M**	**4,732**	**151**	**4,581**	**100.0%**
Unit Rate	$/t rock	1.88	0.06	1.82	
	$/t ore	4.01	0.13	3.89	

Contractor mining represents 2.5%. The majority of the contractor scope of work includes removing all clay overlying the deposit during the initial 5 years of mining (including the pre-strip). Contractor mining costs were based on a competitive tendering process that has led to the pre-selection of Norascon as the mining contractor. Norascon has worked closely with the feasibility study team on many aspects of the study.

Hauling is the largest single cost activity, representing almost 50% of total mining costs. For the site-specific parameters at Dumont, the 230t class trucks that have been selected represent the most economic option, as larger trucks have less favourable ratio of tare-weight-to-payload, and thus forecast to consume more diesel on a unit basis and travel slower on uphill hauls. Diesel represents approximately 45% of total hauling costs. As will be discussed in Section 24, the use of trolley-assisted truck haulage could reduce diesel consumed by the production haul truck fleet by approximately 35 ML, or 35%.

Table 21-8 indicates that diesel, equipment maintenance and labour each represent approximately 25% of the total mining cost. In addition to the 1,066 ML of diesel that would be consumed by the ex-pit production haul truck fleet, a further 250 ML would be consumed by drills, excavators, support equipment during the rehandling of low-grade ore. Additionally, 35 ML of diesel would be added to ammonium nitrate and emulsifiers in the manufacture of explosives.

Key assumptions regarding the cost of equipment maintenance are based on budgetary quotations provided by OEMs.

Labour averages 331 persons over the life of project, including 463 persons while the pit is active, and 116 during reclaim of the low-grade stockpile. These totals do not include personnel allocated to TSF construction, who would average a further 34 persons during the 20 years that construction of the TSF is active.

Consumables represent the majority of the remaining owner mining operating costs. This category includes but is not limited to drilling bits, ground engaging tools (GET), truck tires and explosives. Power costs represent approximately 1.0% of owner mining costs. This reflects the relatively low price of power in Quebec.

21.5.4 Process Plant Operating Costs

The processing plant operating costs are based on the flowsheets described in Section 17. The battery limits for the determination of process operating costs begins with the crushing facilities and end with the TSF, and include plant services.

21.5.4.1 Basis of Estimate

The process plant operating costs were determined from first principles using input from a variety of sources, including:

- process design criteria;
- reagent and equipment supplier quotations;
- staffing levels for processing plant estimated by Ausenco;
- personnel salaries and overheads based on information from similar projects in the region and survey data presented by Coopers Consulting and PWC;
- client recommendations; and
- previous study assessments.

21.5.4.2 Inclusions

The process plant operating cost estimate includes all direct costs associated with the production of nickel concentrate.

Included in the Ausenco operating cost estimate are the following:

- labour for supervision, management, and reporting of on-site organizational and technical activities directly associated with the processing plant;
- labour for operating and maintaining plant mobile equipment and light vehicles, process plant, and supporting infrastructure;
- labour and operational costs for the laboratory;
- costs associated with direct operation of the processing plant, including all reagents, consumables, and maintenance materials;
- maintenance materials used in operating and maintaining the mobile equipment and light vehicles;
- cost of power supplied to the process plant from the power grid;
- operational costs of the waste water treatment facilities; and
- general operations associated costs including consultants, training and general supplies.

21.5.4.3 Exclusions

The plant operating costs exclude the following:

- corporate overheads;
- escalation or exchange rate fluctuations;
- mine operating costs other than grade control assays;
- exploration labour and operating costs;
- environmental permits;
- contingency;
- import duty and taxes;
- sustaining capital;
- interest and financing charges; and
- mine or plant closure/rehabilitation activities.

21.5.4.4 Process Plant Operating Costs Summary

The plant is designed for an initial ore throughput of 52.5 kt/d followed by an expansion to 105 kt/d, both at an availability of 92.0%. Processing costs include labour, power, maintenance materials, reagents and consumables, mobile equipment, and ongoing metallurgical testing. Summarized costs provided in Tables 21-9 and 21-10 include the capitalization of the first month of process operating costs (included in Owner's costs in the capital estimate) and also account for the six-month ramp up to full production for both mill lines. The estimated overall operating cost for the initial processing plant is $5.04/t of ore milled, reducing to $4.76/t of ore milled after expansion.

Table 21-9: Process Plant Cost Summary– Initial Phase at 52.5 kt/d

Area	units	Total	Capitalized	Expensed	$/tonne	M$/annum
Ore Milled	**Mt**	**84**				
Labour	$ M	47	1	46	0.56	11
Power	$ M	140	1	139	1.65	32
Maintenance Materials	$ M	74	1	73	0.86	17
Reagents and Consumables	$ M	160	2	158	1.88	36
Miscellaneous	$ M	7	0	7	0.09	2
Total	**$ M**	**428**	**5**	**423**	**5.04**	**98**

Table 21-10: Process Plant Cost Summary– Expanded Phase at 105 kt/d

Area	units	Total	Capitalized	Expensed	$/tonne	M$/annum
Ore Milled	**Mt**	**1,095**				
Labour	$ M	425	0	425	0.39	15
Power	$ M	1,820	0	1,820	1.66	64
Maintenance Materials	$ M	820	0	820	0.75	29
Reagents and Consumables	$ M	2,075	0	2,075	1.90	72
Miscellaneous	$ M	68	0	68	0.06	2
Total	**$ M**	**5,208**	**0**	**5,208**	**4.76**	**182**

21.5.5 General & Administration (G&A)

The estimated cost for G&A expenses is based upon the level of service required for the size of Dumont's operation and takes into account existing local services. The costs summarized in Table 21-11 are almost entirely fixed in nature, with the result that unit costs at 105 kt/d fall to little more than half the rate for the initial 52.5 kt/d scope of project.

Table 21-11: G&A Cost Summary– Initial Phase at 52.5 kt/d

Area	units	Total	Capitalized	Expensed	$/t	M$/a
Ore Milled	**Mt**	**84**				
Labour	$ M	27	6	21	0.25	5
Consumables	$ M	3	1	2	0.03	1
Maintenance	$ M	0	0	0	0.00	0
Power	$ M	1	0	1	0.01	0
Diesel	$ M	1	0	1	0.01	0
Other	$ M	64	10	54	0.64	12
Total	**$ M**	**96**	**17**	**79**	**0.94**	**18**

Table 21-12: G&A Cost Summary– Expanded Phase at 105 kt/d

Area	units	Total	Capitalized	Expensed	$/t	M$/a
Ore Milled	**Mt**	**1,095**				
Labour	$ M	133	0	133	0.12	5
Consumables	$ M	18	0	18	0.02	1
Maintenance	$ M	3	0	3	0.00	0
Power	$ M	11	0	11	0.01	0
Diesel	$ M	4	0	4	0.00	0
Other	$ M	368	0	368	0.34	13
Total	**$ M**	**537**	**0**	**537**	**0.49**	**19**

The largest line item in Tables 21-11 and 21-12 are for 'Other' items. Table 21-13 lists the 'Other' items that are fixed in nature:

Table 21-13: G&A – Other Cost Items

		$/t	
	$000/a	**52.5 kt/d**	**105 kt/d**
Consultants	$400	0.021	0.010
Conferences	$15	0.001	0.000
Travel	$100	0.005	0.003
General Admin Costs/office supplies	$100	0.005	0.003
Security Contract	$300	0.016	0.008
Cleaning Contract	$250	0.013	0.007
Garbage Disposal	$100	0.005	0.003
Recruitment Budget	$120	0.006	0.003
HR & Training Budget	$800	0.042	0.021
Municipal Taxes	$826	0.043	0.022
Insurance	$1,200	0.063	0.031
Audit	$250	0.013	0.006
IT general	$500	0.026	0.013
Environmental Budget	$1,202	0.063	0.031
First Aid, medical etc	$85	0.004	0.002
Loss Control	$800	0.042	0.021
Community Relations	$500	0.026	0.013
Shipping / Purchasing Budget	$500	0.026	0.013
Total	**$8,048**	**0.420**	**0.210**

'Other' items that are variable in nature include:

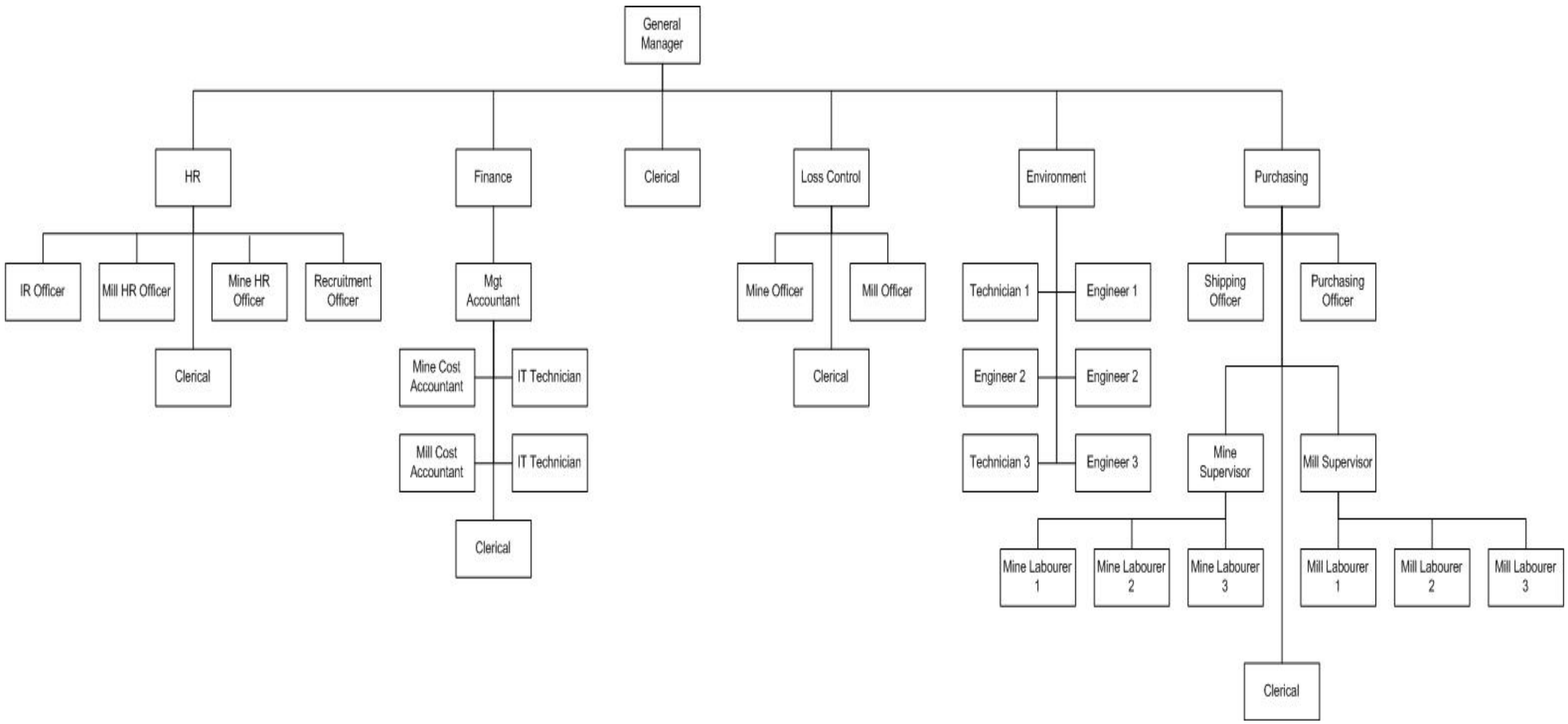
- Employee transport. Based on quotations from local transportation companies, costs of $5,900 per employee annually are estimated for 12 hour shift workers and $7,700 per employee annually for personnel working a five-day week.
- Computer hardware and software. Employees using computers would incur costs of $5,100 pa, which accounts for 80% of personnel using common productivity software with a three-year replacement schedule while the remaining 20% would require more expensive software with annual licenses.
- Safety equipment. All employees would receive $945 pa worth of personal protective equipment.

The other major cost item is labour, which has been based on the organizational structure shown in Figure 21.1 (overleaf).

21.5.6 Contingency

Contingency is not included in the operating cost estimate.

Figure 21.1: G&A Organizational Structure



22 ECONOMIC ANALYSIS

22.1 Summary

This economic analysis of the Dumont Feasibility Study focuses on the base case, which includes use of conventional (diesel powered) truck haulage and does not include the use of trolley-assisted trucks. The base case also assumes production of a nickel concentrate that would be sold to third parties, and does not include the potential benefits from magnetite as a byproduct.

Salient metrics for this base case are presented in Table 22-1. Note that costs and returns have been presented using the assumed macro-economic parameters provided in the next section, as the exchange rate is expected to vary in the short to medium term, the IRR for the C$ and US$ cases differs. As per the summaries in Section 21, capital costs include first fills of consumable items and their associated release at the end of project life.

All amounts expressed are in Canadian dollars unless otherwise indicated.

Table 22-1: Feasibility Study Summary Metrics

	Unit	C$ Basis	US$ Basis
Ore Mined	Mt	1,179	1,179
Payable Ni	Mlbs	2,774	2,774
Payable NiEq[1]	Mlbs	2,922	2,922
Gross Revenue	$/t ore	24.88	22.40
TC/RC	$/t ore	3.33	3.00
Net Smelter Return	$/t ore	21.54	19.40
Site Operating Costs	$/t ore	9.18	8.27
Gross C1 Costs	$/lb Ni	5.32	4.79
Net C1 Costs	$/lb Ni	4.79	4.31
Initial Capital	$M	1,268	1,205
Expansion Capital	$M	997	898
Sustaining Capital	$M	823	741
Total Capital	$M	3,088	2,844
Pre-Tax $NPV_{8\%}$	$M	2,293	2,003
Pre-Tax IRR		19.5%	18.7%
Post-Tax $NPV_{8\%}$	**$M**	**1,330**	**1,137**
Post-Tax IRR		**15.9%**	**15.2%**

Notes: 1. Based on the production profile given in Table 16-1 previously and the price profiles given in Table 22-2.

22.2 Assumptions

Table 22-2 provides price and exchange rate assumptions that have been used in the base case evaluation. With the exception of the forecast price for acid (a key reagent in the mill process), forecasts are based on consensus forecasts by Canadian base-metals equity analysts.

Table 22-2: Price & Exchange Rate Assumptions

Item	Units	2016	2017	2018+
Ni	US$/lb	$10.00	$10.50	$9.00
Co	US$/lb	$14.00	$14.00	$14.00
Pt	US$/oz	$1,800	$1,800	$1,800
Pd	US$/oz	$700	$700	$700
Oil	US$/bbl	$90.00	$90.00	$90.00
Acid	US$/t	$76.80	$79.28	see below
C$ F/X	C$ = US$	$0.95	$0.90	$0.90

Other key assumptions included in the base case analysis are as follows:

- Each of the two process plant lines would ramp up to nameplate production of 52.5 kt/d over six months.
- The metallurgical recovery for Ni as forecast by the model is based on the Standard Test Program (STP) of 105 samples. LOM recovery is forecast to average 43.0%. The average metallurgical recovery for Co is assumed to be 42.0%, almost equal that for Ni, which is based on the understanding of Co deportment to recoverable minerals and associated approximate recoveries for these minerals. The average recovery of Pt and Pd is based on the results of lock-cycle testwork, with recovery expected to average 62.5% and 60.7% for Pt and Pd, respectively.
- Off-site costs are US$64/t concentrate for transport (average based on shipment to a variety of destinations).
- Long-term electricity prices of $44.45/MWh, which is based on the current L-rate tariff for Quebec and Dumont's expected demand profile.
- Long-term prices for acid of US$72/t in 2018, US$71/t from 2019-2024 and US$70 from 2025 onward that were based on a market study performed by the consulting group CRU Strategies.
- The following assumptions are based on the prior experience of RNC management:
  - US$175/t concentrate for smelter treatment and US$0.80/lb for nickel refining inclusive of price participation. This equate to US$1.20/lb over the project life.
  - The cost of refining byproduct cobalt and PGE was assumed to equate to a further US$0.07/lb Ni over the project life (US$3.00/lb for Co and US$50/oz for PGE).
  - Smelter deductions for nickel and cobalt are assumed to be 7% and 50% (for payables of 93% and 50%, respectively). Deductions for PGE are assumed to be 1 g/t, with the average concentrate grade of 4.3 g/t resulting in life-of-project payables of 77%.
- Working capital has been calculated based on the following (based on the prior experience of RNC management unless otherwise noted):
  - Contractual terms for the sale of concentrate would make provision for payment for 90% of concentrate value within 30 days and the remaining 10% in 60 days.
  - Accounts payable would be settled within 30 days.
  - First fills for the mine and G&A areas have been calculated based on a stores holding of one month for all consumable items with the exception of tires (four months), diesel (five days) and electricity (no holding). No advance purchase of mine maintenance items would be required as these would be held on a consignment basis. First fills for the process plant have been calculated by Ausenco from first principles.

NPV is reported using a discount rate of 8%. NPV is expressed in real, Q2 2013 terms with the start date for discounting being the commencement of project construction in September 2014. No material expenditures are included in the economic analysis prior to this date.

Results were calculated on a pre-tax and post-tax basis. The post-tax results included the following assumptions regarding the fiscal regime:

- Planned changes to income taxes announced in the 2013 federal budget have been included, specifically:
  - The 41A category, which allows for accelerated depreciation of a portion of initial capital plant purchases, will be phased out by 2020.
  - The CEE category, which provides for accelerated depreciation of all initial development expenditures, will be phased out by 2018.
  - The investment tax credit will be phased out by 2016.
- Planned changes to the Quebec Mining Tax Code announced in March 2013 will be in place by the time the project commences production. These include:
  - Application of a minimum tax ranging from 1-4% depending on profitability. The methodology used to calculate pre-tax income for this minimum tax is new, and does not allow for accelerated depreciation of pre-production capital expenditures, so the minimum tax is incurred soon after the start of commercial production.
  - A variable tax that is applied to pre-tax income calculated in a manner similar to the previous legislation. The rate varies from 16% for a pre-tax profit margin of ≤ 35% to 28% for a pre-tax margin of 50% or more.

The calculated royalty payments include the assumption that the historic 2% and 3% NSR royalties will be bought down to 1% and 1.5%, respectively, as is provided for in the contracts. The buy-down would occur when the mine achieves commercial production. The calculated royalty payments include the Red Kite 1% NSR and assume that the 0.8% NSR royalty owned by Ressources Québec will be bought out in August 2017, as provided for in the contract.

22.3 Base Case Results

The total life of project can be subdivided into the following periods:

- Construction for a period of 22 months, starting in September 2014;
- Initial production at a concentrator throughput rate of 52.5 kt/d for 54 months to the end of December 2020;
- Expanded production from the open pit, at a concentrator throughput of 105 kt/d, for 186 months (14.5 years) to the end of June 2036; and
- Production from stockpiles following the completion of open pit mining. The concentrator continues to operate at a rate of 105 kt/d for an additional 158 months (12 years, 2 months) to the end of August 2049.

Summary metrics for each of these periods are presented in Table 22-3. It can be seen that the cumulative NPV to the end of pit life is $930 M or 70% of the project total. The remaining 30% of project NPV ($399 M) is realized during the period that the low-grade stockpile is reclaimed, with the benefits of lower costs offsetting lower grade and recovery.

Table 22-3: Summary of Economic Metrics by Period

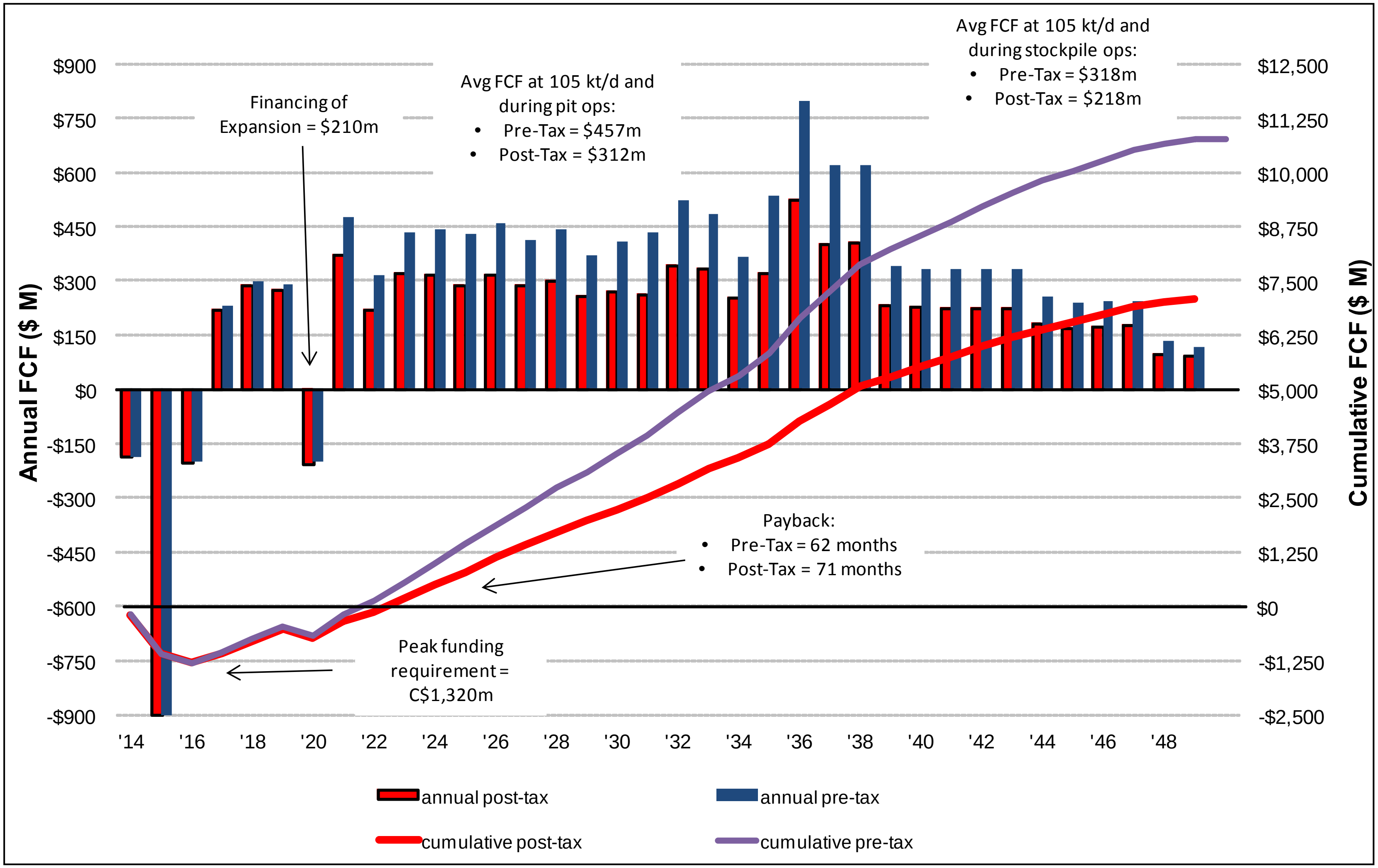
Item	Construct	'16 – '20 52.5kt/d Pit	'21 – '36 105kt/d Pit	'36 – '49 105k Stockpile	Total
Ore Mined (Mt)	21	204	954	0	1,179
Total Mined (Mt)	55	338	2122	0	2,514
Stripping Ratio (waste:ore)	1.62	0.66	1.22	0	1.13
Ore Milled (Mt)	0	84	592	503	1,179
Grade (% Ni)	0.25	0.34	0.28	0.24	0.27
Concentrator Recovery (% of Ni)	0	52.7	47.8	33.9	43.0
Payable Ni (Mlbs)	0	307	1,634	833	2,774
Annual Payable Ni (Mlbs)[1]	0	68	105	63	84
Annual Payable NiEq (Mlbs)	0	71	111	67	88
Net C1 Cash Costs (/lb Ni)	0	4.44	5.04	4.41	4.79
Initial Capital (M)	1,243	25	0	0	1,268
Expansion Capital (M)	0	997	0	0	997
Sustaining Capital (M)	0	12	725	86	823
Total Capital (M)	1,243	1,034	725	86	3,088
Closure + Working Capital (M)	20	51	47	(73)	45
Post-Tax NPV $_{8\%}$ (M)	(1,183)	424	1,690	399	1,330
Post-Tax IRR					**15.9%**

Figure 22.1 provides a life of project graph of cash flow. The following information is highlighted:

- The peak funding requirement of $1,320 M (in 2013 real dollar terms) is reached three months after the start-up of commercial operations (the operation is forecast to be operating cash flow positive during the first quarter of operation and free cash flow positive from the second quarter of operation).
- The financial returns are unlevered and assume 100% of the initial capital will be provided from equity. However, it is likely that a portion of the capital will be provided from debt. The assumed timing of the expansion has been based on the estimated time to repay the initial debt financing. A potential project debt financing of approximately 60% would result in initial borrowings of approximately $800 M, sufficient free cash flow to repay this amount of debt finance could be achieved within 34 months of operation, which would allow for re-investment in the expansion of concentrator capacity to 105 kt/d. Approximately 80% of the investment required for the expansion would be generated from internal free cash flows during the construction period, with additional capital of approximately $210 M required. The expansion is commissioned after month 54. Following expansion to 105 kt/d, annual post-tax free cash flow averages approximately $312 M/a for the period that the pit is operational (or $457 M/a on a pre-tax basis).
- Payback of all invested capital (including the expansion) is achieved approximately six years after initial start-up.
- The project generates in excess of $218 M post-tax free cash flow annually, while the low-grade stockpiles are being treated ($318 M/a on a pre-tax basis).

Table 22-4 provides detailed metrics for the life of mine cash flow, with time periods presented as years after start-up (start-up planned for Q3 2016).

Figure 22.1: Life of Project Cash Flow



Source: RNC.

Table 22-4: Detailed Economic Metrics

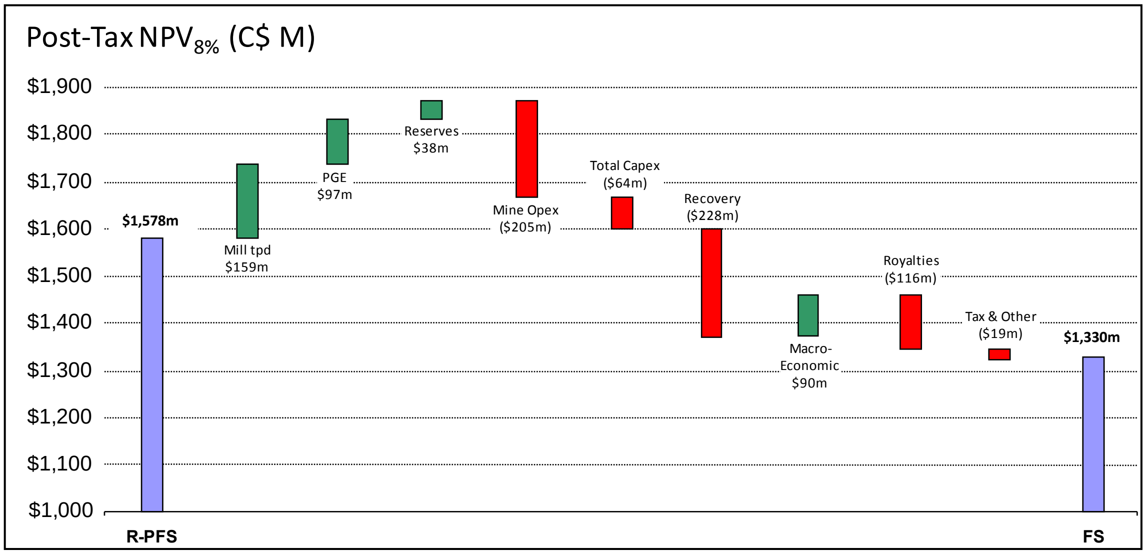
Item	Units	Total	Pre-Prod'n	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11 - 20	Yr21-34
Ore Processed	Mt	1,179	0	17	19	19	19	26	38	38	38	38	38	384	505
Payable Ni	Mlb	2,774	0	43	68	81	81	84	95	98	107	99	117	1,069	832
Payable Co	Mlb	59	0	1	1	1	1	2	2	2	2	2	2	22	21
Payable Pt	koz	157	0	1	4	4	4	4	4	4	5	5	5	59	58
Payable Pd	koz	332	0	2	10	11	10	9	9	9	12	11	11	124	114
Gross Revenue															
Nickel	$ M	$27,847	$0	$487	$732	$805	$814	$840	$945	$976	$1,072	$991	$1,171	$10,690	$8,324
By-Products	$ M	$1,483	$0	$17	$34	$38	$37	$39	$47	$47	$54	$50	$52	$560	$508
Total	$ M	$29,331	$0	$504	$766	$843	$851	$879	$992	$1,023	$1,125	$1,042	$1,223	$11,250	$8,833
Treatment and Refining															
Nickel	$ M	$3,707	$0	$63	$100	$109	$111	$109	$124	$128	$145	$139	$157	$1,406	$1,116
By-Products	$ M	$222	$0	$3	$5	$5	$5	$6	$7	$7	$8	$8	$8	$84	$76
Total	$ M	$3,930	$0	$66	$105	$114	$116	$114	$132	$136	$153	$146	$165	$1,490	$1,193
Net Smelter Return	**$ M**	$25,401	$0	$438	$662	$729	$735	$765	$860	$887	$972	$895	$1,058	$9,759	$7,641
Operating Costs															
Mining	$ M	$4,581	$0	$103	$125	$127	$144	$112	$161	$194	$220	$262	$225	$2,522	$386
Processing	$ M	$5,631	$0	$84	$97	$97	$97	$131	$182	$182	$182	$182	$182	$1,822	$2,393
G&A	$ M	$616	$0	$16	$18	$18	$19	$18	$21	$22	$21	$22	$21	$212	$208
Total Operating Costs	$ M	$10,828	$0	$202	$239	$242	$260	$261	$364	$399	$423	$467	$428	$4,556	$2,987
Net C1 Cash Costs	$ / lb	$4.79	$0.00	$5.77	$4.58	$3.95	$4.16	$4.01	$4.76	$4.99	$4.88	$5.67	$4.62	$5.13	$4.41
Capital Costs															
Initial	$ M	$1,268	$1,243	$25	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Expansion	$ M	$998	($0)	$15	$78	$116	$556	$233	$0	$0	$0	$0	$0	$0	$0
Sustaining	$ M	$823	$0	$5	$5	$1	$1	$3	$59	$168	$46	$27	$55	$366	$87
Working Capital & Closure	$ M	$45	$20	$62	$7	$7	($68)	$85	($17)	$0	$10	($9)	$18	$29	($99)
Royalties & Taxes															
Total NSR Royalties	$ M	$659	$0	$25	$57	$18	$18	$18	$21	$21	$24	$22	$21	$233	$181
Cash Federal Income Taxes	$ M	$1,448	$0	$0	$0	$0	$0	$28	$44	$44	$50	$41	$65	$593	$583
Cash Provincial Income Taxes	$ M	$1,151	$0	$0	$0	$0	$1	$23	$35	$35	$40	$32	$51	$470	$464
Cash Mining Tax	$ M	$1,070	$0	$4	$11	$15	$10	$10	$13	$18	$37	$39	$58	$489	$366
Cash Flow															
Pre-Tax	$ M	$10,781	($1,263)	$103	$275	$346	($32)	$164	$432	$299	$469	$389	$536	$4,576	$4,487
Post-Tax	$ M	$7,112	($1,263)	$99	$264	$331	($44)	$103	$339	$202	$342	$277	$362	$3,023	$3,077

Source: RNC.

22.4 Reconciliation to Revised Pre-Feasibility Study

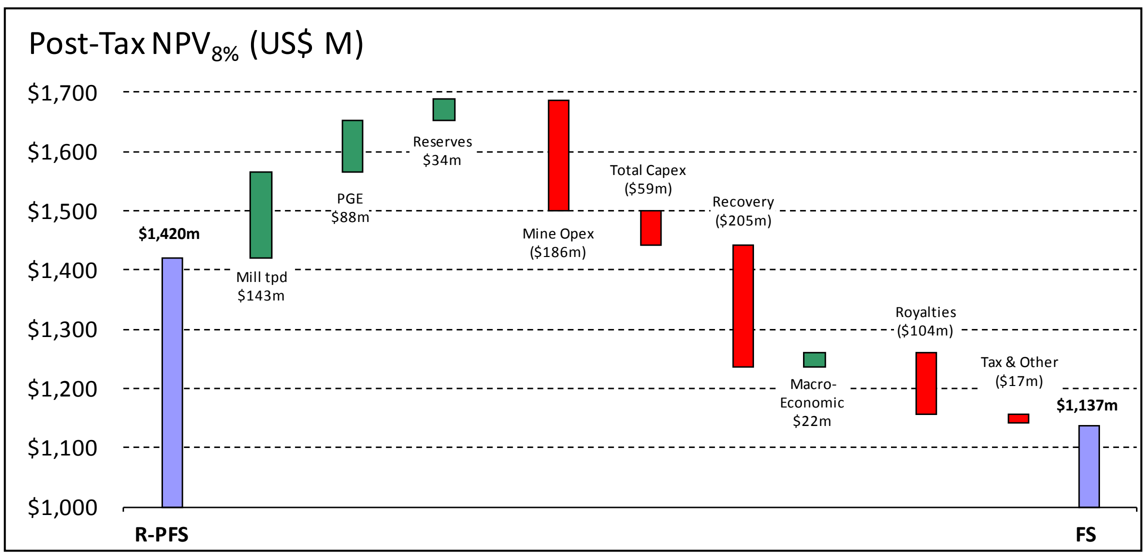
Figures 22-2 and 22-3 provide waterfall graphs that illustrate changes to project NPV, since the revised Prefeasibility Study (PFS) in C$ and US$ terms, respectively.

Figure 22.2: Changes to Project NPV (C$ terms)



Source: RNC.

Figure 22.3: Changes to Project NPV (US$ terms)



Source: RNC.

Key items leading to the change in NPV are as follows:

- Increasing the throughput of each mill line by 5% to 52.5 kt/d will accelerate Ni output and reduce the impact of fixed overhead costs (mainly G&A expenses). The benefit to the project of the increased throughput is $159 M (US$144 M).
- For the revised PFS, approximately 75% of PGE mineralization was classified as Inferred Resources, primarily due to uncertainty regarding the ability to recover low-grade material, and was thus excluded from reserves. The remainder (approximately 250 koz) would produce average concentrate grades of 0.9 g/t, which is less than the smelter deduction of 1.0 g/t, resulting in materially no payable metal. Further testwork performed during the FS confirmed the ability to achieve PGE concentrate grades in excess of 4 g/t, resulting in payable production of 490 koz. The benefit to the project of payable PGE output is $97 M (US$88 M).
- The pit shell has been expanded, to yield an additional 112 Mt ore, representing approximately three years' additional production at the full production rate of 105 kt/d. The additional ore provides a benefit of approximately 3% to the project NPV, or $38 M (US$34 M).
- The mining costs forecast for the FS ($1.64/t ex-pit material excluding rehandle of low-grade stockpiles, or $1.82/t ex-pit material when the cost of rehandling low-grade stockpiles is included) is approximately 19% higher than the cost forecast for the revised PFS. Approximately 11% of the cost increase results from use of conventionally powered trucks, compared to trolley-assist that was assumed for the revised PFS. The NPV impact of trolley assist was $46 M in the revised PFS (US$41 M); it is expected a similar benefit would be realized for the FS design.
- Mining costs are also higher due to increased use of smaller hydraulic excavators, particularly in the initial years of mine production when they will provide increased flexibility at a lower capital cost than the rope shovels that were planned in the revised PFS. As well, the layout has been revised with TSF and dump locations effectively reversed (to minimize the impacts of dust and noise to nearby communities), resulting in longer haulage profiles. As the total impact of higher mining costs is $205 M (US$186 M), the impact of higher cost mining equipment and longer hauls (excluding the impact of trolley-assist) is estimated at $159 M (US$145 M).
- The initial capital cost for the FS is only 3% higher in Canadian dollar terms, with increased costs for the process plant being partially offset by reduced pre-stripping and associated mining costs along with a stronger Canadian dollar and the impact this will have on US dollar denominated purchases. The capital cost for the expansion is 22% higher, due to higher costs for the plant and tailings storage facility (TSF), while sustaining capital costs are 11% lower. The net impact of higher capital costs is $64 M (US$59 M).
- The overall recovery of Ni is approximately two percent lower than forecast in the revised PFS, due to changes in the recovery equations from additional testwork, specifically in the Hz Dominant Domain, and a more conservative capping of the block model to reflect the input limits of the STP data set. The impact of lower recovery is $228 M (US$205 M).
- In US dollar terms, the initial capital cost for the FS is 8% higher.
- The FS incorporates a profile for prices for metals, oil (and hence diesel fuel), acid and the exchange rate over the short to medium term. This profile is based on the consensus of analyst forecasts. From 2018 onwards, flat long-term prices are used (with the exception of acid – only reaches long-term price in 2025). The net impact of the FS macro-economic forecast is an improvement in the C$ NPV of $90m, reducing to only $22 M in US dollar terms, due to variation in the exchange rate. Note that the profile in the exchange rates

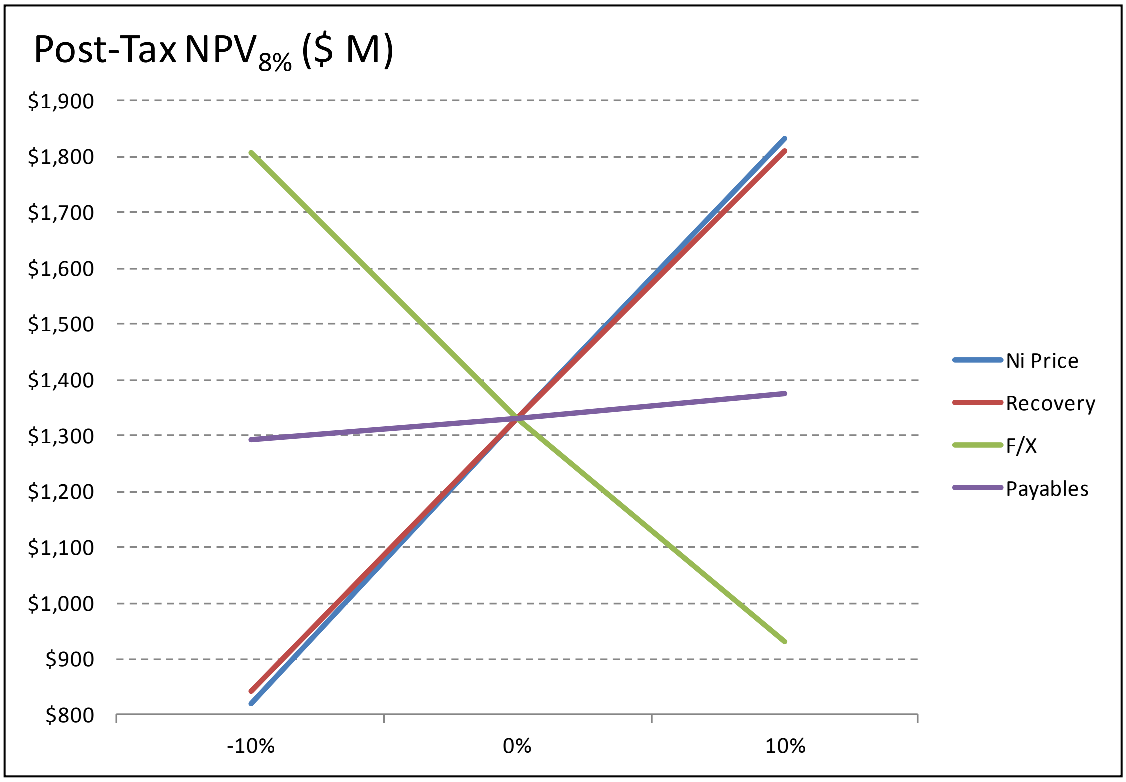
assumed is the reason the Canadian dollar IRR of 15.9% is higher than the US dollar IRR of 15.2%.

- Approximately 47% of the drop in NPV in Canadian dollar terms can be attributed to the sale of royalties to Resources Québec and Red Kite, with proceeds from these sales being used to fund project development. The impact in US dollar terms is lower, at 37% of the total drop.
- Revisions to both the corporate income tax and provincial mining tax regimes, along with a number of other factors, in aggregate account for a reduction in NPV of $19 M (US$17 M).

22.5 Sensitivity Analysis

The project is most sensitive to factors impacting on revenue as well as the Canadian vs. US dollar exchange rate. Figure 22.4 illustrates that a ±10% variation in any of the factors impacting revenue (Ni Price, Ni Recovery) is 37% and symmetric, with the percentage increase in NPV for higher revenue equal to the percentage decrease for lower revenue. Note that variation in recovery is on a relative and not an absolute basis. A change in exchange rate produces asymmetric outcomes, with the upside from a 10% decrease in the exchange rate (a 36% improvement in NPV) is greater than the reduction in NPV resulting from a 10% strengthening in exchange rate (30% decrease in NPV). Payables represents a ±10% change to the smelter deduction (base case assumption is 7%), with a 10% change resulting in a symmetric variation in NPV of 3%.

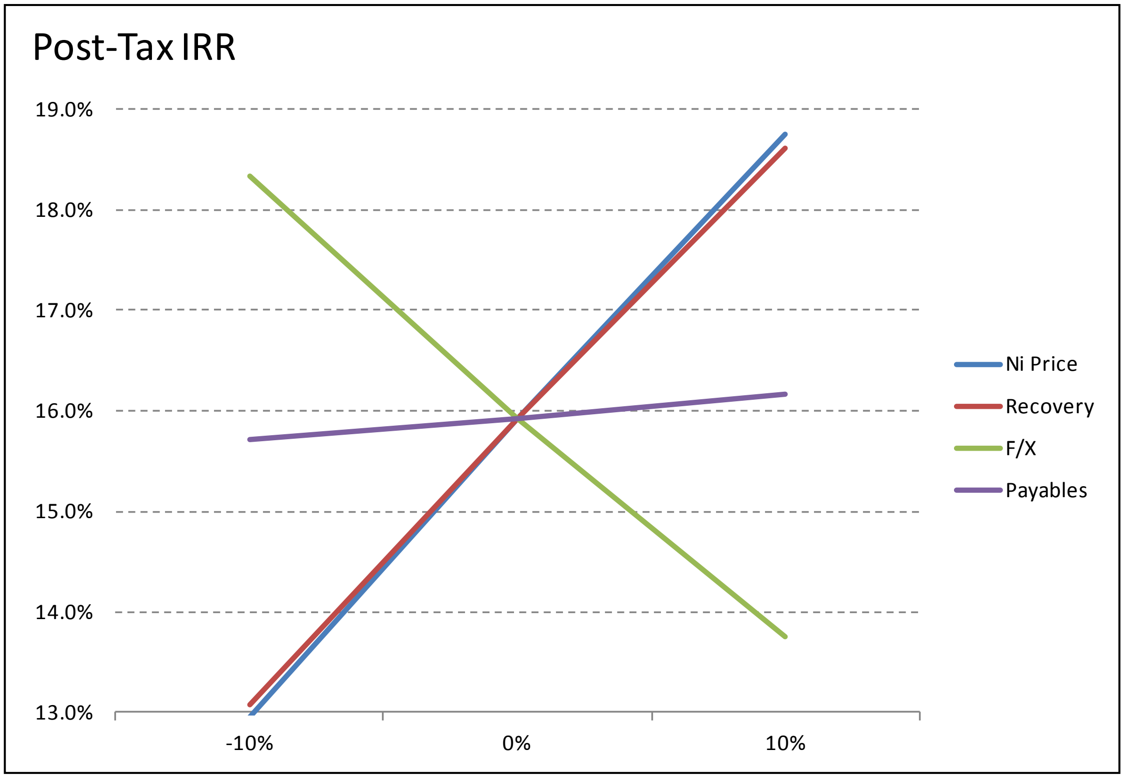
Figure 22.4: Sensitivity of Project NPV to Variation in key Assumptions



Source: RNC.

Figure 22.5 illustrates a similar relationship for the sensitivity of IRR to changes in the key parameters.

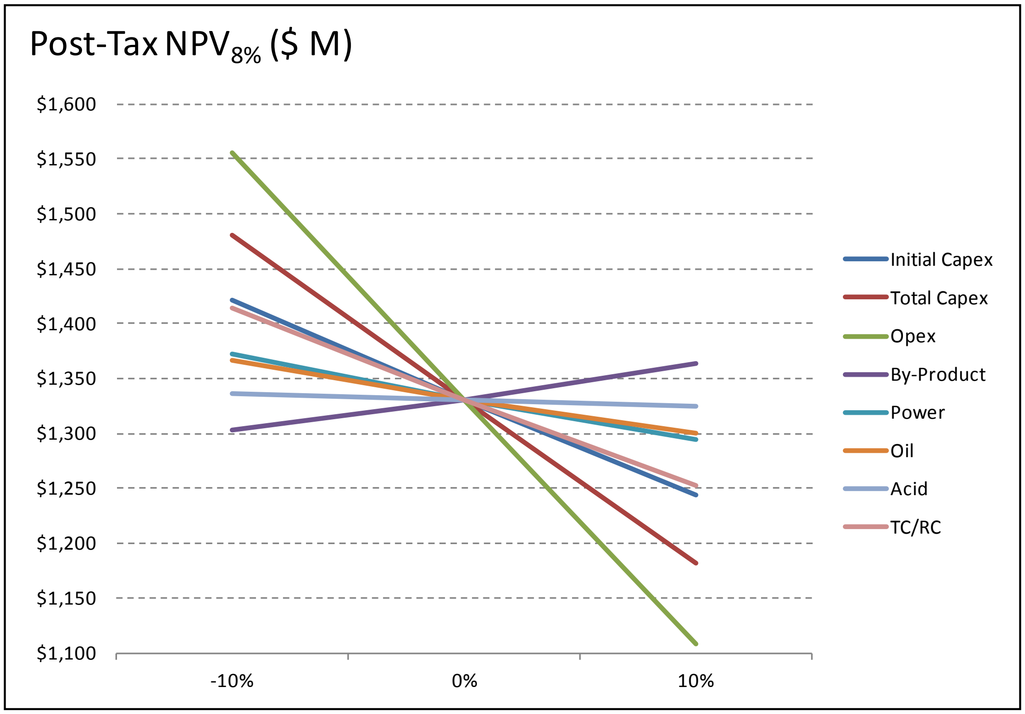
Figure 22.5: Sensitivity of Project IRR to Variation in Key Assumptions



Source: RNC.

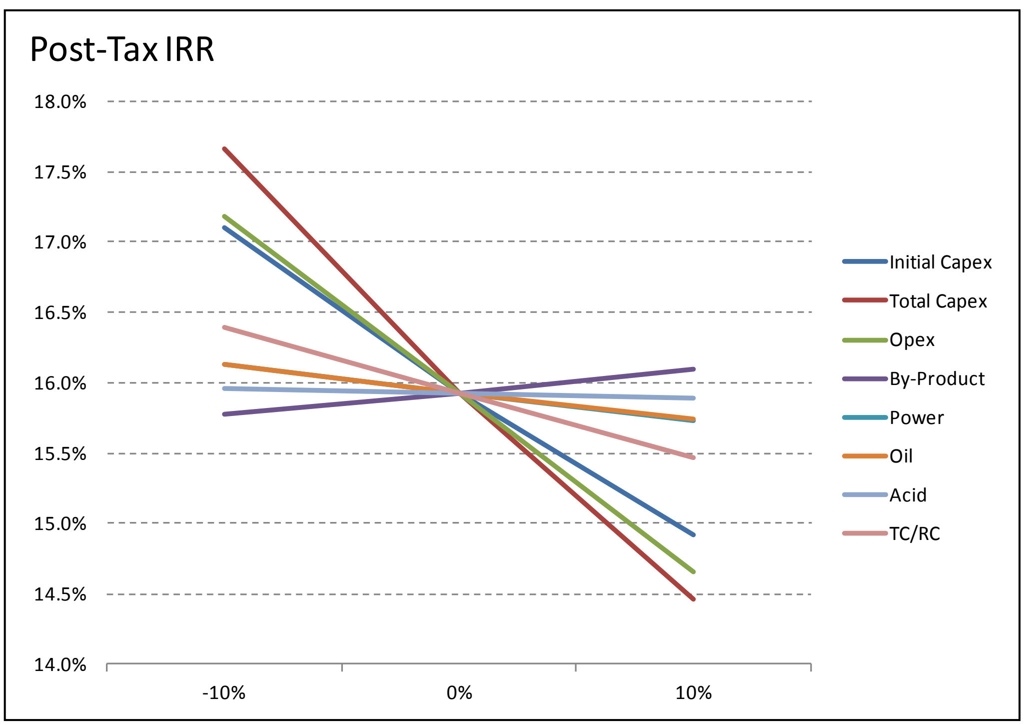
The project returns are less sensitive to the variation of other parameters – with a 10% variation in site operating costs having a 17% impact on project NPV. With the staged development plan, returns are less sensitive to capital costs and a 10% change in total capital cost has a lower impact, at only 11% of NPV. The impact of a 10% variation in TC/RCs is approximately half that of capital cost, at 6% of base case NPV. The project is less sensitive to variation in the cost of energy, with a 10% change in the price of either power or oil (diesel fuel) having only a 3% impact on project NPV. Project returns are insensitive to changes in byproduct prices (2% impact) or the cost of acid (<1% impact).

Figure 22.6: Sensitivity of Project NPV to Variation in Secondary Assumptions



Source: RNC.

Figure 22.7: Sensitivity of Project IRR to Variation in Secondary Assumptions



Source: RNC.

Tables 22-5 to 22-10 tabulate the sensitivity of the project NPV, IRR, cash flow, EBITDA and cash costs to the same parameters. Note that in all tables, Ni payables are expressed as the variance in smelter deductions (±10% = 0.7 percentage points from 92.3% to 93.7%). The following can be seen:

- At higher discount rates, the importance of capital cost and exchange rate increases relative to all other parameters.
- The post-tax breakeven Ni prices (NPV = $0) are as follows:
  - 8% = US$7.00/lb (22% lower than base case forecast);
  - 9% = US$7.25/lb (19% lower than base case forecast); and
  - 10% = US$7.50/lb (17% less than base case forecast)
- Cash costs are relatively insensitive to variation in the price of key consumables, with a 10% change in the prices of power and diesel (oil) having an impact of ~1% on gross cash costs.

Table 22-5: Sensitivity of Project NPV 8%

		Post Tax NPV			Pre-Tax NPV		
Discount Rate = 8%	**Units**	**-10%**	**0%**	**10%**	**-10%**	**0%**	**10%**
Ni Price	$ M	819	1,330	1,832	1,500	2,293	3,085
Recovery	$ M	841	1,330	1,810	1,533	2,293	3,052
Initial Capital Costs	$ M	1,421	1,330	1,244	2,410	2,293	2,175
Total Capital Costs	$ M	1,481	1,330	1,182	2,509	2,293	2,077
Operating Costs	$ M	1,556	1,330	1,108	2,642	2,293	1,943
Byproduct Price	$ M	1,303	1,330	1,364	2,249	2,293	2,337
F/X	$ M	1,809	1,330	932	3,059	2,293	1,666
Power	$ M	1,372	1,330	1,295	2,349	2,293	2,236
Oil	$ M	1,366	1,330	1,300	2,341	2,293	2,245
Acid	$ M	1,337	1,330	1,324	2,303	2,293	2,283
TC/RC	$ M	1,414	1,330	1,252	2,417	2,293	2,169
Accountability	$ M	1,292	1,330	1,374	2,232	2,293	2,353

Table 22-6: Sensitivity of Project NPV 9%

		Post-Tax NPV			Pre-Tax NPV		
Discount Rate = 9%	**Units**	**-10%**	**0%**	**10%**	**-10%**	**0%**	**10%**
Ni Price	$ M	596	1,057	1,508	1,183	1,893	2,603
Recovery	$ M	616	1,057	1,489	1,213	1,893	2,573
Initial Capital Costs	$ M	1,149	1,057	970	2,010	1,893	1,776
Total Capital Costs	$ M	1,206	1,057	910	2,102	1,893	1,684
Operating Costs	$ M	1,260	1,057	857	2,205	1,893	1,580
Byproduct Price	$ M	1,033	1,057	1,087	1,854	1,893	1,932
F/X	$ M	1,483	1,057	701	2,574	1,893	1,336
Power	$ M	1,094	1,057	1,026	1,943	1,893	1,843
Oil	$ M	1,090	1,057	1,030	1,936	1,893	1,850
Acid	$ M	1,063	1,057	1,052	1,902	1,893	1,884
TC/RC	$ M	1,133	1,057	987	2,004	1,893	1,782
Accountability	$ M	1,023	1,057	1,097	1,839	1,893	1,947

Table 22-7: Sensitivity of Project NPV 10%

		Post-Tax NPV			Pre-Tax NPV		
Discount Rate=10%	**Units**	**-10%**	**0%**	**10%**	**-10%**	**0%**	**10%**
Ni Price	$ M	409	827	1,235	917	1,556	2,195
Recovery	$ M	427	827	1,218	944	1,556	2,168
Initial Capital Costs	$ M	919	827	738	1,672	1,556	1,440
Total Capital Costs	$ M	974	827	681	1,759	1,556	1,354
Operating Costs	$ M	1,010	827	646	1,837	1,556	1,276
Byproduct Price	$ M	805	827	854	1,521	1,556	1,591
F/X	$ M	1,209	827	507	2,165	1,556	1,058
Power	$ M	860	827	799	1,601	1,556	1,512
Oil	$ M	857	827	802	1,595	1,556	1,517
Acid	$ M	832	827	822	1,564	1,556	1,548
TC/RC	$ M	895	827	763	1,656	1,556	1,457
Accountability	$ M	796	827	863	1,508	1,556	1,605

Table 22-8: Sensitivity of Project IRR

		Post-Tax IRR (%)			Pre-Tax IRR (%)		
IRR	**Units**	**-10%**	**0%**	**10%**	**-10%**	**0%**	**10%**
Ni Price	$ M	13.0	15.9	18.7	15.7	19.5	23.0
Recovery	$ M	13.1	15.9	18.6	15.9	19.5	22.9
Initial Capital Costs	$ M	17.1	15.9	14.9	20.9	19.5	18.2
Total Capital Costs	$ M	17.7	15.9	14.5	21.7	19.5	17.6
Operating Costs	$ M	17.2	15.9	14.7	21.0	19.5	17.9
Byproduct Price	$ M	15.8	15.9	16.1	19.3	19.5	19.7
F/X	$ M	18.3	15.9	13.7	22.5	19.5	16.7
Power	$ M	16.1	15.9	15.7	19.7	19.5	19.2
Oil	$ M	16.1	15.9	15.7	19.7	19.5	19.2
Acid	$ M	16.0	15.9	15.9	19.5	19.5	19.4
TC/RC	$ M	16.4	15.9	15.5	20.0	19.5	18.9
Accountability	$ M	15.7	15.9	16.2	19.2	19.5	19.7

Table 22-9: Sensitivity of Project Cash Flow & EBITDA

		Avg. Operating Cash Flow per Annum			Avg. EBITDA per Annum		
Cash Flow/EBITDA	**Units**	**-10%**	**0%**	**10%**	**-10%**	**0%**	**10%**
Ni Price	$ M	258	303	347	339	411	482
Recovery	$ M	260	303	345	342	411	480
Initial Capital Costs	$ M	301	303	305	411	411	411
Total Capital Costs	$ M	298	303	308	411	411	411
Operating Costs	$ M	323	303	283	443	411	379
Byproduct Price	$ M	300	303	306	406	411	415
F/X	$ M	351	303	263	487	411	348
Power	$ M	307	303	299	417	411	405
Oil	$ M	305	303	301	414	411	407
Acid	$ M	303	303	302	412	411	410
TC/RC	$ M	310	303	296	422	411	399
Accountability	$ M	299	303	307	405	411	416

Table 22-10: Sensitivity of Project Cash Costs

		Net Cash Costs			Gross Cash Costs		
Cash Costs	**Units**	**-10%**	**0%**	**10%**	**-10%**	**0%**	**10%**
Ni Price	$/lb Ni	4.68	4.79	4.89	5.22	5.32	5.42
Recovery	$/ lb Ni	5.16	4.79	4.48	5.76	5.32	4.96
Initial Capital Costs	$/ lb Ni	4.79	4.79	4.79	5.32	5.32	5.32
Total Capital Costs	$/ lb Ni	4.79	4.79	4.79	5.32	5.32	5.32
Operating Costs	$/ lb Ni	4.40	4.79	5.18	4.93	5.32	5.71
Byproduct Price	$/ lb Ni	4.84	4.79	4.73	5.32	5.32	5.32
F/X	$/ lb Ni	4.94	4.79	4.66	5.53	5.32	5.15
Power	$/ lb Ni	4.71	4.79	4.86	5.25	5.32	5.39
Oil	$/ lb Ni	4.74	4.79	4.83	5.28	5.32	5.36
Acid	$/ lb Ni	4.77	4.79	4.80	5.31	5.32	5.33
TC/RC	$/ lb Ni	4.64	4.79	4.93	5.18	5.32	5.46
Accountability	$/ lb Ni	4.81	4.79	4.76	5.35	5.32	5.29

23 ADJACENT PROPERTIES

There are no immediately adjacent mineral properties which affect the interpretation of the geology or exploration potential of the Dumont property.

24 OTHER RELEVANT DATA & INFORMATION

24.1 Base Case Design

24.1.1 Site Wide Geotechnical Overview

24.1.1.1 Introduction

Geotechnical data (for both soil and rock) was collected at the Dumont project area using a variety of field and laboratory procedures between late 2010 and late 2012. The work was performed in phases and addressed all major mine facilities as well as the proposed open pit. The site investigation commenced with the development of a terrain map based on aerial photographs. Overburden field investigation programs, including test pits, trenches, sonic drilling, metasonic drilling and cone penetration tests, were completed in the overburden soils around the property. Core drilling of the bedrock was largely confined to the proposed open pit and the plant site area. Laboratory testing was carried out on select samples to evaluate the engineering properties of each soil and rock unit. The results of these programs are summarized in a series of reports by SRK (SRK, 2011 and 2013).

A summary of the morphology of the local soils, the geotechnical data collection methods, and the corresponding findings, with a specific focus on the soils (overburden) at the property, are presented in the sections below.

24.1.1.2 General Morphology of the Soils

The region in and around the Dumont property has been glaciated numerous times. The bedrock surface has been shaped, in part, by glacial scour. Multiple glacial and post-glacial deposits are therefore responsible for most of the regional soils. These include till, glaciofluvial sand and gravel, glaciolacustrine silt and clay, fluvial sand and gravel. More recently, organic deposits have formed over much of the site.

Numerous subdued bedrock outcrops are evident over the site with a generally greater concentration of outcrops in the northern half of the site. Between the outcrops, discontinuous dense to very dense till pockets often overlie the bedrock. The till is typically overlain by dense to very dense glaciofluvial sand and gravel from outwash channels. Glaciolacustrine silt and clay of varying consistency are generally present in flat, low laying areas. Fluvial sand and gravel from post glacial process are found along the existing streams and rivers. Exclusive of the outcrops and active stream channels, most of the site is covered by topsoil and, in some depressions, by peat.

24.1.1.3 Site Investigation

Terrain Analysis

The terrain analysis was based on 1:15,000-scale black and white air photographs covering a 10 km x 9 km rectangle between longitudes 78°23.75’ and 78°34’ and between latitudes 48°36’ and 48°42’. The air photo interpretation was digitized and, with the benefit of surveyed bedrock outcrops, was superimposed on a 1:20,000 map. Available drill hole data were used to calibrate the interpretations, the results of which are summarized in a report and terrain map (SRK, 2011).

Cone Penetration Testing

Two phases of cone penetration testing (CPT) were undertaken using a track-mounted vehicle specifically built for CPT programs. In total, CPT probes were completed at 143 locations. The electronic piezocone measured parameters such as tip pressure, sleeve friction and porewater pressure every 5 cm as the cone was advanced into the ground. Pore pressure dissipation and seismic tests were carried out in select locations to provide additional information on soil characteristics. The CPT was terminated in each hole when the probe met refusal, which occurred in very dense soil or when bedrock was encountered. The CPT results from each probe hole are summarized as a series of plots showing tip pressure, sleeve friction and porewater pressure, along with the interpreted soil profile of the probe hole (SRK, 2011 and 2013).

Sonic Drilling Program

Drill holes were completed over the course of two drilling programs at 129 locations using a track-mounted sonic drill rig. The sonic drill employs high frequency, resonate energy to advance the core barrel and casing into the ground. The drill hole plan and locations were strongly influenced by site accessibility, particularly in relation to areas outside the proposed open pit. Core recovery was high and was complemented by various field performance tests to evaluate the geotechnical properties. The groundwater level was measured immediately after completion of each drill hole and a number of monitoring and pumping wells were installed for field permeability testing. An array of other laboratory tests were subsequently completed on select samples obtained during the drilling program. A log of each drill hole was prepared and the laboratory test results were added to the respective drill hole log (SRK, 2011 and 2013).

Other Field Programs

Overburden test pitting, trenching, metasonic drilling and overburden diamond drilling programs completed the site investigation. The test pits were completed using an excavator at 67 locations around the project area, while seven trenches were excavated at the plant site, also with an excavator. Metasonic drill holes were completed at 84 locations using a lightweight, tripod-mounted drill that penetrated soft overburden soils (clay and silt) but met refusal on compact/dense soils or bedrock. Lastly, a total of 27 NQ-sized holes were completed in the vicinity of the plant site using a diamond drill equipped with a split tube core recovery system.

24.1.1.4 Site Soils

The soils present at the project area typically fall into the following major classifications (in a generally descending stratigraphic order): (1) organic soil, (2) clay, (3) silt and silty soil, and (4) sand and gravel. The general geotechnical characteristics of each of these major soil classifications, as determined from the site investigation, are provided below.

Organic Soil

Where present, the organic soils are encountered at ground surface and typically consist of topsoil or peat with thicknesses ranging between 0.5 and 1.5 m. The topsoil is typically a dark brown, poorly graded, high plasticity material comprised of clay, silt, sand, some gravel and organics. The peat is brown, amorphous and likely soft to very soft. It tends to be present in flat or low lying areas and is, therefore, typically saturated.

Clay

The clay varies in thickness from approximately 2 to 15 m and is generally encountered in flat, low lying areas immediately below a layer of organic soil. The clay typically has medium to high plasticity and, as such ranges from CL to CH according to the Unified Soil Classification System

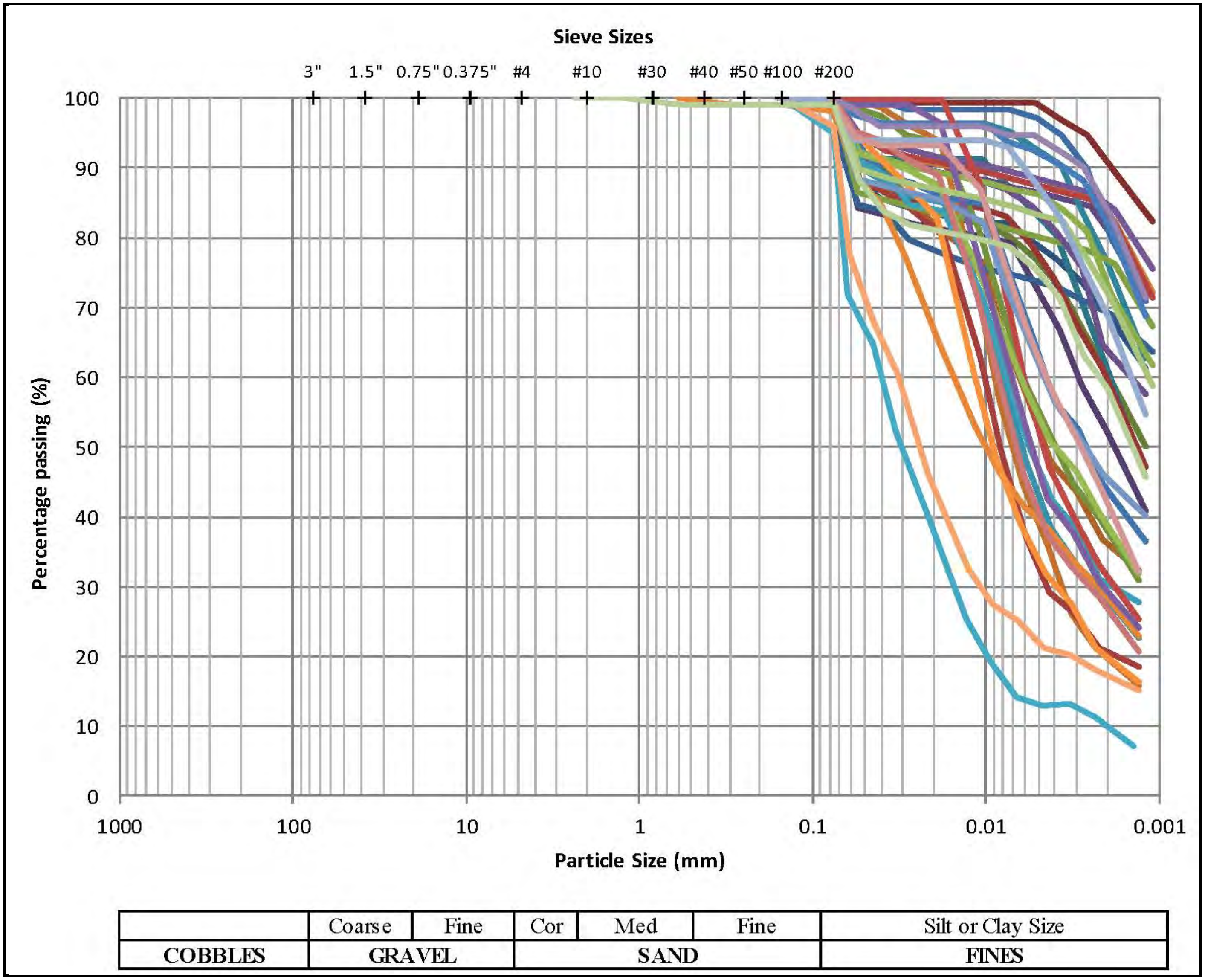
(USCS). The upper 1 to 1.5 metres of clay is generally light brown and moist with a firm to stiff consistency. This consistency can persist with depth if the total thickness of soil overlying bedrock is less than about 6 to 8 m. However, in locations where the clay is thicker than about 6 to 8m, the properties of the clay change significantly with depth. Over an interval of 1 to 2 m, the clay typically becomes grey and wet with a consistency that can range from very soft to firm. CPT data indicates the undrained shear strength of the grey clay in these locations can be as low as 10 to 30 kPa. Furthermore, the clay can be quite sensitive, meaning there can be a large difference between the peak (undisturbed) strength and the residual (highly disturbed) strength. Table 24-1 shows the summarized field and laboratory test results. Figure 24.1 shows the gradation results for the clay and Figure 24.2 summarizes the Atterberg Limits results in a plasticity chart.

Table 24-1: Field & Laboratory Test Results Summary for the Clay

Test Method	Number of Results	Minimum Value	Maximum Value	Average Value
Pocket Penetrometer				
Uniaxial Compressive Strength Undrained Shear Strength	235	No reading (1) No reading (1)	490 kPa (2) 245 kPa (2)	47.1 kPa 23.6 kPa
Torvane				
Peak Undrained Shear Strength Residual Undrained Shear Strength	222	No-reading (1) No-reading (1)	98 kPa(3) 49 kPa(3)	16.0 kPa 6.2 kPa
Nilcon Shear Vane				
Peak Strength Residual Strength Sensitivity Value	24	0.11 kPa 0.06 kPa 1.1	3.36 kPa 0.68 kPa 41	1.52 kPa 0.30 kPa 11
Moisture Content	97	22%	100%	51%
Atterberg Limits				
Liquid Limit Plastic Limit Plasticity Index Liquidity Index	83	20% 14% 4 0.1	75% 30% 47 3.8	47% 23% 25 1.3
Unit Weight	19	14.4 kN/m^3	20.3 kN/m^3	16.9 kN/m^3
Specific Gravity	20	2.63	2.87	2.74
CU Triaxial – 3 pts				
Friction Angle (ø') Cohesion (c')	7	25.3° 3.3 kPa	29.8° 11.4 kPa	27.6° 7.3 kPa
UU Triaxial – 3 pts				
Undrained Shear Strength	6	15.2 kPa	24.4 kPa	18.7 kPa
Consolidation				
Swelling Index (C_s) Compression Index (C_c)	14	0.02 0.36	0.679 3.14	0.06 1.27
Clay Mineralogy (XRD)				
Muscovite-Illite	5	2.6%	11.4%	7.2%
Montmorillonite		N/A	N/A	7%(4)
Clay Activity	31	1.5	3.5	2.2

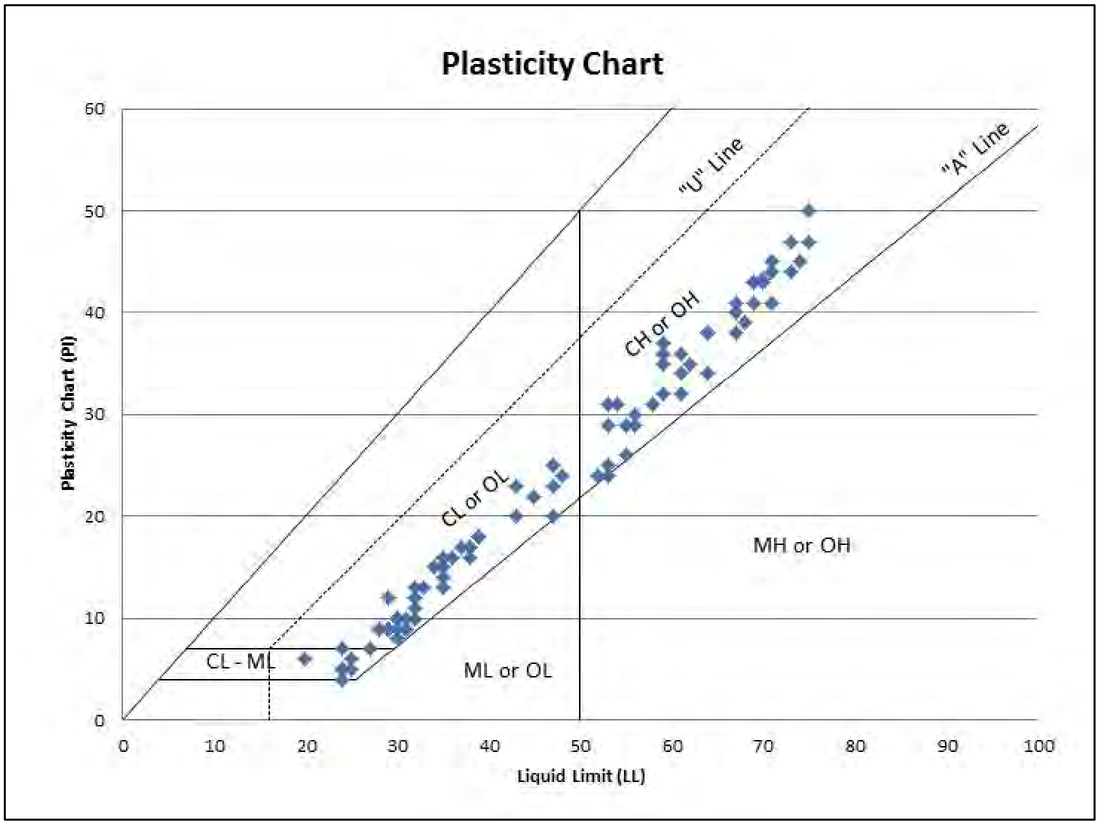
Notes: 1. "No-reading" means the soil strength was below the sensitivity of the respective instrument. **2.** This is the maximum recordable value of the penetrometer. The actual soil strength could be higher than the reported value. **3.** This is the maximum recordable value of the Torvane. The actual soil strength could be higher than the reported value. **4.** Montmorillonite was only found in one sample.

Figure 24.1: Gradation Summary for the Clay



Source: SRK.

Figure 24.2: Atterberg Limit Summary for the Clay



Source: RNC.

Silt & Silty Soil

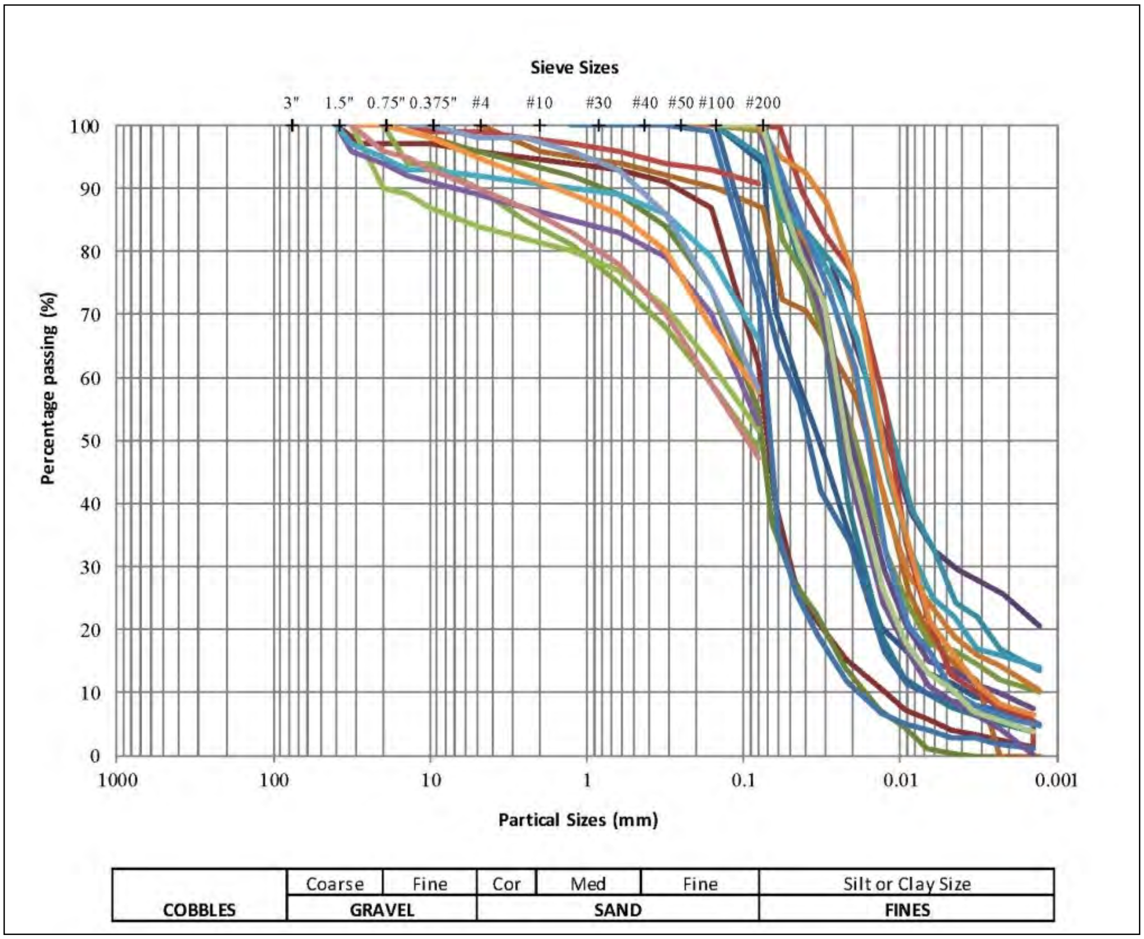
The silt and silty soil occur throughout the project area at various depths, with thicknesses that range from 1 to 16 m. They have been grouped as a single soil type that is generally associated with till or glaciofluvial deposits and can contain gravel, sand and clay. This unit ranges between having no plasticity and medium plasticity and, as such, ranges from CL-ML to MH. This soil is typically greyish-brown silt with more than 76% of material passing the No. 200 sieve. The silty material often contains lenses of coarse sand within a fine grained matrix. Overall, the silty soils found on site are soft to stiff, with estimated unconfined compressive strength values that range from 0 kPa to greater than 400 kPa. Furthermore, some of the field tests show the silt material will exhibit sensitivity potentially at a level similar to the clay unit. Table 24-2 shows the summarized field and laboratory test results for the silt and silty soil. Figure 24.3 shows the gradation results for this soil.

Table 24-2: Field & Laboratory Test Results Summary for the Silt & Silty Soil

Test Method	Number of Tests	Minimum Value	Maximum Value	Average Value
Pocket Penetrometer				
Uniaxial Compressive Strength Undrained Shear Strength	165	No-reading(2) No-reading(2)	490 kPa(3) 245 kPa(3)	106 kPa 53 kPa
Torvane				
Peak Undrained Shear Strength Residual Undrained Shear Strength	151	No-reading No-reading	98 kPa(4) 49 kPa	19 kPa 7.5 kPa
Nilcon Shear Vane				
Peak Strength Residual Strength Sensitivity Value	9	1.14 kPa 0.57 kPa 2.3	51.3 kPa 6.8 kPa 90	17.6 kPa 3.0 kPa 17
Moisture Content	20	18%	69%	25%
Limits				
Liquid Limit Plastic Limit Plasticity Index Liquidity Index	24	11% 14% 2 0.4	30% 22% 8 5.5	20% 19% 5 2.1
Unit Weight	8	15.5 kN/m^3	20.2 kN/m^3	18.0 kN/m^3
CU Triaxial – 3 pts				
Friction Angle (ø’) Cohesion (c’)	4	31.7° 0 kPa	34.5° 10.6 kPa	33.2° 4.45 kPa
Consolidation				
Swelling Index (Cs) Compression Index (Cc)	2	0.05 0.40	0.08 0.41	0.07 0.41

Notes: **1.** Average is not representative as only two test results were available. **2.** A no-reading mean the soil parameter is below sensitivity of the instrument used. **3.** This is the maximum recordable value of the penetrometer. The actual soil parameter could be higher than the reported value. **4.** This is the maximum recordable value of the Torvane. The actual soil parameter could be higher than the reported value.

Figure 24.3: Gradation Summary for the Silt & Silty Soil



Source: RNC.

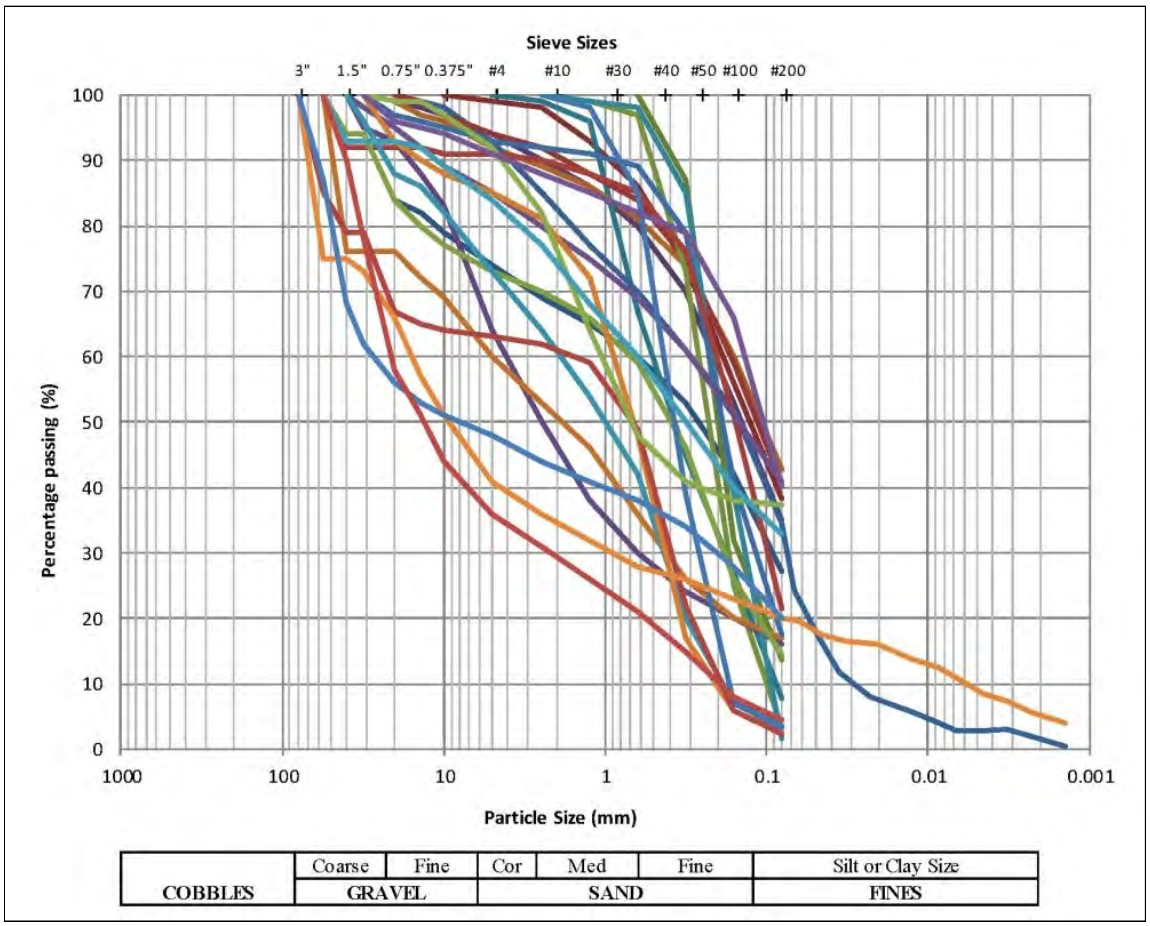
Sand & Gravel

The sand and gravel in the area have been grouped as a single soil type which is generally associated with post-glacial fluvial activity or as eskers from glacial outwash. The sand and gravel material is located in eskers and in lenses with the dense silty material with thicknesses that range between 1 and 40 m. It is typically a medium dense to very dense brown sandy material, well to poorly graded, with less than 8% of material passing the No. 200 sieve and up to 60%, by weight, as gravel. Table 24-3 shows the summarized field and laboratory test results. Figure 24.4 shows the gradation results for the sand and gravel deposits.

Table 24-3: Field & Laboratory Test Results Summary for the Silt & Silty Soil

Test Method	Number of Tests	Minimum Value	Maximum Value	Average Value
Standard Penetration Test (N-value)	32	3 blows/foot	+60 blows per foot	24 blows per foot
Pocket Penetrometer	102	No-reading	441 kPa	85 kPa
Torvane				
Peak Strength Residual Strength	28	No-reading No-reading	343 kPa 98 kPa	50 kPa 14 kPa
Moisture Content	52	1%	61%	12%

Figure 24.4: Gradation Summary for the Sand & Gravel



Source: RNC.

24.1.1.5 Borrow Materials

Borrow materials will be required, as a minimum, for the construction of the following:

- access roads;
- pads for laydown areas and working surfaces in the plant site area;
- dams (engineering components of the TSF and water management structures);
- containment structures for grey, wet clay at the overburden stockpiles; and
- containment structures for grey, wet clay excavated from the water management structures (ponds and ditches).

The specific materials required for use as borrow will largely consist of gabbro and basalt waste rock that would be mined as waste. These rock types will both be available in outcropping areas that will be mined during the pre-strip as well as throughout the entire life of mine and it is expected there will be an adequate supply for all construction requirements. In the event there was insufficient supply available in the short term, granular material, such as the sand and gravel that is present at depth over large portions of the open pit could also be used. Compactable clay is needed for dams at the TSF and for select parts of some water management structures. The firm to stiff brown clay that will be excavated from the open pit will be the principal source of compactable clay. All of the required borrow materials will be obtained from stripping (or quarrying) operations at the open pit.

24.1.1.6 Aggregate

The principal use of aggregate will be for concrete associated with the construction of the plant site infrastructure.

Testwork has confirmed the gabbro and basalt waste rocks are excellent materials for aggregate, containing very low levels of impurities that are deleterious to concrete. As with borrow material, gabbro and basalt would be produced by stripping operations, and then crushed at a plant located close to the primary crusher. This plant would subsequently be used for the roadstone crusher.

24.1.2 Plant Site

24.1.2.1 Location

The plant site will be located northwest of the open pit and east of TSF Cell 2, and will be aligned in a northwest-southeast direction to take advantage of the presence of a series of large bedrock outcrops. The suitability of these bedrock outcrops as a foundation for the large, settlement-sensitive structures required as part of plant site design was a key factor in the siting and layout of the plant site infrastructure.

24.1.2.2 Geotechnical Database

The geotechnical database in the proposed plant site is comprised mainly of bedrock outcrop mapping, test pits, trenches, metasonic drilling and overburden diamond drilling. A few CPT probes and sonic drill holes in the vicinity of the plant site complete the data base.

24.1.2.3 General Stratigraphy & Geotechnical Conditions

The overburden stratigraphy at the plant site mainly comprises granular material (sands and silts) overlying bedrock, with prominent outcrops in the south part of the mill and near the crusher. Fine grained materials (silt and clay) were encountered in some locations, but in general their presence is not common at the plant site. The usual thickness of the granular overburden is about 8 m and can change significantly over short distances. Deep overburden pockets up to 52 m of thickness can be found at the location of the ore feed stockpile.

24.1.2.4 Design Recommendations

Foundation design recommendations for the plant site infrastructure can be summarized as follows:

- Heavy, settlement-sensitive structures should be founded on bedrock. Foundations can be placed directly on rock in the outcrop sites and in shallow rock, or on piles when the rock surface is buried deeply.
- Lighter structures may be founded on shallow foundations placed on sand and silt provided they meet the bearing capacity and settlement requirements. Otherwise, they should be founded on shallow bedrock to the extent possible, but where this is not possible, consideration should be given to raft-style foundations or end-bearing piles.
- Shallow foundations should be placed 2.5 m below the ground surface in order to take into account the frost depth.

To the extent practicable, the infrastructure should avoid the grey, wet, very soft to firm clay. However, where it cannot be avoided, the selected foundation will have to satisfy the allowable settlement and bearing capacity criteria.

24.1.3 Project Implementation

24.1.3.1 Implementation Strategy

RNC recognizes that project implementation affects all aspects of project development, particularly capital cost, schedule, and risk management. As such**,** a preliminary project implementation strategy has been prepared.

RNC has prepared a strategy for the Project's implementation and contracting strategy and overall approach to construction. The resulting strategy has, and will continue to, guide the work being conducted in connection with the feasibility study. The strategy contemplates the development of the Project on an EPCM basis with the contractor being responsible for project design, purchase of supplies, equipment and services. Additionally, all or portions of the process plant may be constructed on a fixed price, turnkey EPC basis. The EPCM contractor, in these circumstances, would assist RNC in the management of the individual EPC contractors.

During the engineering phase of the project, the EPCM Contractor will develop a contracting plan setting out the scope prior to the EPCM Contractor award, certain construction packages for early site activities may be developed for tender and award. These contract packages may cover bulk earthworks packages, infrastructure work, construction power distribution, temporary facilities, site preparation and concrete supply, material, and equipment requirements for the field construction effort.

The Contract Packages are anticipated to include Major, Minor, Service and Technical Support Contracts similar to the following distribution. This distribution may be modified to fit Qualified Contractors ability to perform and support multiple discipline activities and have the corporation depth to man and provide the major construction equipment for such an effort.

Conversely, some areas of common construction may have multiple contractors working adjacent to each other in order to support the schedule or weather imposed time restraints, i.e.

Pre-engineered Process Building being divided between the Grinding Area, Floatation Areas and the balance of the building including Scavenging/Cleaning and Concentrate load out.

The EPCM Contractor is to understand, that even though the Contract "Philosophy" is for large horizontal contracts, flexibility to meet the schedule is important.

RNC will optimize opportunities to expedite a timely construction start and maximize the site construction progress prior to winter impacts.

Prior to mobilization, an EPCM Contractor kick off meeting will have been held with RNC and schedule, deliverable items and potential qualified contractors will already be selected and on board.

Work will begin on:

- the overall site development for access, stripping, bulk excavations, drainage control and work area development;
- preparation of the temporary facilities: trailers, laydown and warehouse areas;
- prepare the access construction road into the project site; and
- extension of the 13.8 kV power to the contractor area.

As soon as the EPCM Contractor trailer facility is complete, the EPCM Contractor will mobilize a limited field force to oversee and install the temporary power, fresh water relocation and coordinate the initial construction issues.

Initial major earthwork will be by the RNC. Mining group and will set the stage for mobilization of the early construction and supply contracts. Development of the site grading will open the site for the balance of the identified contracts and material/equipment receiving. At this point, the work will become discipline driven with multiple parallel operations.

As the detailed excavations continue and the areas open up for concrete, the project will be able to support construction activities on all fronts from the Primary Crusher through the concentrate load out.

Engineering and procurement activities will become construction driven to support the field and measures taken to establish winter weather protection with temporary structures and heaters.

Key to this is the erection of the grinding building over the SAG and ball mill areas. Structural steel can be erected during cold weather but consideration is to be given to roofing and siding installations concerning wind and snow. The building erection will need to be erected concurrent with the foundation work and precautions taken for overhead and ground personnel safety.

Procurement of the mill process buildings will be an early activity. To include all of the buildings with priority of:

- grinding bays;
- desliming;
- floatation;
- cleaning/scavenging;
- concentrate load out;
- stockpile storage enclosure; and
- primary crusher structure.

It is anticipated that the erection of these structures can be concurrent activities due to the size and distinct profile of each section.

Enclosure of the process buildings is critical in maintaining construction activities during the winter months and maintaining scheduled milestones.

Summer 2015 will be key for construction of the coarse ore stockpile enclosure and adjacent conveyors from the primary crusher and the sag mill feed conveyor.

Concurrent with the completion of the process equipment, conveyors and piping the final road grading, site grading, and cleanup will be done. The temporary construction facilities will be demobilized on a progressive basis, contractor contracts closed out and a systematic turnover of the project to the Operations Group will be completed.

The EPCM Contractor will supplement the team with commissioning engineers and technicians assigned to each defined commissioning area, and assist in the planning of work and completion of testing in each area.

The EPCM Contractor will be responsible to develop comprehensive commissioning safety and tagging procedures specific to the Dumont project. The procedures are to address the transition from construction to commissioning and from commissioning to RNC operations.

The definition of the project implementation strategy will continue to evolve where it will guide and inform commercial and logistical evaluations undertaken with the ultimate aim of optimizing and de-risking the project's development and construction.

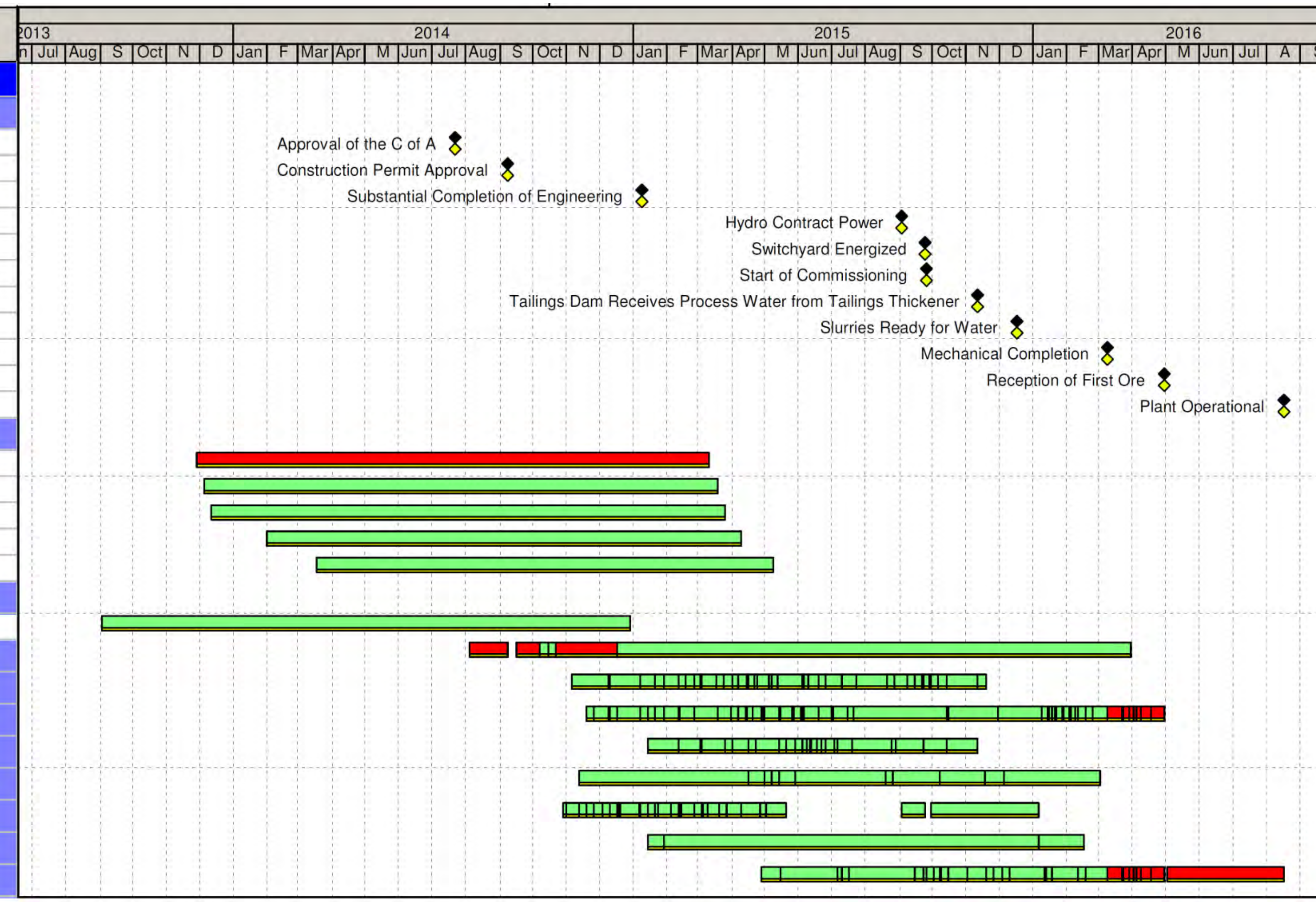
24.1.3.2 Project Schedule

The summarized project schedule is shown in Figure 24.5. The current schedule shows:

- The overall schedule duration from the start of basic engineering in order to procure long-lead equipment to the end of ore commissioning is 36 months. Key milestone dates are described in Table 24-4.
- The duration of the schedule is driven primarily by the construction permit approval, early purchase of long lead equipment, detailed engineering, and SAG mill installation.
- The approval of the C of A in Q3 2014.
- Approval of a Site Construction Permit is scheduled for Q3 2014.
- The project as described in this feasibility study will form the basis for the Environmental and Social Impact Assessment (ESIA).
- Geotechnical drilling for detailed engineering will commence in Q3 2013 and be completed by Q4 2013.
- Basic engineering will commence in Q3 2013, with a commitment to purchase major mechanical capital items like the mills, mill motors, primary crusher, and flotation cells in Q4 2013.
- Award of the EPCM contract will be in Q4 2013, with full engineering effort commencing in Q1 2014.

Figure 24.5: Summarized Project Schedule

ity ID	Activity Name	Original Duration	Start	Finish
Royal Nickel Implementation Schedule		154	03/Sep/2013	16/Aug/2016
Project Milestones		108	21/Jul/2014	16/Aug/2016
A11780	Approval of the C of A	0		21/Jul/2014*
A11790	Construction Permit Approval	0		08/Sep/2014*
A1210	Substantial Completion of Engineering	0		08/Jan/2015*
A23170	Hydro Contract Power	0	03/Sep/2015*	
A23180	Switchyard Energized	0		24/Sep/2015
A1360	Start of Commissioning	0	25/Sep/2015	
A23370	Tailings Dam Receives Process Water from Tailings Thickener	0	11/Nov/2015	
A7580	Slurries Ready for Water	0		16/Dec/2015
A1450	Mechanical Completion	0		09/Mar/2016
A1420	Reception of First Ore	0	29/Apr/2016	
A1440	Plant Operational	0		16/Aug/2016
Procurement		75	28/Nov/2013	08/May/2015
A23760	SAG Mill	67	28/Nov/2013	11/Mar/2015
A23770	Ball Mill #1	67	05/Dec/2013	18/Mar/2015
A23780	Ball Mill #2	67	12/Dec/2013	25/Mar/2015
A23790	Regrind Mill	62	31/Jan/2014	09/Apr/2015
A23800	Gyratory Crusher (Primary)	60	17/Mar/2014	08/May/2015
Engineering		69	03/Sep/2013	29/Dec/2014
A1240	Engineering - All	69	03/Sep/2013*	29/Dec/2014
100 - Mining		79	04/Aug/2014	30/Mar/2016
200 - Crushing		49	05/Nov/2014	18/Nov/2015
300 - Process		68	19/Nov/2014	29/Apr/2016
400 - Concentrate Loadout		40	14/Jan/2015	10/Nov/2015
500 - Tailings		61	12/Nov/2014	01/Mar/2016
600 - Utilities		55	29/Oct/2014	05/Jan/2016
700 - Onsite Infrastructure		52	14/Jan/2015	16/Feb/2016
900 - Indirects		61	28/Apr/2015	16/Aug/2016



Project Baseline Bar — Remaining Work — Baseline Milestone
Actual Work — Critical Remaining Work — Milestone

Page 1 of 1

TASK filter: Implementation Summary.

Table 24-4: Dumont Nickel Project Schedule – Key Milestone Dates

Criteria	Date
Commence Detailed Engineering for Long Lead Equipment	Q3 2013
Order Long Lead Equipment	Q4 2013
Commence Full EPCM	Q1 2014
C of A Approval	Q3 2014
Construction Permit Approval	Q3 2014
Substantial Completion of Engineering	Q1 2015
Hydro Contract Power	Q3 2015
Start of Commissioning	Q3 2015
Mechanical Completion	Q1 2016
Reception of First Ore	Q2 2016
Plant Operational	Q3 2016

The schedule considers the following broad contracting strategy and major equipment deliveries:

- SAG and Ball mills: 57 weeks (FOB China) for large mills; primary crusher: 50 weeks; flotation cells: 70 weeks (ordered in batches), fabricated in China.
- Tender long-lead items in Q4 2013 to enable commitments to be made soon after project approval is obtained.
- Lump sum tendering for all major contracts and purchases.
- Tendering with engineering drawings at 60% complete.
- Award a single contract to a mill supplier for the supply, transportation, installation, and commissioning of the mills.
- Fabricate structural steel and free issue to structural, mechanical and piping (SMP) contractor on site.
- Fabricate platework and free issue to SMP contractor on site.
- All equipment purchased by EPCM engineer on behalf of the principal, and free issued to SMP contractors.
- In the plant area:
  - one contractor for bulk earthworks, roads and drainage, and water dams, and tailings storage facility (TSF)
  - one or two civil contractors for detailed earthworks and concrete works; this contract would include the supply of all reinforcing bar, holding-down bolts, formwork, etc.
  - one or two SMP contractors erecting structural steel, and installing equipment, plate work, and pipe work. This contract would also include the supply of minor equipment and materials
  - one contractor for the electrical and instrumentation installation.
- Infrastructure:
  - one contractor for installation of the 10.5 km powerline to the Dumont site
  - one contract for the supply, transportation, and installation of the field construction facilities

- one contractor for the supply and installation of rail spur
- one contractor for the supply and operation of explosives facility
- one contractor to supply the mining fleet
- one contractor to execute the pre-strip earthworks
- one contract for the supply and installation of all field piping.

Ideally, the number of site contractors should be minimized, although this may be dictated by market and commercial considerations at the time.

24.1.4 Closure & Rehabilitation Plan

24.1.4.1 Background & Project Sequencing

The closure and rehabilitation plan was takes into consideration the specific requirements outlined in the Guidelines for Preparing a Mining Site Rehabilitation Plan and general Mining Site Rehabilitation Requirements (MRN and DEF 1997) published by the Quebec jurisdictional authorities.

The Dumont project will be comprised of five stages of significance to the implementation of the closure and rehabilitation activities described in the plan. The five stages and their principal operational activities are as follows:

- Stage 1 (Year 2 to start-up): Pre-stripping and early development of the pit; construction of the mine infrastructure including the starter dam of the TSF beginning with the first tailings cell (TSF Cell 1).

- Stage 2 (Start-up to approximately Year 6): Continued development of the open pit; mineral processing and operation of TSF Cell 1. While analysis conducted to date suggests that a water treatment plant would not be required at this stage of the project, it may prove necessary.

- Stage 3 (approximately Year 7 to Year 22): Development of the open pit to its maximum limit; mineral processing; construction/operation of the second tailings cell (TSF Cell 2) and construction of the water treatment plant (if not constructed in Stage 2) and operation as needed; closure and rehabilitation of TSF Cell 1 occurs early in this stage.

- Stage 4 (approximately Year 23 to Year 35): Processing of the ore from the low-grade ore stockpile, deposition of the tailings in the open pit and operation of the water treatment plant, as needed; closure and rehabilitation of TSF Cell 2 occurs early in this stage, along with closure and rehabilitation of the overburden stockpiles, portions of the waste rock dumps and the footprint of the low-grade ore stockpile; ongoing monitoring and potential rehabilitation of those elements closed during this or previous stages.

- Stage 5 (from Year 33): Operation of the water treatment plant, as needed; flooding of the open pits; closure of any remaining elements; and monitoring and potential rehabilitation of those elements closed during this or previous stages.

24.1.4.2 Objectives

The overall closure objective is to establish stable chemical and physical conditions that protect the environment and human health. To the extent practicable, rehabilitation efforts will endeavor to return the site to a condition which generally conforms to the surrounding terrain. The site will be monitored and maintained post closure in order to demonstrably meet these conditions.

In determining the most suitable closure strategy for each infrastructure component, the closure criteria were aligned with the requirements for closure and rehabilitation as detailed in MRN and DEF 1997. The following closure criteria were considered during this process:

- demolish and remove all construction and camp and industrial facilities and rehabilitation of affected footprints
- achieve long-term slope stability of all dumps/piles
- establish adequate vegetation density to ensure erosion protection of the soil slopes
- re-establish vegetation on areas returned to normal land use.

24.1.4.3 Closure & Rehabilitation Activities

The project facilities or areas, and the timing of the corresponding closure and rehabilitation activities, are summarized in Table 24-5. The general closure and rehabilitation activities for each of the project facilities or areas are summarized in Table 24-6.

Table 24-5: Summary of Project Elements & Rehabilitation Schedule

Project Facility or Area	Rehabilitation Timing by Stage
Plant Site and Mechanical Shop	Stage 5
Administrative and Office Complex	Stage 5
Explosives Plant	Stage 4
Fuel Storage Facility	Stage 5
Roads and Rail Spur	Progressive rehabilitation throughout Stages 4 and 5
Overburden and Reclaim Piles	Progressive rehabilitation throughout Stages 3, 4, and 5
Waste Rock Dumps	Progressive rehabilitation throughout Stages 3, 4, and 5
Low-grade Ore Stockpiles	Progressive rehabilitation throughout Stages 4 and 5
TSF	Progressive rehabilitation throughout Stages 3 and 4
Open Pit	Stage 5
Water Treatment Plant	Stage 5
Water Management System	Progressive rehabilitation throughout Stages 4 and 5

Table 24-6: Summary of Project Elements & Rehabilitation Activities

Project Facility or Area	Closure & Rehabilitation Activities
Plant Site and Mechanical Shop	Salvage where possible, demolish and appropriately dispose of the balance, rehabilitate hydrocarbon contaminated soils, regrade, cover as necessary and revegetate
Administrative and Office Complex	Salvage where possible, demolish and appropriately dispose of the balance, regrade, cover as necessary and revegetate
Explosives Plant	Salvage where possible, demolish and appropriately dispose of the balance, regrade, cover as necessary and revegetate
Fuel Storage Facility	Salvage where possible, demolish and appropriately dispose of the balance, rehabilitate hydrocarbon contaminated soils, regrade, cover with soil as necessary and revegetate
Roads and Rail Spur	Deconstruct all roads not needed post closure, and all rail lines, regrade, cover with soil as necessary and revegetate
Overburden and Reclaim Piles	Regrade as necessary and revegetate
Waste Rock Dumps	Regrade as necessary, cover with soil and revegetate
Low-grade Ore Stockpiles	Regrade footprint, scarify and revegetate
TSF	Cover with soil, revegetate and establish surface water drainage outlet/spillway at each cell
Open Pit	Drain pit of water from tailings disposal, refill with runoff and direct precipitation, establish spillway to un-named creek and revegetate the pit walls above the discharge level
Water Treatment Plant	Operate for as long as needed, then decommission and demolish, regrade, cover with soil and revegetate
Water Management System	Decommission as/when possible: regrade and revegetate

24.1.4.4 Post-Closure Monitoring & Maintenance

Post-closure monitoring to confirm that the closure objectives are being met will be based on the following typical activities:

- regular inspections of the various project facilities by professional engineers and revegetation specialists; and
- annual seepage and surface water sampling programs designed to monitor the changes in local water quality.

Maintenance will be performed on areas that monitoring identifies as needing repairs.

24.1.5 Cost Benchmarks

As will be discussed below, the capital and operating costs forecast for Dumont are considered achievable due to its unique site specific features and have been validated by appropriate benchmarks.

24.1.5.1 Capital Costs

The capital cost to construct Dumont is expected to be favourably impacted by its location and proximity to infrastructure already in place, along with features of the deposit that will be mined and processed.

Specifically:

- Proximity to the towns of Amos, Val d'Or and Rouyn-Noranda obviates the need for a construction camp.
- As the property is bordered by a highway and crossed by a rail line, no significant expenditures are required for transportation links.
- The property can be fed electricity from an existing high voltage line that is approximately 10km distant. The cost of installing a connection to this line will ultimately be borne by Hydro Quebec.
- The deposit has a low strip ratio and the starter pit outcrops at surface, resulting in a low tonnage of pre-stripping and modest initial fleet requirements.
- The process flowsheet is conventional
- The proximity to infrastructure, low tonnage of pre-stripping and straightforward mill flowsheet are expected to allow construction to be completed in a short time frame.

Key benchmarks that were referenced during the generation of the Dumont initial capital cost estimate of $1,268 M include the following:

- Based on public disclosure, Osisko's Canadian Malarctic gold mine was understood to be constructed for $1,029 M. This mine is located approximately 65 km from Dumont, and its scale and processing operations are similar to what is planned at Dumont. Note that capital costs for Canadian Malartic included approximately $170 M for community resettlement, which will not be required at Dumont. Construction of Canadian Malartic was completed within 18 months.
- Copper Mountain's open pit copper mine of the same name, located near Princeton, British Columbia. The mining rate for this operation is comparable to Dumont, while the nameplate concentrator throughput of 35 kt/d is approximately 65% of that planned at Dumont. Based on public disclosure, Copper Mountain was understood to be constructed for $429 M.

24.1.5.2 Operating Costs

Operating costs are also favourably impacted by location and features of the deposit that will be mined and processed. Specifically:

- Proximity to the towns of Amos, Val d'Or and Rouyn-Noranda obviates the need for camp-based accommodation at the mine site. The distance to these towns is such that workers will have a relatively short commute.
- The mining cost per tonne mined is expected to be low given factors that include the massive nature of mineralization (allows for use of large rope shovels), high production rates and very low abrasion index for 75% of rock that will be mined. Coupled with the low strip ratio of approximately 1.1:1, this will result in a low cost per tonne of ore.
- The processing cost is expected to be low due to the low cost of power in Quebec (currently less than 5₵/kWh), large scale of operation and low abrasion of all ore that will be treated.

The key benchmark that was referenced during the generation of the Dumont site operating cost estimate of $9.18/t ore (life of project average) was Taskeo's Gibraltar open pit mine located in BC. Despite a much higher stripping ratio of approximately 4:1, Gibraltar reported site costs of $9.10/t in 2011 and $10.72/t in 2012. Note that 2012 costs were adversely impacted by a scaling up of output in advance of the completion of a mill expansion. Attention is particularly directed to the mining costs at Gibraltar, which are reported to be $1.03/t mined in 2011 and

$1.33/t in 2012; the 2012 cost being approximately 20% lower than the $1.68/t life of mine average forecast for Dumont.

24.2 Opportunities

24.2.1 Trolley-Assisted Truck Haulage

24.2.1.1 Background

A typical diesel-electric haul truck utilizes a diesel engine to drive a traction alternator, which produces the electricity used to drive the wheel motors. The truck's control cabinet conditions the power, in terms of volts and amps, so the motors will provide the desired speed and torque - much as a transmission would do in a mechanical drive truck. The speed of the vehicle on grade is limited by the horsepower output of the diesel engine. With trolley assist, two pantographs are mounted to the front of the truck so that electric power can be collected from overhead lines. The lines are supported by rigid poles, and electricity is fed to the line by a sub station. Additional control devices are added to the truck, so that power from the overhead lines can be properly applied to the wheel motors (Figure 24.6).

When on trolley, a truck's diesel engine and alternator are not used for propulsion. The engine's speed automatically drops to an idle, with all the power for propulsion coming from the overhead lines. The speed of a diesel-powered truck is limited by its engine horsepower, but the speed of a trolley truck is limited by the capabilities of its traction motors (and inverters, on AC drives).

Figure 24.6: Trolley-Assist at Palabora

Source: RNC.

Savings realized from trolley assist can be categorized as follows:

- Energy cost savings – which occur as power is supplied to wheelmotors from an overhead line (and thus from the electrical grid) rather than being generated using the on-board diesel engine. The value of savings is a function of the kilometers traveled on trolley and the relative prices for fuel and electricity.
- Productivity Savings – which result from the increased speed of haul trucks traveling uphill on trolley, with improvements of almost 100% being possible. This allows the mine plan to be achieved with fewer trucks and an associated reduction in labour.
- Reduced maintenance costs – the maintenance interval for diesel engines can best be modelled as a function of fuel consumption. With the lower consumption rate for a truck traveling on trolley, the interval between overhauls / replacements can be extended.

In addition to the cost benefits listed above, trolley assist also has significantly environmental benefits, resulting from the reduction in particulate matter and greenhouse gases associated with generating energy from hydro-carbons.

The savings associated with trolley-assist are partially offset by costs associated with operating the system that include:

- Fixed infrastructure – including the trolley line, pole and substation.
- Truck infrastructure – including the pantograph and associated on-board control devices.
- Ongoing maintenance of fixed and truck-based infrastructure.
- Wider ramps –to accommodate trolley-assist infrastructure (primarily the sub stations), the width of equipped ramps would be increased by 5 m. This could result in flatter overall slopes and increased waste stripping.

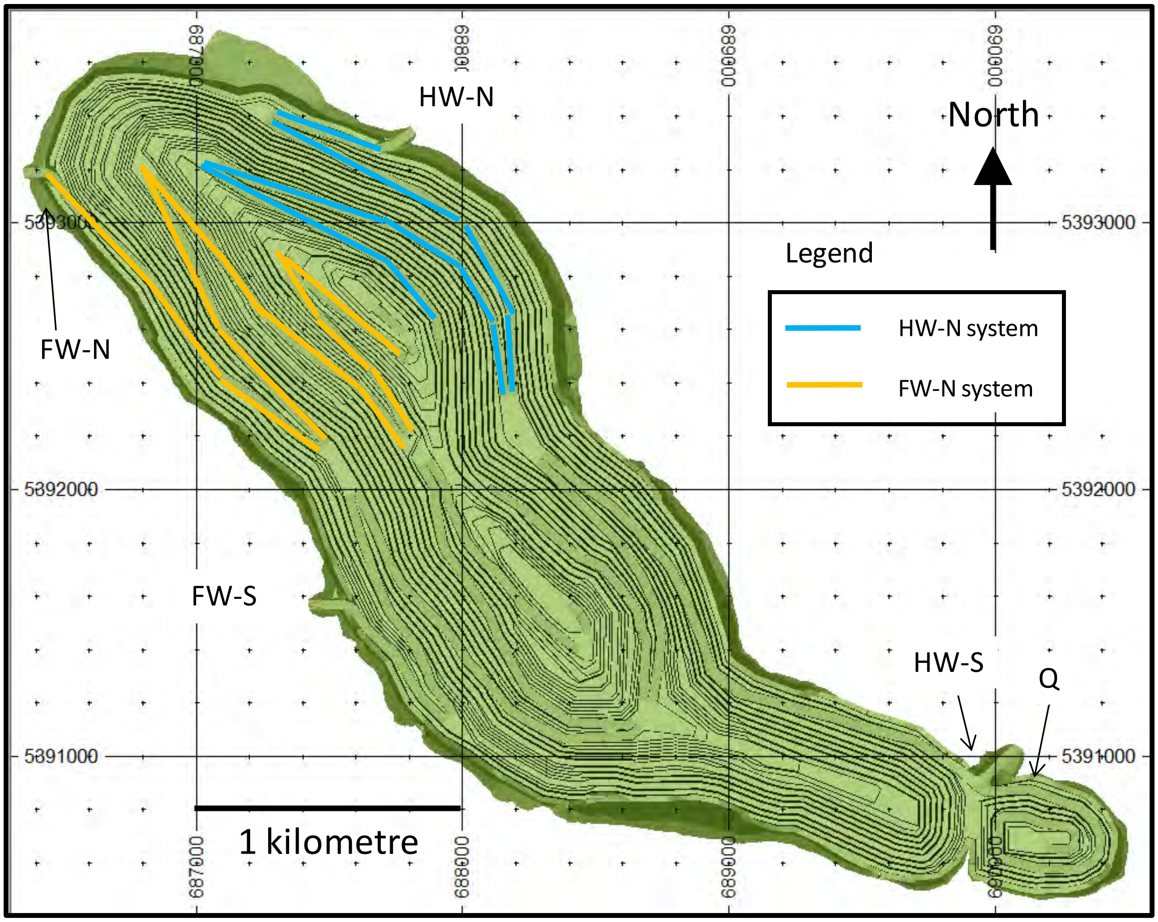
24.2.1.2 Application of Trolley-Assist at Dumont

The base case mine design for Dumont was determined by iteration and the selected design maximized overall project NPV for the base case design criteria. As design criteria for the trolley assist option are materially different than the base case (e.g., the mining cost ≥ 10% lower, wider ramps necessitate increased stripping), the optimal design for trolley assist will likely be different than the current base case. A trolley operation would aim to maximize the tonnage travelled on trolley-equipped ramps, which could impact the overall ramp geometry and sequencing of LG stages (see Section 16, Figure 16.6 previously). There is sufficient flexibility inherent in the current Dumont design to accept changes that would optimize a trolley operation.

As the current base case is not optimized for a trolley operation, a detailed feasibility level discounted cash flow analysis of the trolley option has not been performed. Rather, a high level assessment was conducted to determine the potential benefits of the system.

This assessment showed the base case design included sufficient traffic on the hanging wall – north (HW – N) and footwall – north (FW – N) ramp systems to justify installation of trolley. The hanging wall – south (HW – S), footwall – south (FW – S) and southeast pit (Q) systems did not have sufficient traffic to justify trolley. See Figure 24.7 for an illustration of the ramp systems at the end of mine life.

Figure 24.7: Dumont Ramp Configuration



Source: RNC.

For reasons that include the overall depth of hauls and density of traffic, trolley would not be implemented before yr4 of mill operation (2020). From this date onwards, approximately 80% of material mined exits from the HW-N and FW-N ramp systems. There is also sufficient traffic on the ramp accessing the various lifts of waste rock dump 1 (WR1) and low-grade ore 1 stockpile (LGO1) to justify an installation. After taking account of the expected trolley line utilization, approximately 60% of total uphill hauls over the life of mine could be transported using the system.

This level of trolley utilization would result in diesel savings for the production haul truck fleet of approximately 350 million litres, or approximately 35% of base case consumption. The diesel consumption would be replaced by approximately 1.8 GWh of power, or approximately 0.7 kWh/t mined. For the assumed prices of diesel and electricity that were used in the base case evaluation, energy savings would total approximately $250 million over the 16 years of expit operation that trolley-assist would be operated. Energy savings would represent approximately 80% of the gross operating cost savings from trolley, or 90% of the net savings after the application of system maintenance costs.

Offsetting these operating cost savings are the capital costs associated with a trolley system. The trolley system envisaged would reach a maximum installed length of approximately 12 km.

The system would be relocated when ramps were pushed back, with each of the two ramp systems considered for installation (HW-N and FW-N) being pushed back once. A total of 21 km of line would be installed during the life of mine, with 12 km being equipped with new infrastructure and the remaining 9 km with infrastructure that was being re-used (i.e., relocated when a ramp is pushed back). The cost of installing new infrastructure would total approximately $4 M/km, while the cost of re-installing infrastructure would be approximately 75% less. It is also possible that widening of the HW-N and FW-N ramps by 5m each would flatten overall slopes, though the incremental waste that would be associated with such flattening has not been quantified. Finally, implementation of trolley assist would require modifications to haul trucks, including installation of the pantograph and associated wiring and controls that would total approximately $0.5 M per truck.

The single most important issue that must be addressed in order for trolley to be successfully implemented at Dumont is the impact of weather – particularly the spring freshet – on road conditions. Uneven road surfaces will cause the pantograph to lose contact with the line, with the truck being rejected from line (and thus reducing system utilization) and possibly inflicting damage on the system through arcing. This will be addressed through the planned use of roadstone to continuously resurface all haul roads. It should be noted that trolley-assist has successfully been used in similar climates, including the Labrador Trough (at Lac Jeaninne – the world’s first trolley assist site) and Nevada (at Barrick’s Goldstrike, where annual snowfall is approximately 50% that at Dumont). One benefit of trolley-assist is that the enforced discipline of maintaining roads results in improved haulage cost performance, such as in speeds achieved and tire life.

Trolley-assist remains a significant potential opportunity for Dumont. Developments will continue to be monitored prior to a final decision on implementation – which under the current schedule would likely be in 2019.

24.2.2 Magnetite Byproduct

24.2.2.1 Introduction

Prefeasibility testwork assays indicated that there are significant quantities of magnetite in the tailings of the awaruite circuit. As a result, RNC requested that Ausenco complete a conceptual study to investigate the flowsheet amendments required and potential economic benefits of implementing a magnetite separation circuit. Some of the testwork undertaken also investigated the process requirements to produce a saleable magnetite product.

24.2.2.2 Laboratory Testwork

Dumont ore grades approximately 4% magnetite. Preliminary testing was performed to evaluate whether this could be upgraded to a saleable product. Most of the magnetite in the feed reports to the magnetic concentrate and is then sent to tails stream after sulphide and awaruite recovery.

A sample of the magnetic concentrate from the 2011 mini-plant campaigns was reground to 40 µm and underwent a series of sequentially decreasing magnetic separation stages to determine the optimum gauss to produce a high-grade magnetite concentrate.

Table 24-7 shows the results from 5 discrete samples from different areas of the deposit.

Table 24-7: Magnetite Concentrate Testwork Summary

	Magnetite Ore	Wt to Mag Con	Wt to 1000 Gauss Conc	Wt to Fe Conc	Fe Conc	Magnetite Recovery	Fe Grade
Outcrop Fe-T6	6.2	29.6%	29.6%	20.3%	1.8%	28.8%	66.6
218DF Fe-T4	5.8	29.3%	29.7%	26.7%	2.3%	40.3%	61.0
A-Comp T1	5.9	35.5%	35.5%	30.7%	3.9%	65.8%	63.7
S-Comp T3	5.6	26.8%	35.9%	28.5%	2.7%	48.6%	65.6
M-Comp T3	4.5	30.2%	30.3%	23.3%	2.1%	47.3%	59.1
				Average	**2.6%**	**46.2%**	**63.2**

24.2.2.3 Design Criteria

The major design criteria outlined in the concept study are primarily composed of recent magnetite testwork data and testwork data and design criteria from the PFS.

Table 24-8 shows the assumed feed grades to the plant and feed grades of the streams feeding the proposed magnetite circuit. The magnetite circuit as referred to in this report is inclusive of the regrind mill and the awaruite circuit because recirculating streams around the magnetic separators have an affect on those areas.

Table 24-8: Estimated Magnetite Circuit Feed Grades

Stream	% Fe	% Ni
Plant Feed (design)	5.71	0.35
LIMS Magnetic Concentrate	25.0	0.30
Sulphide 1st Cleaner Tailings	5.00	0.19
Combined New Feed to Circuit	10.4	0.22

Note: Both iron and nickel grades are based on samples selected for the magnetic separation testwork.

Key data include feed compositions, product compositions and overall recoveries. Table 24-9 depicts these criteria.

Table 24-9: Overall Design Criteria

Parameter	Units	
Design Metal Grades		
Plant Feed (design)	% Fe	5.71
	% Ni	0.37
Combined New Feed to Circuit	% Fe	10.4
	% Ni	0.22
Concentrate Grade – Magnetic Sulphide Scavenger	% Fe	12.0
	% Ni	2.10
Concentrate Grade - Magnetite	% Fe	65.0
	% Ni	0.10
Concentrate Grade – Awaruite	% Fe	9.51
	% Ni	25.0
Design Metal Recoveries (on Magnetic Circuit Feed)		
Magnetic Sulphide Scavenger	% Fe	6.01
	% Ni	32.7
Magnetite	% Fe	85.0
	% Ni	6.11
Awaruite	% Fe	0.29
	% Ni	33.0
Concentrate Production		
Magnetite Concentrate	t/h	54.9
Awaruite	t/h	1.18

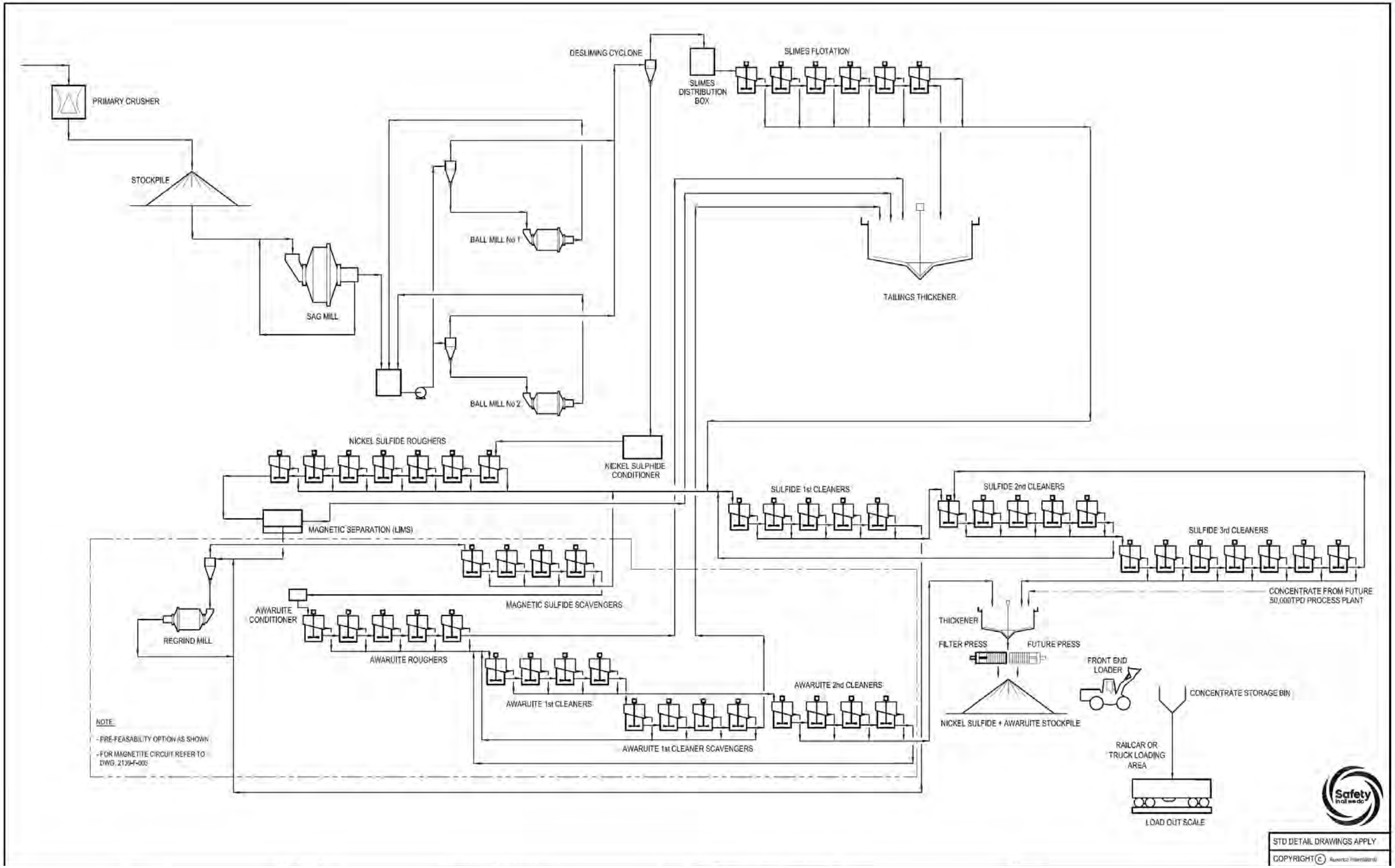
Note: The mass balance shown in this table is for 50 kt/d.

24.2.2.4 Flowsheet

The proposed magnetite separation circuit includes a four-stage cleaner separation; 1st stage non-magnetics report to the tailings thickener and the non-magnetics of the following stages are recirculated to the regrind mill. Additionally, the awaruite 1st cleaner scavenger concentrate reports to the regrind mill as opposed to the head of the awaruite 1st cleaner as it did in the original PFS flowsheet. These flowsheet changes (as well as a finer regrind target grind size) will increase the power requirement of the regrind mill.

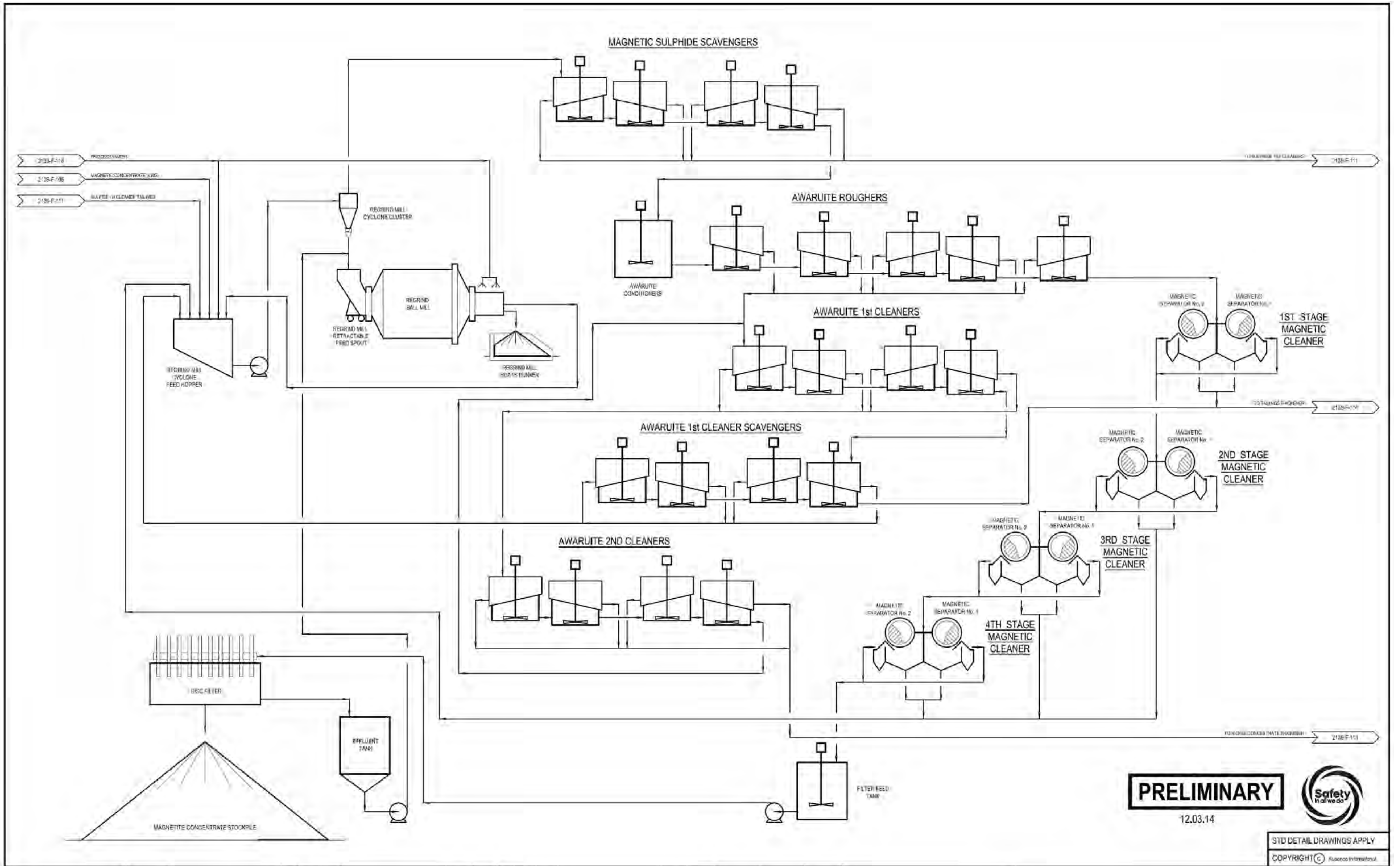
Figure 24.8 below is the overall flowsheet from the PFS. The streams and equipment affected by the addition of a magnetite separation circuit are boxed with a dotted line. The flowsheet shown in Figure 24.9 represents the replacement to the contents of the dotted box in Figure 24.8.

Figure 24.8: PFS Flowsheet Area Affected by Magnetic Separation



Ausenco

Figure 24.9: Potential Flowsheet Amendment



24.2.2.5 Capital Costs (2012)

The conceptual capital cost estimate (C$M, ±40%, 2012) has been prepared using pricing obtained during the PFS and was derived by factoring equipment costs and benchmarking against other Ausenco in-house data. Indirect costs were factored based on total mechanical equipment costs using in-house data and previous Ausenco experience with similar projects. A summary of the capital cost estimate is displayed in Table 24-10.

Table 24-10: Summary of Capital Costs ($M)

Cost Element	$M		
	Yr. 1 - 5	Expansion	Total
Directs			
Magnetite plant	22.8	22.8	45.6
On-site infrastructure	7.4	7.4	14.8
Total Directs (NET)	**30.2**	**30.2**	**60.4**
Indirects			
Capital spares and first fills	0.9	0.9	1.8
Freight	2.4	2.4	4.8
Start-up and Commissioning	0.3	0.3	0.6
Temporary facilities, cranage etc.	1.5	1.5	3.0
Owners costs	1.5	1.5	3.0
EPCM + Fee	5.4	5.4	10.8
Total Indirects (NET)	**12.0**	**12.0**	**24.0**
Escalation to Q1 2012	0.0	0.0	0.0
Contingency (40%)	12.1	12.1	24.2
Total Cost Estimate (NET)	**54.3**	**54.3**	**108.6**

24.2.2.6 Operating Costs (2012)

The conceptual operating costs estimate (C$/t, ±40%, 2012) for the magnetite circuit has been calculated by difference from the PFS mill estimate. Table 24-11 displays the net affect on the mill operating costs both before and after the planned mill expansion.

Table 24-11: Summary of Operating Costs ($/t)

	Net Increase in Plant Operating Costs ($/t)	
	Magnetic Separation Year 1 -5	Magnetic Separation Expansion
Labour	0.06	0.04
Power	0.05	0.05
Maintenance	0.05	0.05
Reagents and Consumables	0.04	0.04
Miscellaneous	0.03	0.02
Total Operating Directs	**0.23**	**0.20**
Total Transportation	**2.20**	**2.20**
TOTAL	**2.43**	**2.40**

24.2.3 Ferronickel Process Option

24.2.3.1 Testwork Summary

A 1 kg sample of concentrate, generated from the 2010-2011 locked cycle tests, underwent laboratory scale testing smelting tests. The concentrate grade was approximately 31% Ni. The laboratory testwork was completed by Kingston Process Metallurgy.

The concentrate was first roasted to remove the sulphur and then underwent a reduction process to make ferronickel.

The tests produced a high-grade ferronickel between 55-61% Ni. The main impurities are copper and cobalt.

Figure 24.10: RNC Ferronickel Buttons

Source: RNC.

24.2.3.2 Marketing Impact

The concentrate that RNC anticipates will be produced from Dumont is ideally suited for utilizing this type of roasting and reduction process as the nickel and iron content of the Dumont concentrate is expected to be relatively high, grades of other metals such as copper and cobalt are expected to be relatively low, and impurity levels for elements such as phosphorus, arsenic, and other impurities are expected to be very low.

The target RNC ferronickel specification is shown in Table 24-12.

Table 24-12: Target RNC Ferronickel Specification

Element	Target (%)
Nickel	55-60
Iron	Balance
Cobalt	<0.4
Copper	<0.7
Carbon	<0.2
Phosphorus	<0.01
Sulphur	<0.15
Silicon	<0.2

This alternate downstream processing option has the potential to yield significant economic benefits for the Dumont project versus conventional smelting and refining by increasing the potential value realized to 98-99% from 92-93% of nickel contained in concentrate due to higher recoveries, lowering costs due to a simpler, cheaper processing path, and earning a product premium due to production of a final ferronickel product.

RNC also believes that this alternate downstream processing option also significantly increases third party processing options for the concentrate anticipated to be produced from Dumont as there are a large number of electric furnaces already operating in China, Korea, and, Taiwan.

24.2.3.3 Memorandum of Understanding

In March 2013, RNC announced it had entered into a memorandum of understanding ("MOU") with Tsingshan Holding Group Co., Ltd. The MOU sets out the objectives of the two companies to work together in relation to downstream concentrate processing and the potential to enter into an offtake and/or partnership arrangement with respect to the Dumont project. Tsingshan is the second largest Chinese stainless steel company and one of the leading innovators in the development of vertically integrated NPI and stainless steel production operations. After working in cooperation with RNC for more than a year, Tsingshan completed its own analysis and testwork on a sulphide nickel concentrate (utilizing a process similar to the one tested by RNC) in its integrated NPI/stainless steel production facilities and plans to make the necessary investment in plant and equipment once concentrate feed is secured. This innovation represents the first time that nickel sulphide concentrate would have been used directly to create stainless steel. This contemplated plant is also expected to be capable of handling nickel sulphide concentrate like that anticipated to be produced from Dumont.

25 INTERPRETATION & CONCLUSIONS

The following conclusions arise from the information provided in the previous sections:

- The Dumont deposit represents a significant ore reserve that remains open at depth and along strike to the northwest.
- Reserves are reported at a cut-off grade of 0.15% nickel inside an engineered pit design based on a LG optimized pit shell that was generated using a nickel price of US$5.58/lb, which is 62% of the long-term forecast of US$9.00/lb, average metallurgical recovery of 43%, marginal processing and G&A costs of US$6.30/t milled, long-term exchange rate of C$1.00 equal US$0.90, overall pit slopes of 42° to 50° depending on the sector and a production rate of 105 kt/d. Mineral reserves include mining losses of 0.28% and dilution of 0.49% that will be incurred at the bedrock overburden interface, which corresponds to mining losses of 1 m and 2 m of dilution along this contact.
- It has been demonstrated that the deposit can be economically developed using large-scale open pit methods.
- This scope of design is estimated to require an initial capital investment of $1,268 M, an expansion capital investment of $997 M and sustaining capital of $823 M.
- Over the 33-year project life, Dumont is expected to produce 2,774 Mlbs of payable nickel and the equivalent of a further 150 Mlbs payable nickel in by-product cobalt and PGE. The average cost to produce nickel over the entire life is $4.79/lb, and includes lower costs of $4.44/lb in the initial five years of production.
- Based on a long-term Ni price of US$9.00/lb and C$ exchange rate of US$0.90, the after-tax NPV8% for the project is $1.3 billion while the after tax IRR is 16%. There is consequently justification for approving construction of the project.
- A key element of the mine plan is the accelerated release of ore relative to the requirements of the mill. The open pit mine is thus completed after 20 years, compared to the 33-year life of project. The costs associated with stockpiling 606 Mt lower value ore are more than offset by the elimination of risk that the mill will be undersupplied with ore from the mine, the favourable Ni production profile and ability to impound 43% of tailings in the mined-out pit.
- The design is optimal for the alternatives that were considered during this phase of work. Nonetheless, there are likely further opportunities to enhance the design, including a steepening of slope angles, acceleration of access to higher value ore and increased electrification of the pit.
- The mine plan is achievable but should not be considered conservative. Good systems and practices will need to be implemented at an early stage to meet the plan. Mine plan optimisation is heavily dependent on sinking rate in order to follow down dip the highest revenue ore. Multiple pushbacks are planned with up to three stages being mined at one time. Top notch mine planning will be required along with high productivities to achieve the planned sinking rates and tonnages. The rock conditions are favourable and water pumping is not expected to be onerous. The mine benefits from multiple ramp access design and long strike lengths of mining faces. It is expected and planned that reduced productivities and higher costs will be experienced on the top levels while mining through the overburden and establishing the upper benches in rock. Opportunities to improve results over the FS plan lie in achieving higher productivities, lower costs and adjusting the sequence to follow the better ore as geological and metallurgical knowledge is gained. There is essentially no risk of the plant not having sufficient feed as the mine capacity far exceeds the mill. The high mine

capacity allows the mine to send high value ore to the plant while stockpiling the lower grade material. Therefore in essence, mine plan optimisation revolves around the time value of money and moving metal production (through treating ores with higher grade and recovery) forward in time.

- A staged development approach has been adopted to mitigate technical and financial risk during the initial years of operation. The processing plant will initially be comprised of a single line with a nameplate throughput of 52.5 kt/d. The plant will be expanded to two lines with a nameplate throughput of 105 kt/d after 54 months.
- The groundwater regime is not expected to negatively impact the open pit design based on the hydrogeology work carried out to date. Groundwater inflows to the open pit are expected to average 5,000 m^3/d.
- Groundwater drawdown at the Launay Esker is expected to be minimal. Preliminary modelling using the PEA pit estimates drawdown at approximately 0.1 m at the end of pit operations. The draw down effect of the pit will then reduce as it is partially refilled with tailings.
- The Dumont sill and immediate hanging and footwall are characterized as a relatively strong anisotropic (sill parallel) rock mass, punctuated by oblique and parallel to sub-parallel fault damage-zones.
- The bearing capacity of surficial deposits and subsurface conditions at key development sites, such as the plant site, tailings deposition area, and waste dump area have been considered from a geotechnical perspective for the envisioned project development.
- Environmental geochemistry characterization of tailings, waste and ore indicate that these materials will be non-acid-generating due to their low sulphur content and high neutralization potential. Static tests indicate that waste rock, tailings and ore are leachable under the conditions of the tests, but more site-condition representative laboratory and field tests suggest that mine wastes will leach low levels of rock-derived constituents.
- The testwork proved that the Dumont material could be processed in a conventional wet grinding circuit followed by hydrocyclone desliming, nickel flotation and magnetic recovery. The cleaning circuit is a multiple stage circuit with a regrind on the magnetic concentrate and cleaner tails.
- The Dumont mineralization increases in hardness as the particle size decreases which is typical for many deposits. The average hardness results for 102 samples are as follows: Axb 54, BWi 21, RWi 15, CWi 14, and Ai 0.009.
- The rougher recovery equations were divided into four categories based on Hz/Pn ratio and degree of serpentinization. LOM Ni recovery averages 43% at a head grade of 0.27% Ni.
- Flotation testwork indicates that nickel recovery is relatively insensitive to grind sizes (P_{80}) up to about 150 µm.
- The locked cycle tests showed a large range of cleaner recoveries based on the grade and weight recovery of the rougher concentrate and the level of nickel in silicates in the sample.
- Both rougher and cleaner nickel recovery is driven by the sulphur assay in the feed or the ratio of S/Ni in the feed.
- The most effective unit operation for improving flotation performance is an aggressive desliming stage to remove the fine particles that cause viscosity problems in the rougher stage.
- The life of mine average concentrate grade is 29%.
- Cobalt recovery was estimated at 42%.

- The main impurity in the concentrate is MgO, which ranges between 3% and 13%. Other impurities, such as As, Pb, Cl, and P, were all near or below detection limits in the measured concentrate samples.
- A trade-off study was conducted to compare the costs of transporting nickel concentrate by truck and by rail. It was decided to include a rail spur as the assumptions in the economic analysis transport 50% of the Dumont nickel concentrate overseas from a port in Quebec. The most economic option was a truck-rail combination, where concentrate is trucked to an existing transfer facility in Rouyn-Noranda for furtherance by rail to Sudbury for the remainder of the concentrate.
- In order to limit environmental impact to one drainage basin, RNC has elected to contain project infrastructure within the Villemontel-St. Lawrence drainage basin. Consequently, the Chicobi River watershed will not be impacted by the project. Both watersheds, however, were covered in the environmental baseline studies.
- Current project definition is sufficient to provide a basis upon which most anticipated social and environmental impacts can be identified and assessed through the environmental and social impact study. Principal impacts anticipated at this stage relate to air quality, wetlands, fish habitat, water resources, and the social environment.
- The major project risks, as demonstrated by the financial analysis, are those parameters related to revenue, specifically nickel recovery, percentage payables and selling price for nickel. Project returns are also sensitive to the USD/CAD exchange rate.
- The project is less sensitive to other risks, including capital and operating costs. Returns are relatively insensitive to the cost of individual consumable items, such as power, oil and acid.
- Political, labour, location, environmental, social, and permitting risks are generally commensurate to those experienced by other mining projects in the Abitibi region of the province of Quebec and are considered low by global standards.

26 RECOMMENDATIONS

Recommendations for future work are listed below.

- Complete detailed design that considers the following points:
  - Evaluate opportunities for pit optimization, including:
    - Alternative mining sequences that may allow access to higher value ore to be accelerated and/or deferral of waste stripping.
    - Evaluate alternative ramp locations in the pit stages taking advantage of changes in wall slopes.
  - Re-evaluate use of trolley assisted truck haulage as an option based on fuel and electricity market rates.
  - Begin detailed engineering in Q3 2013 to procure long lead equipment in order to maintain the Q3 2016 plant operational date.
  - Undertake detailed geotechnical evaluations of the early rock exposures, throughout the open pit areas, to assess the reliability of structural and geotechnical models. Optimize design based on field performance of pit slopes in the various geotechnical domains.
  - Continue to evaluate pore pressures within the pit slope areas to verify the assumption that these will have a limited impact on slope stability.
  - Conduct further geotechnical investigations in order to complete detailed engineering design of all surface infrastructure, including the plant site and related facilities, rail lines, TSF Cell 1, the low-grade ore stockpile within the pit limits, and water management features that have a significant earthworks component to them and are required within the first two years of operation.
  - Implement a metallurgy testwork program that will include:
    - Trade-off study to evaluate removal of slimes circuit
    - Reagent optimization testwork
    - Concentrate Thickening and Filtration testwork
    - Slimes cyclone pilot scale testing for detailed engineering design
    - Awaruite recovery circuit optimization
    - Recovery opportunities from scavenger non-magnetic stream
    - Complete testwork to quantify grindability characteristics of regrind mill feed
- Specific high voltage power studies as recommended for confirmation of high voltage supply by Hydro Quebec.
- Continue mining lease process.
- Initiate surface lease process.
- Continue environmental baseline studies.
- Continue environmental permitting process.
- Continue to investigate the natural cementation of tailings and waste fines and its impact on reducing the potential for these project components to act as dust sources.

- Continue stakeholder consultation during detailed engineering as well as during mine operations to minimize and/or mitigate the impact of the project and foster acceptance. Define the structure of stakeholder committees that will be created during mine construction and operations.
- Continue to assess the carbon sequestration potential of spontaneous mineral carbonation of tailings and waste rock on an operational basis and its impact on the carbon footprint of the project.

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CERTIFICATE OF AUTHOR

Leonard Paul Staples

To accompany the report entitled, 'Technical Report on the Dumont Ni Project, Launay and Trecesson Townships, Quebec, Canada' prepared for Royal Nickel Corporation (RNC) and dated July 25, 2013 (the 'Technical Report'), I, L. Paul Staples, P.Eng., do hereby certify that:

1. I am General Manager, Technical Solutions, Ausenco Services Pty Ltd., 44 St. Georges Terrace, Perth, Western Australia, 6000, Australia.
2. I graduated with a BSc. degree in Materials and Metallurgical Engineering from Queens University in 1993.
3. I am a registered Professional Engineer of New Brunswick (membership number 4832); as well, I am a member in good standing of the Canadian Institute of Mining Metallurgy and Petroleum and the Australian Institute of Mining and Metallurgy (MAusIMM).
4. I have worked as a Metallurgist continuously since my graduation from University. For the past 6 years I have been employed with Ausenco. During this period I have fulfilled roles as Principal Process Engineer, Manager Development, GM Canada, Director Technical Solutions Americas and am currently employed as General Manager Technical Solutions for the Asia Pacific and Africa region.
5. I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a 'qualified person' for the purpose of NI 43-101.
6. I am responsible for Sections 1, 2, 3, 4, 5, 6, 17, 18 (except 18.6, 18.7 and 18.16), 19, 21 (except 21.3.1, 21.3.3 and 21.5.3), 23, 24.1.3, 24.1.5, 25, 26 and 27 of the Technical Report.
7. I visited the property on May 19, 2011 and August 8, 2012.
8. Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of RNC, or any associated or affiliated entities.
9. Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of RNC, or any associated or affiliated companies.
10. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from RNC, or any associated or affiliated companies.
11. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
12. I have previous involvement with the project as QP for the technical report of December 16, 2011 (PFS) and the revised technical report of June 22, 2012 (revised PFS).
13. I have read NI 43-101 and Form 43-101F1, and confirm that the Technical Report has been prepared in compliance with that instrument and form.
14. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25th day of July, 2013.

Leonard Paul Staples, P.Eng.

REGISTERED PROFESSIONAL ENGINEER
Province of New Brunswick
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Jeffery Marc Bowen

To accompany the report entitled, "Technical Report on the Dumont Ni Project, Launay and Trecesson Townships, Quebec, Canada" prepared for Royal Nickel Corporation (RNC) and dated July 25, 2013 (the "Technical Report"), I, Jeffery Bowen, MAusIMM (CP), do hereby certify that:

1. I am Manager Studies for Ausenco Solutions Canada Inc. 555 boul. René-Lévesque Ouest, Bureau 200, Montreal, QC, H2Z 1B1, Canada.
2. I graduated with a BSc. (HON) degree in Metallurgy from the University of Ballarat, in 1998.
3. I am a Chartered Professional with the AusIMM.
4. I have worked as a Metallurgist continuously since my graduation from University. For the past 6 years I have been employed with Ausenco Minerals and Metals. During this period I have fulfilled roles as Senior Process Engineer, Study Manager and am currently employed as the Manager Studies in Montreal.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purpose of NI 43-101.
6. I am responsible for Sections 13, 24.2.2 and 24.2.3 of the Technical Report.
7. I visited the property on August 8, 2012.
8. Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of RNC, or any associated or affiliated entities.
9. Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of RNC, or any associated or affiliated companies.
10. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from RNC, or any associated or affiliated companies.
11. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
12. I have had no prior involvement with the property that is the subject of the Technical Report.
13. I have read NI 43-101 and Form 43-101F1, and confirm that the Technical Report has been prepared in compliance with that instrument and form.
14. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25th day of July, 2013

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Sébastien Bernard Bernier

To accompany the report entitled, "Technical Report on the Dumont Ni Project, Launay and Trecesson Townships, Quebec, Canada" prepared for Royal Nickel Corporation (RNC) and dated July 25, 2013 (the "Technical Report"), I, Sébastien Bernier, PGeo, do hereby certify that:

1. I am a Principal Consultant (Resource Geology) with the firm of SRK Consulting (Canada) Inc. ("SRK") with an office at Suite 101, Regent Street South, Sudbury, Ontario, Canada;

2. I am a graduate of the University of Ottawa in 2001 with B.Sc. (Honours) Geology and I obtained M.Sc. Geology from Laurentian University in 2003. I have practiced my profession continuously since 2002. I worked in exploration and commercial production of base and precious metals mainly in Canada. I have been focussing my career on geostatistical studies, geological modelling and resource modelling of base and precious metals since 2004;

3. I am a Professional Geoscientist registered with the Ordre des Géologues du Québec (OGQ #1034), the Association of Professional Geoscientist of Ontario (APGO #1847) and Professional Engineers and Geoscientists of Newfoundland and Labrador (PEGNL #05958);

4. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a recognized professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of National Instrument 43-101;

5. I have personally inspected the subjected property and surrounding areas from April 27 to May 2, 2011, and on May 17, 2013;

6. I have prepared the mineral resource statement for the Dumont Ni Project as documented in section 12 and 14 of this technical report. I have also contributed to sections 2, 25 and 26 and have also supervised the compilation of sections 7, 8, 9 10 and 11 that were compiled by Alger St-Jean, Vice-President of Exploration for Royal Nickel Corporation. I accept professional responsibility for those parts of the technical report;

7. I, as a qualified person, am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;

8. Prior to this technical report, I was involved on an internal drilling density optimization study in 2010 of the Dumont Ni Project. I was also responsible for the mineral resource estimation and assay quality control at the Dumont Project between June 2011 and December 2011 included in a technical report filled on December 16, 2011 and in the revised technical report of June 22, 2012. I undertook these three assignments as an employee of SRK Consulting (Canada) Inc., under contract to Royal Nickel Corporation;

9. I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;

10. SRK Consulting (Canada) Inc. was retained by Royal Nickel Corporation to contribute to a technical report for the Dumont Ni Project, compiled by Ausenco Limited, in accordance with National Instrument 43-101 and Form 43-101F1 guidelines. The preceding report is based on a site visit, a review of project files and discussions with Royal Nickel Corporation personnel;

11. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Dumont Ni Project or securities of Royal Nickel Corporation;

12. That, as of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 25th day of July, 2013

Sébastien B. Bernier, PGeo

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CERTIFICATE OF AUTHOR

David A. Warren

To accompany the report entitled, "Technical Report on the Dumont Ni Project, Launay and Trecesson Townships, Quebec, Canada" prepared for Royal Nickel Corporation (RNC) and dated July 25, 2013 (the "Technical Report"), I, David A. Warren, Eng., do hereby certify that:

1. I am a Senior Principal Consultant, Mining, for Snowden Mining Industry Consultants Inc. (Snowden), Suite 600, 1090 West Pender Street, Vancouver, B.C., V6E 2N7.
2. I graduated with a B.A.Sc. degree in Mineral Engineering from University of British Columbia 1978 and a M.Sc. degree in Technology (Mining) from Helsinki University of Technology (HUT) in 1997.
3. I am a registered Engineer of l'Ordre des ingénieurs du Québec (membership number 121481); as well, I am a member in good standing of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM).
4. I have worked as a mining engineer since my graduation from University. For the past 2 years I have been employed with Snowden and prior to that at mine sites for 15 years. During this period I have fulfilled roles as Principal Consultant (mining consultant) and Chief Mine Engineer (mine sites).
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purpose of NI 43-101.
6. I am responsible for Sections 15, 16.3, 16.4, 21.3.1 (mining operating costs), 21.5.3 (mining capital costs) and 24.2.1 of the Technical Report.
7. I visited the property on August 8, 2012.
8. Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of RNC, or any associated or affiliated entities.
9. Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of RNC, or any associated or affiliated companies.
10. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from RNC, or any associated or affiliated companies.
11. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
12. I have had no prior involvement with the property that is the subject of the Technical Report.
13. I have read NI 43-101 and Form 43-101F1, and confirm that the Technical Report has been prepared in compliance with that instrument and form.
14. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25th day of July, 2013

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Cameron C. Scott, P.Eng.

To accompany the report entitled, "Technical Report on the Dumont Ni Project, Launay and Trécesson Townships, Quebec, Canada" prepared for Royal Nickel Corporation (RNC) and dated July 25, 2013 (the "Technical Report"), I, Cameron C. Scott, P.Eng., do hereby certify that:

1. I, Cameron C. Scott, am a Professional Engineer, employed as a Principal Consultant with SRK Consulting (Canada) Inc.
2. I graduated with a B.A.Sc. Degree in Geological Engineering from the University of British Columbia in 1974 and subsequently was granted an M.Eng. Degree in Civil Engineering (Geotechnical Option) by the University of Alberta in 1984.
3. I have been a registered member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#11523) since 1978.
4. I have worked as a Geotechnical Engineer for a total of 39 years. Most of my professional practice has focused on the geotechnical and hydrogeological aspects of mining, including the site selection, design, permitting, operation and closure of mine waste facilities in Canada, the US, Mexico, Central and South America, Europe and various countries within the former Soviet Union.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101) and certify that by reason of education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purpose of this NI 43-101.
6. I am the responsible QP for Chapters 16.2.4, 18.6, 18.7, 21.3.3, 24.1.1, 24.1.2 and 24.1.4 of this technical report.
7. I visited the Dumont Project site in 2011 on February 2, May 19 & June 21, & in 2012 on July 13 & August 8.
8. Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of RNC, or any associated or affiliated entities.
9. Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of RNC, or any associated or affiliated companies.
10. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from RNC, or any associated or affiliated companies.
11. I am independent of RNC as independence is described by Section 1.5 of NI 43-101.
12. I was responsible for the majority of the geotechnical investigations of the overburden completed at the Dumont Project site between December 2010 and May 2012. I directed this work as an employee of SRK Consulting (Canada) Inc., under contract to RNC and Ausenco Solutions Canada Inc. In addition, I was involved in the technical report of December 16, 2011 and the revised technical report of June 22, 2012.
13. I have read National Instrument 43-101 and Form 43-101F1, and confirm that the Technical Report has been prepared in compliance with that instrument and form.
14. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

PROFESSIONAL
PROVINCE OF
CAMERON C. SCOTT
BRITISH COLUMBIA
ENGINEER

Cameron C. Scott, P.Eng.

Dated: July 25, 2013

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CERTIFICATE OF AUTHOR
John F. Duncan, P.Eng.

To accompany the report entitled, "Technical Report on the Dumont Ni Project, Launay and Trécesson Townships, Quebec, Canada" prepared for Royal Nickel Corporation (RNC) and dated July 25, 2013 (the "Technical Report"), I, John F. Duncan, P.Eng., do hereby certify that:

1. I, John F. Duncan, am a Professional Engineer, employed as a Principal Consultant with SRK Consulting (Canada) Inc.
2. I am a graduate of the University of Manitoba in 1986, 1992 and 1998; where I obtained a Bachelor of Science, Bachelor of Science in Civil Engineering and a Master of Engineering.
3. I am a Professional Engineer registered with the Professional Engineers Ontario (#100024399), the Association of Professional Engineers and Geoscientists of Manitoba (#8330), and the Association of Professional Engineers and Geoscientists of British Columbia (#38749).
4. I have practiced my profession continuously since 1992. I am a hydrotechnical engineer with 20 years' experience in the water resources industry. I lead Vancouver's SRK hydrotechnical team and am tasked with overall quality regarding hydrology and hydraulic analyses performed for office clientele. I have extensive experience developing overall water management strategies to reduce mine-contact water and optimizing surface water demand sources for mine process water.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101) and certify that by reason of education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purpose of this NI 43-101.
6. I am the responsible QP for Chapters 16.1 and 18.16 of this technical report.
7. I visited the Dumont Project site in 2012 on April 10 and 11, and August 8.
8. Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of RNC, or any associated or affiliated entities.
9. Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of RNC, or any associated or affiliated companies.
10. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from RNC, or any associated or affiliated companies.
11. I am independent of RNC as independence is described by Section 1.5 of NI 43-101.
12. I was responsible the surface water management, including hydrology, hydraulics and water balance. I undertook this work as an employee of SRK Consulting (Canada) Inc., under contract to Ausenco and RNC.
13. I have read National Instrument 43-101 and Form 43-101F1, and confirm that the Technical Report has been prepared in compliance with that instrument and form.
14. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

LICENSED PROFESSIONAL ENGINEER
07/25/2013
J. F. DUNCAN
JFD
PROVINCE OF ONTARIO

John F. Duncan, P.Eng.

Dated: July 25, 2013

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CERTIFICATE OF AUTHOR

Bruce Andrew Murphy

To accompany the report entitled, "Technical Report on the Dumont Ni Project, Launay and Trecesson Townships, Quebec, Canada" prepared for Royal Nickel Corporation (RNC) and dated July 25, 2013 (the "Technical Report"), I, Bruce Murphy, FSAIMM, do hereby certify that:

1. I am a Principal Consultant, Mining Rock Mechanics SRK Consulting (Canada) Inc, 1066 West Hastings Street, Vancouver, BC V6E 3X2 Canada.
2. I graduated with a MSc. Eng (Mining) degree from the University Witwatersrand, in May 1996.
3. I am a Fellow of the South African Institute of Mining and Metallurgy and as a result of my experience and qualifications, I am a Qualified Person ("QP") as defined in National Instrument 43-101 Standards of Disclosure of Mineral Projects (NI 43-101).
4. I have been involved in mining since 1990 and have practised my profession continuously since then. I have been involved in mining operations, mining related rock mechanics and consulting covering a wide range of mineral commodities in Africa, South America North America and Asia.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101) and certify that by reason of education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purpose of this NI 43-101.
6. I am responsible for Section 16.2.1, 16.2.2 and 16.2.3 of the "Technical Report on the Dumont Ni Project, Launay and Trecesson Townships, Quebec, Canada", effective date of dated July 25, 2013.
7. I visited the property on the June 17 and 18, 2011.
8. Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of RNC, or any associated or affiliated entities.
9. Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of RNC, or any associated or affiliated companies.
10. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from RNC, or any associated or affiliated companies.
11. I am independent of RNC as independence is described by Section 1.5 of NI 43-101.
12. I was responsible for the geotechnical investigation of the bedrock at the Dumont Project open pit area between December 2010 and May 2012. I undertook this work as an employee of SRK Consulting (Canada) Inc., under contract to RNC. In addition, I was involved in the technical report of December 16, 2011 and the revised technical report of June 22, 2012.
13. I have read National Instrument 43-101 and Form 43-101F1, and confirm that the Technical Report has been prepared in compliance with that instrument and form.
14. As of the date of this certificate, to the best of my knowledge, information and belief, this technical report contains all the scientific and technical information that is required to be disclosed to make this technical report not misleading.

Bruce Murphy, FSAIMM

Dated: July 25, 2013

CERTIFICATE OF AUTHOR

Valérie Johanne Bertrand

To accompany the report entitled, "Technical Report on the Dumont Ni Project, Launay and Trecesson Townships, Quebec, Canada" prepared for Royal Nickel Corporation (RNC) and dated July 25, 2013 (the "Technical Report"), I, Valérie J. Bertrand, géo., do hereby certify that:

1. I am an Associate and Senior Geochemist employed at Golder Associates Ltd. located at 32 Steacie Drive, Kanata Ontario. K2K 2A9, Canada.
2. Graduated with a Bachelor of Science degree in Geology from the University of Ottawa in Ottawa, Ontario in 1991 and have a Master of Applied Science degree in Mining Engineering from the University of British Columbia in Vancouver, B.C. which I obtained in 1999.
3. I am a registered Professional Geoscientist in Ontario (membership number 1458) and a member in good standing of l'Ordre des Géologues du Québec (membership number 1221).
4. I have worked as a geoscientist since my graduation from the University of Ottawa. For the past 14 years I have been employed with Golder Associates Limited. During this period I have fulfilled the role of geochemist on mining projects directing and completing environmental geochemistry investigations on mine wastes, soils and water. I currently hold the position of Associate, Senior Geochemist.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purpose of NI 43-101.
6. I am responsible for Sections 20.7.1, 20.7.2, 20.7.5 and the Environmental Issues portion of Table 18.1 in Section 18.7.2, and those portions of the summary, conclusions and recommendations that are based on those sections of the Technical Report.
7. I visited the property on November 15 and 16, 2010 for the purpose of the Phase 2 environmental geochemistry testing program. During this visit, I discussed the geology of the deposit with RNC geologists, viewed selected samples of each rock type selected for environmental geochemistry analysis and supervised the sampling procedure employed by RNC for the environmental geochemistry testing program.
8. Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of RNC, or any associated or affiliated entities.
9. Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of RNC, or any associated or affiliated companies.
10. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from RNC, or any associated or affiliated companies.
11. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
12. I have been involved in baseline studies for the project and have authored sections of the Technical Report of December 16, 2011 (PFS) and the revised Technical Report of June 22, 2012 (the revised PFS).
13. As of the date of this certificate, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25th day of July, 2013

GÉOLOGUE
VALÉRIE J.
BERTRAND
1221
QUÉBEC

Valérie J. Bertrand, géo. M.A.Sc.
Associate, Senior Geochemist

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CERTIFICATE OF AUTHOR

Kevin Scott

To accompany the report entitled, "Technical Report on the Dumont Ni Project, Launay and Trecesson Townships, Quebec, Canada" prepared for Royal Nickel Corporation (RNC) and dated July 25, 2013 (the "Technical Report"). I, Kevin Scott, P. Eng., do hereby certify that:

1. I am Manager, Process and Studies for Ausenco Solutions Canada Inc. 855 Homer Street, Vancouver, BC V6B 2W2, Canada.

2. I am a graduate of University of British Columbia, Vancouver, Canada in 1989 with a Bachelor of Applied Science degree in Metals and Materials Engineering.

3. I am registered as a Professional Engineer in the Province of British Columbia (Licence # 25314) and the Province of Ontario (License # 90443342).

4. I have worked as a Metallurgist continuously for a total of 23 years since my graduation from University. My relevant experience for the purpose of the Technical Report is:
   - Reviews and reports as a metallurgical consultant on a number of mining operations and projects for due diligence and financial monitoring requirements.
   - Process engineer at three Canadian base metals mineral processing operations.
   - Senior metallurgical engineer working for multi-national engineering and construction companies on feasibility studies and engineering design of mineral processing plants.
   - Senior process manager in charge of process design and engineering for a metallurgical processing plant in South America.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purpose of NI 43-101.

6. I am responsible for Chapter 22 of the Technical Report.

7. I have not visited the property.

8. Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of RNC, or any associated or affiliated entities.

9. Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of RNC, or any associated or affiliated companies.

10. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from RNC, or any associated or affiliated companies.

11. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

12. I have had no prior involvement with the property that is subject of the Technical Report

13. I have read NI 43-101 and Form 43-101F1, and confirm that the Technical Report has been prepared in compliance with that instrument and form.

14. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25th day of July, 2013.

Kevin C. Scott, P. Eng.

PROFESSIONAL PROVINCE OF K.C. SCOTT 25314 BRITISH COLUMBIA ENGINEER

GENIVAR

CERTIFICATE OF AUTHOR

Simon Latulippe

To accompany the report entitled, "Technical Report on the Dumont Ni Project, Launay and Trecesson Townships, Quebec, Canada" prepared for Royal Nickel Corporation (RNC) and dated July 25, 2013 (the "Technical Report"), I, Simon Latulippe, Engineer, do hereby certify that:

1. I am currently employed as an Engineer and Project Manager by: GENIVAR Inc. 5355 des Gradins Blvd. Quebec, Quebec CANADA G2J 1C8.

2. I graduated with a Bachelor's degree in Geological Engineering from Laval University, Quebec, Canada in 1998.

3. I am a registered Professional Engineer of Quebec (membership number 121692).

4. I have worked as a geological engineer on a continuous basis for 15 years since my graduation from university, mainly providing services to the environment industry as a consultant. My relevant experience for the purpose of the Technical Report includes environmental studies as project manager, characterization and monitoring, soil rehabilitation and groundwater treatment, site tests and design, Mine site reclamation projects as water management and tailings leader, Mine Closure plans design, Mine projects permitting leader, Mine project water management and tailings design.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purpose of NI 43-101.

6. I am responsible for Sections 20 (except 20.7.1, 20.7.2 and 20.7.3), of the Technical Report.

7. I visited the property on July 13, 2013.

8. Neither I, nor any affiliated entity of mine, is at present, under an agreement, arrangement or understanding or expects to become, an insider, associate, affiliated entity or employee of RNC, or any associated or affiliated entities.

9. Neither I, nor any affiliated entity of mine, own, directly or indirectly, nor expect to receive, any interest in the properties or securities of RNC, or any associated or affiliated companies.

10. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from RNC, or any associated or affiliated companies.

11. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

12. I have had no prior involvement with the property that is the subject of the Technical Report.

13. I have read NI 43-101 and Form 43-101F1, and confirm that the Technical Report has been prepared in compliance with that instrument and form.

14. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25th day of July, 2013

INGÉNIEUR - ENGINEER
Simon Latulippe
121692
QUÉBEC

Simon Latulippe, Eng.