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Minerals & Metals

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Royal Nickel Corporation

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

11 July 2019

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Table of Contents

Ausenco

Ausenco

Tables

Figures

Ausenco

1 SUMMARY

1.1 Introduction

The Dumont Project will be an open pit mine/mill operation, using conventional drilling and blasting, with loading by a combination of hydraulic excavators and electric rope shovels into trucks ranging in size from 45 – 290 tonnes. The process plant will be constructed in two phases. Phase I will have an initial average throughput of 52.5 ktpd using a single SAG mill and two ball mills for grinding, desliming using cyclones, conventional flotation and magnetic separation, to produce a nickel concentrate also containing cobalt and PGEs. Phase II throughput will be doubled to 105 ktpd in Year 7 by mirroring the first line.

RNC Minerals (RNC) is a multi-asset mineral resource company headquartered in Toronto, Canada primarily focused on the development and production ramp-up of its Beta Hunt gold mine and the development of the large ultramafic Dumont Nickel-Cobalt Project 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. On April 20, 2017, RNC closed a joint venture transaction with Waterton Precious Metals Fund II Cayman, LP and Waterton Mining Parallel Fund Onshore Master, LP (collectively, "Waterton"). Under the terms of the transaction, Waterton acquired a 50% interest in the Dumont Project. RNC and Waterton formed the Dumont JV, a 50/50 nickel joint venture that owns the Dumont Nickel-Cobalt Project through Magneto Investments Limited Partnership (the Dumont JV). On July 23, 2018 RNC announced its interest in the Dumont JV would be reduced to approximately 28% as a result of the conversion by Waterton of its US$10 million RNC convertible note into additional units of the Dumont JV.

RNC manages the project on behalf of the Dumont JV. The mineral claims covering the Dumont deposit are currently held 98% by Magneto Investments Limited Partnership and 2% by Ressources Québec.

In September 2018, Ausenco Engineering Canada Inc. (Ausenco) was commissioned by RNC, in its capacity as Manager of the Dumont Joint Venture, to complete the feasibility study (FS) update 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 and geotechnical inputs, Wood PLC (Wood) was engaged to prepare tailings management, site water balance, geotechnical and closure planning inputs, David Penswick (Penswick) was retained for mine design, mine operating costs, mine capital costing, reserve estimation and financial evaluation. WSP Global Inc. (WSP) was engaged to provide inputs to the environmental and permitting aspects of the project. Golder Associates Ltd. (Golder) contributed to the hydrology, hydrogeology, and environmental geochemistry inputs.

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 12,823 (2016 Census) and is the seat of the Abitibi County Regional Municipality (Figure 1-1).

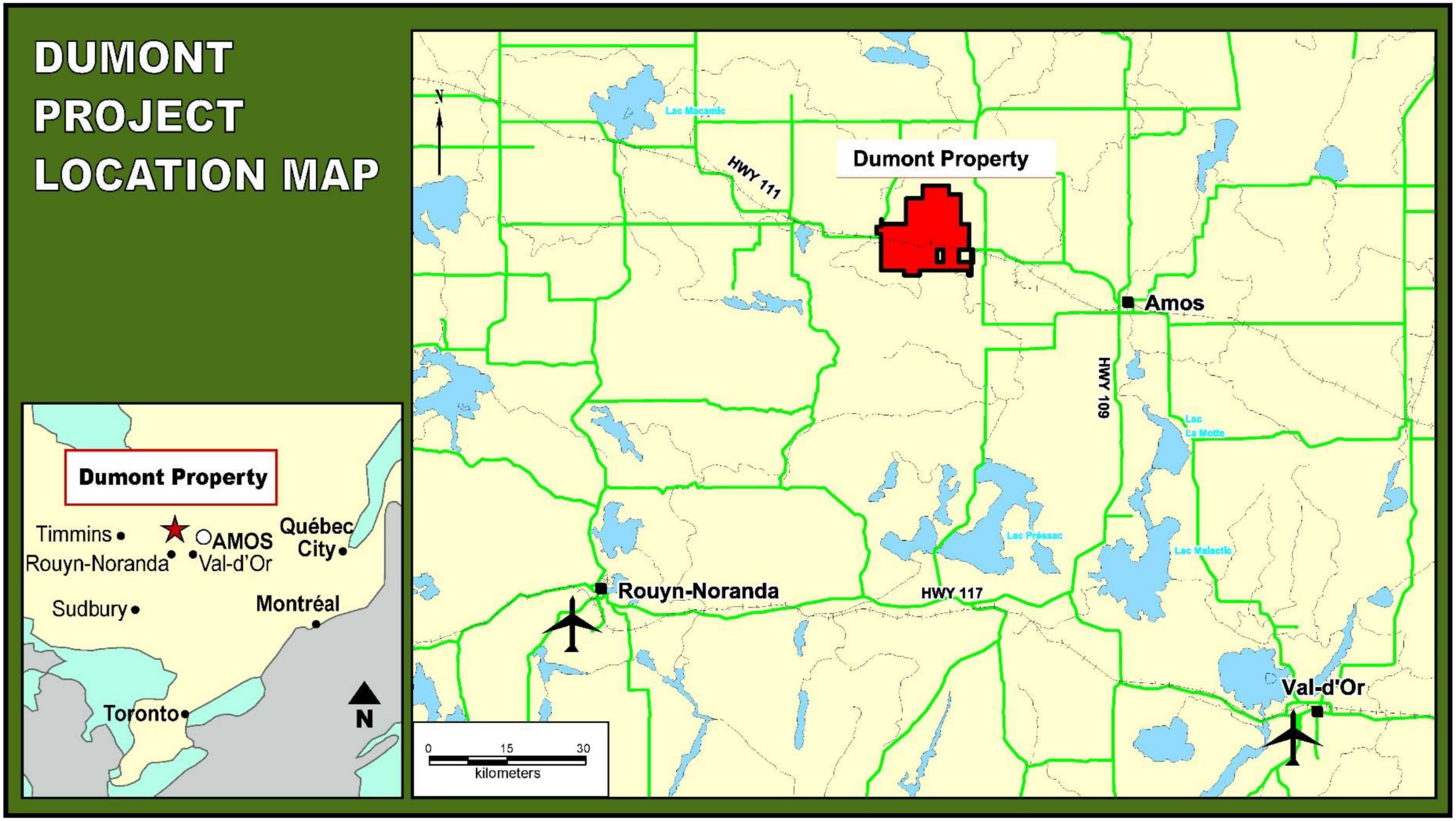
No historical mining or production has been conducted on the Dumont property. However, for the past 100 years, the Val d'Or - Rouyn-Noranda region surrounding the Dumont property has been and continues to be a prolific mining area.

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 focused 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 construction of the mineral resource model was a collaborative effort between RNC and SRK Consulting (Canada) Inc. 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). An update to the parameters of the block model definition was completed by Chelsey Protulipac, P.Geo (APGO #2608). Dr. Oy Leuangthong, P.Eng (APEGA#82746, PEO#90563867), assisted with the geostatistical analysis, variography, and the selection of resource estimation parameters. The effective date of the current resource estimate is May 30th, 2019. The mineral resource estimate considers drilling information available to 31 December 2012, as no new drilling information is available beyond that date 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: Dumont Nickel Project, Quebec, SRK Consulting (Canada) Inc., May 30th, 2019[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	1,050	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)			
Measured	-	-		-			
Indicated	1,114,300	4.27		47,580			
Measured + Indicated	**1,114,300**	**4.27**		**47,580**			
Inferred	832,000	4.02		33,430			

Notes: 1. *Reported at a cut-off grade of 0.15 percent nickel inside conceptual pit shells optimized using nickel price of US$7.50 per pound, average metallurgical and process recovery of 43 percent, processing and G&A costs of US$4.33 per tonne milled, exchange rate of C$1.00 equal US$0.77, overall pit slope of 42 degrees to 50 degrees depending on the sector, and a production rate of 105,000 tonnes per day. The qualified person considers that the conceptual pit shells would not be materially different to that if current (2019) conceptual pit optimization assumptions were considered. The technical parameters would be unchanged and with the metal price in Canadian dollars constant due to the decrease in US$ nickel price assumption compensated by corresponding decrease in US$:CAD$ exchange rate, the qualified person considers the reporting cut-off grade of 0.15 percent nickel to be reasonable. Values of cobalt, palladium, platinum and magnetite are not considered in the cut-off grade calculation as they are by-products 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* (May 30, 2019)[1]

Category	(kt)	Grades Ni (%)	Grades Co (ppm)	Grades Pd (g/t)	Grades Pt (g/t)	Contained Metal Ni (Mlb)	Contained Metal Co (Mlb)	Contained Metal Pd (koz)	Contained Metal Pt (koz)
Proven	163,140	0.33	114	0.031	0.013	1,174	41	162	67
Probable	864,908	0.26	106	0.017	0.008	4,908	202	466	220
Total	**1,028,048**	**0.27**	**107**	**0.019**	**0.009**	**6,082**	**243**	**627**	**287**

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$4.05 per pound, average metallurgical recovery of 43%, marginal processing and G&A costs of US$4.10 per tonne milled, long-term exchange rate of C$1.00 equal US$0.75, overall pit rock slopes of 40° to 50° depending on the sector, and a production rate of 105 kt/d. Mineral Reserves include mining losses of 0.33% and dilution of 0.43% that will be incurred at the contact between mineralization and waste. The life of mine stripping ratio is 1.02:1. 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. The mineral abundance model is coextensive and of identical dimensions to the element model.

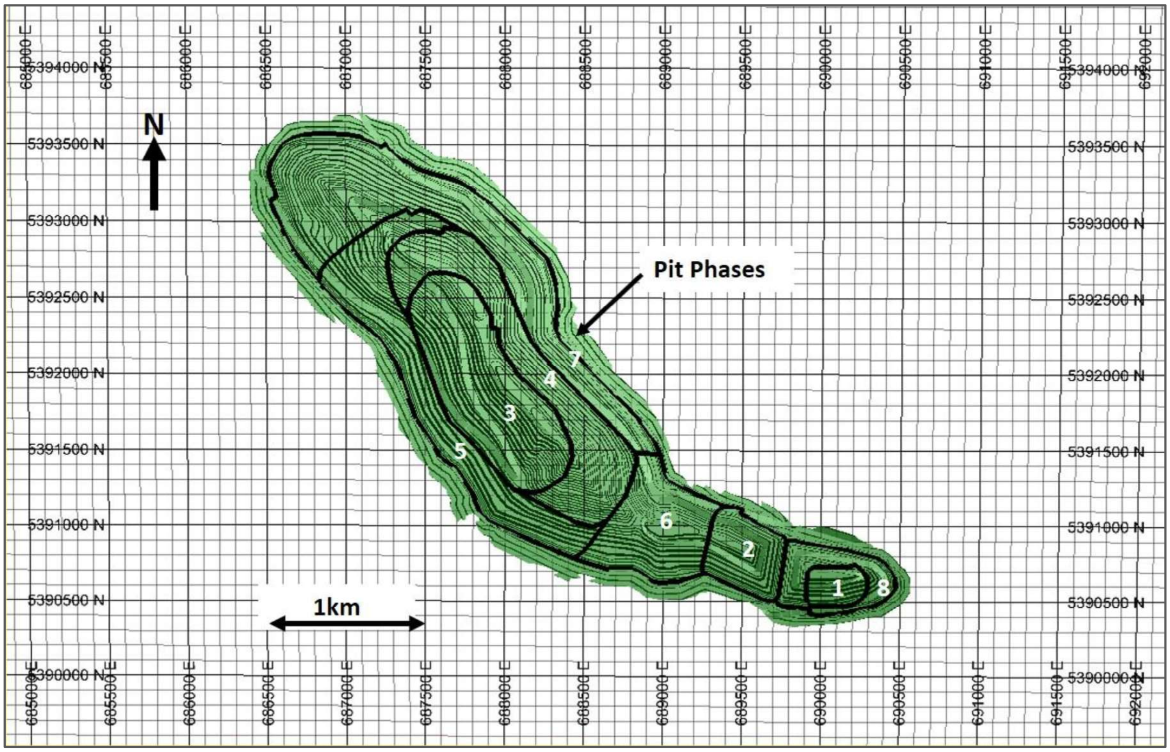
Reserves were estimated by Dave Penswick, P.Eng. These are based on the mineral resource block model described above. Reserves are contained within an engineered pit design that is based upon a Lerchs-Grossmann (LG) optimized pit shell generated using a nickel price of US$4.05/lb, which is considerably lower than the long-term forecast of US$7.75/lb. Reserves include dilution of 0.43% and mining losses of 0.33%.

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 sequence of mining phases is given in Figure 1-2, with a high-level summary of the overall mining sequence being as follows:

- Phase 1: The Starter Quarry, which targets the only outcrop and will provide waste rock for construction purposes along with ore to be stockpiled and used for commissioning the mill. The void created by mining of Phase 1 will also serve as a temporary reservoir to hold the start-up water requirements for the mill. Longer term, while the Main Pit (Phases 2 – 7) is in operation, the Quarry will also provide contingent surge storage capacity for the freshet and other periods of higher precipitation.
- Phase 2: Additional construction rock will be provided by Phase 2, which is located within the South East Extension (SEE) immediately west of the 'Saddle' separating it from the Quarry.
- Phase 3: This is the highest value portion of the entire pit and is targeted as soon as sufficient construction rock has been liberated from Phases 1 & 2.
- Phases 4 and 5: Are Main Pit pushbacks to the hanging wall and footwall.
- Phase 6: An extension to the final limits of the SEE
- Phase 7: The final phase of the Main Pit, extending to the west, hanging wall and at depth.
- Phase 8: Following completion of the Main Pit, tailings will be impounded inpit and there will no longer be a requirement for the contingent water storage within the Quarry. Phase 8 is an extension to the ultimate limits of the Quarry. A rock pillar will remain between this satellite pit and the SEE immediately adjacent.

Figure 1-2: Mining Phase Sequence



Source: RNC.

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 511 Mt will be loaded to the low-grade stockpiles. Of this, 112 Mt of the highest value stockpile material will be reclaimed during the initial 19 years that the main pit is active. The remaining 398 Mt will be reclaimed after completion of the Main Pit, extending the life of project to a total of 30 years and 3 months. For simplicity, the remainder of this document refers to project life as 30 years.

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 73 Mlbs Ni recovered to concentrate during the first initial period when the concentrator throughput is 52.5 kt/d. After throughput is increased to 105 kt/d, output increases to an average of 111 Mlbs recovered Ni while the Main Pit is active. Over the 30 year life of project, output averages 87 Mlbs.

The strategy of accelerated mining has the additional advantage of creating a void, which would accommodate approximately 42% of the tailings produced, thus reducing the surface footprint of operations.

The bench height at Dumont will increase progressively. At the outcrop / subcrop, the initial bench in rock will be mined on a nominal 5 m bench height. Blast holes measuring 115 mm will be drilled by diesel powered percussion drills. Below the initial bench and to the lowest level of the overburden – rock contact (a vertical window of 70 m), a 10 m bench height will be employed. Blast holes measuring 270 mm will be drilled using diesel powered rotary drills. Thereafter, a 15 m bench height will be used. Blast holes will measure 311 mm and be drilled using the same rotary drills as for the 10 m benches. All holes will be charged with emulsion. All final walls will be pre-split.

Approximately 71% of the total 2,080 Mt that will be excavated from the Dumont pit will be loaded using electric rope shovels (nominal dipper capacity 100t) into 290 t payload trucks. A further 22% of the expit total will be loaded using large, electrically powered hydraulic excavators (nominal dipper capacity 61 t) also into 290 t trucks. Smaller diesel-powered hydraulic excavators (nominal dipper capacity 30 t) will predominantly load dry overburden totalling 5% of the expit tonnage into 90 t trucks. The remaining 1% of material will be predominantly wet overburden and will be loaded by small backhoe excavators (nominal dipper capacities of 8 and 15 t) into 45 t articulated trucks.

From year 3 onwards, the 290 t haul trucks will be equipped with pantographs to utilize trolley-assist on the main ramps. The use of trolley-assist will result in faster cycle times and reduce diesel consumption by over 35%, or approximately 450 M litres as compared to the 2013 Feasibility Study configuration.

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.

All mining fleet will be purchased by the Owner. A local mining contractor with experience operating in similar environments has been pre-selected to assist during the pre-strip period, particularly with mining clay. Thereafter, all mining will be performed by the Owner.

The 2,080 Mt of material excavated from the pit will include 1,028 Mt ore, 879 Mt waste rock, 124 Mt overburden that is mainly sand and gravel, and 49 Mt clay. The Life-of-Mine stripping ratio is 1.02:1. Approximately 16% 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 52% 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.

Approximately 11% of clay will be used in construction of the TSF (as an impermeable membrane) or for reclamation activities. The remainder will be impounded within cells constructed using sand and gravel or waste rock and located on the hanging wall (northeast) side of the pit. Sand and gravel will be used for some construction activities, as well as reclamation of waste dumps. 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;
- electrical substations to feed the electrified equipment and trolley assist infrastructure;
- 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 labour complement averages 298 persons over the life of project, including 441 while the Main Pit is active and 110 during reclaim of the low-grade stockpile.

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 fell 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 samples 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.

Full circuit 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.

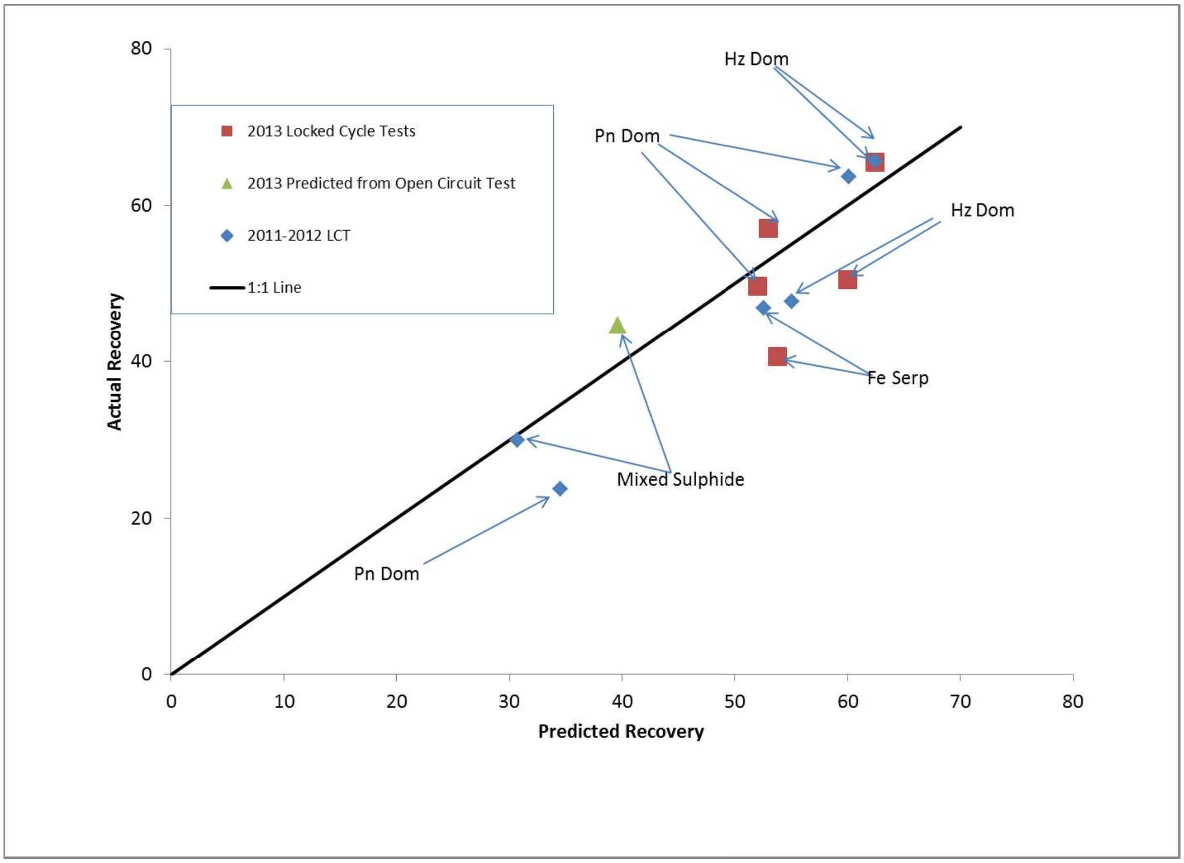
Overall, once the rougher and cleaner recovery equations were applied, the average nickel recovery over the life of the project is 43%.

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.

By-product credits for cobalt (Co), platinum (Pt) and palladium (Pd) were not included in the financial analysis, as the assumption was that all the concentrate is roasted and converted to ferronickel to feed the stainless steel industry. However, Co, Pt and Pd are still recovered to the concentrate and in the right metal-price environment could be payable if the concentrate (or a portion of the concentrate) were sent to a smelter for traditional smelting and refining. The cobalt recovery to concentrate is 33% over the life of the project. The calculated Pt + Pd grade in concentrate over the life of the project is 4.4 g/t, based on an average PGE recovery to concentrate of 62%.

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 34%. 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 a 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 for transport to third-party concentrate processing. Coarse and slime tailings will be thickened in dedicated thickeners prior to deposition in the TSF.

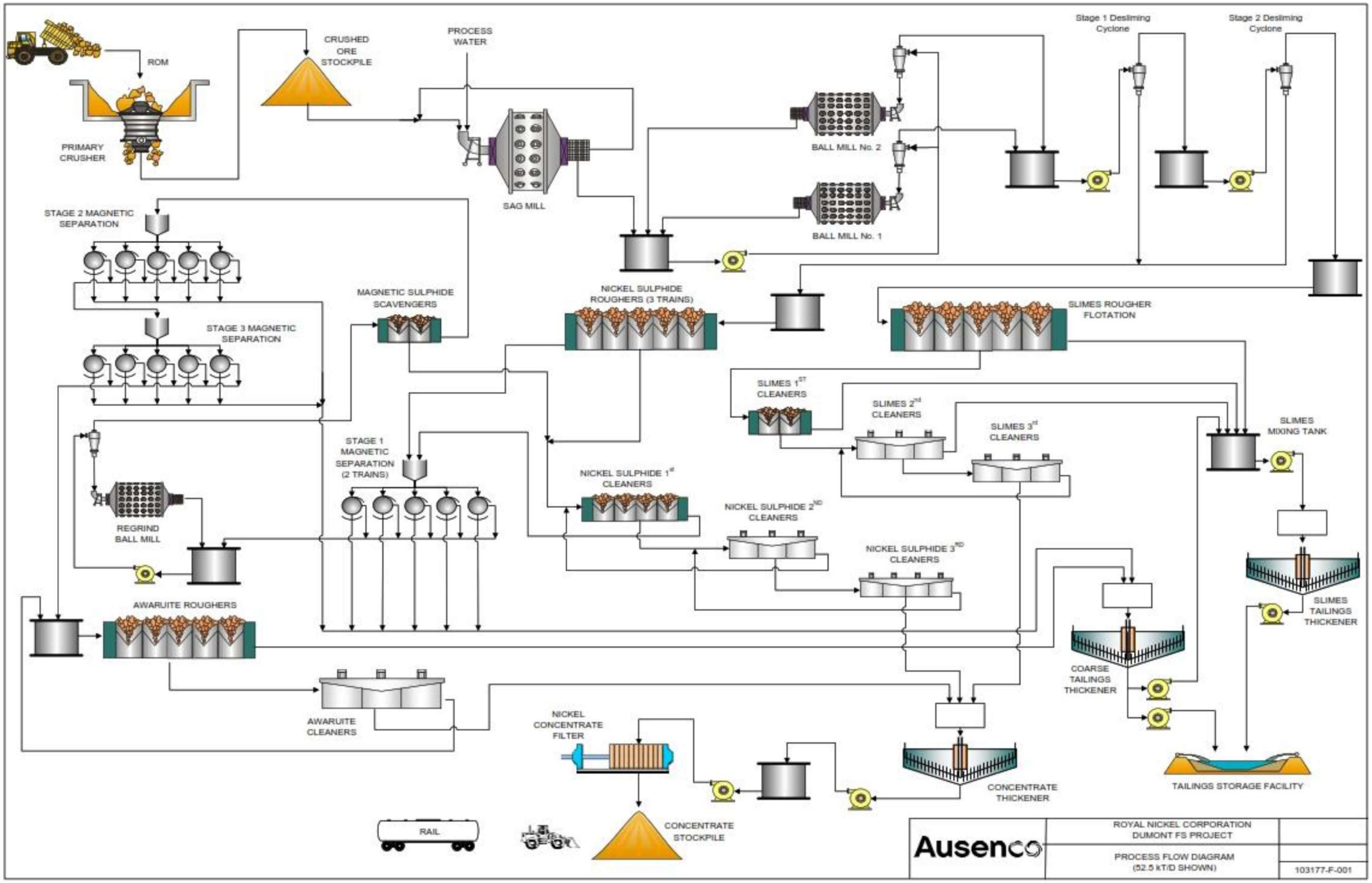
The process plant will be built in two phases. Initially, the plant will be designed to process 52.5 kt/d with allowances for a duplicate process expansion to increase plant capacity to 105 kt/d. Common facilities will include concentrate thickening and handling and sulphuric acid off-loading and containment.

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 flowsheet 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 run up to the process plant main 120 kV substation.
- Water – Water for start-up will be provided by surface water stored at the TSF and at the Quarry during the construction period. During operations, water demand will largely be met by recycling water from the TSF or the Pit (during the inpit tailings disposal phase). Make-up water and freshwater requirements will be provided by the Quarry or from the pit (during inpit tailings disposal phase). A water treatment plant will be available from the beginning of the operation to treat excess water from the TSF prior to its discharge to the Villemontel River.
- Gas – The use of propane gas is considered for heating buildings in this study, deliveries will be by tanker truck. For future supply considerations, an existing natural gas pipeline extends to within approximately 25 km 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 6 km. The rail spur consists of a fuel drop-off and pick up siding near the mining truck shop and the main track extends north of the process plant. A rail car drop-off and pickup siding is located north of the main security entrance, northwest of the water treatment plant for dropping off and picking up the rail cars used to deliver consumables for the mill and nickel concentrate. 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 is located approximately 400 m west of the process plant and consists of a tailings impoundment and a Recycle Water Basin (RWB). It is designed to store approximately 596 Mt of tailings over nineteen (19) years. Once mining at the open pit has ceased, stockpiled ore will be processed for approximately 12 years and those tailings, approximately 428 Mt, will be deposited in the open pit.

1.8 Environment and Permitting

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 twelve years. Biophysical data come mainly from three distinct fieldwork programs performed from 2007 to 2009, with some complementary information extracted from the baseline studies designed to support the Environmental and Social Impact Assessment in 2011 and 2012. RNC has hired consultants over the past 5 years to optimize the project and consequently, additional data were acquired from 2013 to 2018 by consultants or RNC. 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	2008	2009	2011	2012	2013	2014	2015	2016	2017	2018
Climate				√	√	√	√	√	√	√	√
Air quality							√	√	√	√	
Hydrology and bathymetric survey				√	√	√	√	√	√		
Water and sediments quality	√	√	√	√		√	√				
Groundwater quality				√	√	√					
Soil characterization					√	√					
Rare and protected plants	√			√							
Vegetation and wetlands		√		√			√				
Wildlife	√	√	√								
Small mammals				√							
Fish	√	√	√	√	√		√				
Benthic invertebrates	√	√	√								
Birds		√		√				√			
Reptiles and amphibians				√		√		√			
Ambient noise				√		√					
Infrastructures								√			
Archaeology		√				√					
Public and Stakeholders				√	√		√	√	√	√	√

Notes: 1. References are specified in the sections 20.1 to 20.4). **Source:** RNC.

These data and 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. Although, there are some sensitive elements in the surrounding footprint, the optimization work conducted on the mining plan and design significantly eliminate or reduce significantly the effect of the project on these components.

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 demonstrate 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 rock, overburden 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 force in 2013) in the long-term. The waste rock and tailings also demonstrate significant potential for permanent carbon sequestration through spontaneous mineral carbonation.

The Dumont Project received the Provincial Certificate of Authorization from the Quebec Ministry of Sustainable Development, Environment and the Fight Against Climate Change in July 2015 and received a positive Environmental Assessment Decision from the Federal Minister of the Environment in July 2015.

As part of the current study in 2018 and 2019, modifications were made to the 2013 Feasibility Study project design that was considered in the ESIA. An update of the environmental and social impacts evaluation was therefore carried out to consider these modifications. The negative impacts previously identified in the preliminary ESIA remain the same but the intensity of some of these impacts will be reduced. However, the negative impact reduction is not significant enough to result in a change in the impact importance evaluation when the impact evaluation methodology is applied. The environmental components where the project effects are reduced are air quality and noise (section 20.5).

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 roundtable; and

Information and consultation processes for the Abitibiwinni First Nation in Pikogan.

In May 2017, RNC and the local Algonquin First Nation Conseil de la Première nation Abitibiwinni (“PNA”) announced the signing of an Impact and Benefit Agreement (IBA) for the Dumont Nickel Project. RNC’s interest in the is agreement was assigned to the Dumont JV at the time of the joint venture transaction. Consequently, the parties to the IBA are PNA and the Dumont JV.

The IBA serves as a framework to govern the relationship with the PNA and lays out the commitments of the parties regarding the impacts and benefits of the Dumont Project. The parties to the IBA are the PNA and the RNC-Waterton nickel joint venture.

The IBA provides for meaningful PNA participation in the Dumont Project through training, employment, business opportunities, collaboration in environmental protection and other means.

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. Consultation activities will be planned to share the results of the updated feasibility study.

1.10 Capital Cost Estimate

All amounts are expressed in Canadian dollars (CAD) unless otherwise indicated.

Table 1-4 provides a summary of the capital costs estimate, including initial capital (phase 1), expansion capital (phase 2), and sustaining capital. Table 1-5 shows the total capital costs by area, excluding sustaining capital. The costs are expressed in real, Q1 2019 Canadian dollars and include all mining, site preparation, process plant, dams, sumps, first fills, buildings, and roadworks.

Items originally received in foreign currencies were converted into Canadian dollars. For USD currency, an exchange rate of 0.75 was used. For other currencies, published exchanged rates as of 2019-05-10 from “Oanda.com” were used.

The estimates are considered to have an overall accuracy ±15% for the FS portion 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

Description	Initial Capital (CAD $M)	Expansion Capital (CAD $M)	Sustaining Capital (CAD $M)	Total Capital (CAD $M)
Mine [3,4]	298	0	600	898
Process Plant [2,4]	461	447	64	971
Tailings	48	31	168	247
Infrastructure [2,4]	275	157	0	432
Indirect Costs[1]	164	95	-16	242
Contingency	111	71	0	182
Total	**1,357**	**801**	**814**	**2,973**

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

Table 1-5: Initial Capital Costs by Area – Not including Sustaining Capital

Area	Direct Costs	Initial Capital (CAD $ M)	Expansion Capital (CAD $ M)	Total Cost (CAD $ M)
01	Mining	298	0	298
02	Crushing	61	59	120
03	Process	400	388	788
04	Concentrate Loadout	0.3	0	0.3
05	Tailings	48	31	79
06	Utilities	180	133	312
07	Onsite Infrastructure	79	24	103
08	Off-site Infrastructure	16	1	17
Total Direct Costs		**1,082**	**635**	**1,717**
09	Indirect Costs	124	87	212
10	Owner's Costs	40	7	47
Total Indirect Costs		**164**	**95**	**259**
Total Direct & Indirect Costs		**1,246**	**730**	**1,976**
11	Escalation	Excluded		
11	Contingency	111	71	182
Total Project Costs		**1,357**	**801**	**2,158**

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 Yr1-7	105 kt/d Yr8-19	LOM Average
Mine	$/t ore milled	$7.11	$5.46	$3.82
Process	$/t ore	$5.31	$5.20	$5.20
G&A	$/t ore	$0.97	$0.53	$0.54
Site Costs	$/t ore	$13.39	$11.19	$9.56
Site Costs	US$/t ore	US$10.04	US$8.40	US$7.17
Site Costs	US$/lb	US$2.83	$3.14	$3.07
Realization	US$/lb	US$0.15	$0.16	$0.16
C1 Cash Cost[1]	US$/lb	US$2.98	$3.30	$3.22

Note: **1.** The Base Case design assumes roasting of concentrate, which will not produce payable by-product metals. An alternate case that considers treatment and refining with associated payable production of Co and PGEs is discussed in Section 24.

1.12 Economic Analysis

The Dumont Nickel project is expected to produce 2.6 billion pounds Ni recovered to concentrate over 30 years of operation. Table 1-7 summarizes key metrics for the Base Case design. The costs and returns for the FS assume a long-term nickel price of US$7.75/lb Ni and a Canadian dollar exchange rate of US$0.75. 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	1,028	1,028
Payable Ni	Mlbs	2,402	2,402
Gross Revenue	$/t ore	25.60	19.20
Realization [1]	$/t ore	1.94	1.45
Net Smelter Return	$/t ore	23.66	17.75
Site Operating Costs	$/t ore	9.56	7.17
C1 Costs [2]	$/lb Ni	4.30	3.22
EBITDA	$/t ore	13.23	9.92
Peak Funding Requirement [3]	$M	1,386	1,039
Total Investment [4]	$M	3,047	2,285
AISC [5]	$/lb Ni	5.07	3.80
Total Costs [6]	$/lb Ni	5.94	4.46
Pre-Tax $NPV_{8\%}$	$M	6,725	5,043
Post-Tax $NPV_{8\%}$	**$M**	**1,226**	**920**
Post-Tax IRR		**15.4%**	**15.4%**

Notes: **1.** Realization includes the cost of concentrate transport and implied costs of metal deductions, **2.** C1 Costs include Realization and Site Operating Expenditures, **3.** Peak Funding represents the cumulative unlevered investment prior to generation of positive cash flow, **4.** Total Investment includes all Capital and Closure expenses, **5.** All In Sustaining Costs include C1 Costs, Royalties, IBA, Sustaining Capital and Closure expenses, **6.** Total Costs include AISC, Initial Capital and Expansion Capital.

Key assumptions included in the Base Case evaluation include:

- The use of trolley-assisted truck haulage in the mine, but no use of autonomous equipment. The potential impact of autonomous equipment is discussed as an opportunity in Chapter 24.
- The process plant throughput will be 52.5 ktpd initially. A project to double capacity will start in Year 6 and process the first incremental ore 18 months later.
- All concentrate produced will be sold to third parties for roasting at a facility located outside of the province of Quebec. With roasting, no revenues would be realized from by-product cobalt or platinum group elements (PGE).
- The potential benefits from magnetite as a by-product have not been included but are discussed as a potential opportunity in Chapter 24.

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 approximately 25%. The project is less sensitive to costs, with a 10% change in total site operating costs having a 16% impact on NPV, while a 10% change in total capital has an 12% impact.

1.13 Project Implementation

Overall schedule duration from commencement of basic engineering (to order long-lead equipment) to the end of ore commissioning is 33 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	-11Q[1]
Commence Full EPCM	-10Q
Order Long Lead Equipment	-9Q
Construction Permit Approval	-8Q
Substantial Completion of Engineering	-7Q

Hydro Contract Power	-4Q
Start of Commissioning	-3Q
Mechanical Completion	-2Q
Reception of First Ore	-1Q
Plant Operational	0

1. Q = quarter, time 0 refers to the time plant is operational

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 baseline studies as required;
- Complete detailed design that considers the following points:
  - Using a smaller SMU size to reblock Measured Resources planned to be mined with smaller excavators. This could result in delivery of higher grade and/or recovery material to the plant in initial years of operation.
  - Begin detailed engineering upon additional financing and procure long lead equipment in order to maintain the schedule outlined in Section 1.13;
  - 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;
  - 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, 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;
- Specific high voltage power studies as recommended for confirmation of high voltage supply by Hydro Quebec;
- Continue mining lease process;
- Continue surface lease process;
- 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 Minerals (RNC) is a multi-asset mineral resource company headquartered in Toronto, Canada primarily focused on the development and production ramp-up of its Beta Hunt gold mine and the development of the large ultramafic Dumont Nickel-Cobalt Project 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. On April 20, 2017, RNC closed a joint venture transaction with Waterton Precious Metals Fund II Cayman, LP and Waterton Mining Parallel Fund Onshore Master, LP (collectively, "Waterton"). Under the terms of the transaction, Waterton acquired a 50% interest in the Dumont Project. RNC and Waterton formed the Dumont JV, a 50/50 nickel joint venture that owns the Dumont Nickel-Cobalt Project through Magneto Investments Limited Partnership (the Dumont JV). On July 23, 2018 RNC announced its interest in the Dumont JV would be reduced to approximately 28% as a result of the conversion by Waterton of its US$10 million RNC convertible note into additional units of the Dumont JV.

RNC manages the project on behalf of the Dumont JV. The mineral claims covering the Dumont deposit are currently held 98% by Magneto Investments Limited Partnership and 2% by Ressources Québec.

This technical report, prepared for RNC and dated June 21, 2019, 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.

In September 2018, Ausenco Engineering Canada Inc. (Ausenco) was commissioned by RNC, in its capacity as Manager of the Dumont Joint Venture, to complete the feasibility study (FS) update and the NI 43-101 compliant technical report on the project. This study 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 and geotechnical inputs, Wood PLC (Wood) was engaged to prepare tailings management, site water balance, geotechnical and closure planning inputs, David Penswick (Penswick) was retained for mine design, mine operating costs, mine capital costing, reserve estimation and financial evaluation. WSP Global Inc. (WSP) was engaged to provide inputs to the environmental and permitting aspects of the project. Golder Associates Ltd. (Golder) contributed to the hydrology, hydrogeology, and environmental geochemistry inputs.

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 information in relation to exploration, data, and detailed geology outside of this deposit in Section 10.6.

The Dumont Project will be an open pit mine/mill operation, using conventional drilling and blasting, with loading by a combination of hydraulic excavators and electric rope shovels into trucks ranging in size from 45 – 290 tonnes. The process plant will be constructed in two phases. Phase I will have an initial average throughput of 52.5 ktpd using a single SAG mill and two ball mills for grinding, desliming using cyclones, conventional flotation and magnetic separation, to produce a nickel concentrate also containing cobalt and PGEs. Phase II throughput will be doubled to 105 ktpd in Year 7 by mirroring the first line.

2.3 Qualified Persons

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

Table 2-1: Participants in the Dumont Feasibility Study

NI-43-101 Chapter		LEAD	Name of QP	Organization
1	**Summary**	Thomas Zwirz	Thomas Zwirz	AUSENCO
	1.1 Introduction	Alger St-Jean	Thomas Zwirz	AUSENCO
	1.2 Geology & Mineralization	Alger St-Jean	Chelsey Protulipac	SRK
	1.3 Resources & Reserves	Chelsey Protulipac	Chelsey Protulipac	SRK
	1.4 Mining	Dave Penswick	Dave Penswick	DP
	1.5 Metallurgy	Johnna Muinonen	Paul Staples	AUSENCO
	1.6 Mineral Processing	Johnna Muinonen	Paul Staples	AUSENCO
	1.7 Infrastructure	Thomas Zwirz	Thomas Zwirz	AUSENCO
	1.8 Environmental	Simon Latulippe	Simon Latulippe/ Valerie Bertrand	WSP/GOLDER
	1.9 Community	Simon Latulippe	Simon Latulippe/ Valerie Bertrand	WSP/GOLDER
	1.10 Capital Cost Estimate	Jean-Marc Lepine	Jean-Marc Lepine	AUSENCO
	1.11 Operating Cost Estimate	Genevieve Clayton	Paul Staples	AUSENCO
	1.12 Economic Analysis	Dave Penswick	Dave Penswick	DP
	1.13 Project Implementation	Thomas Zwirz	Thomas Zwirz	AUSENCO
	1.14 Conclusions & Recommendations	Thomas Zwirz	Thomas Zwirz	AUSENCO
2	**Introduction**	Alger St-Jean	Thomas Zwirz	AUSENCO
3	**Reliance on Experts**	Thomas Zwirz	Thomas Zwirz	AUSENCO
4	**Property Description & Location**	Alger St-Jean	Thomas Zwirz	AUSENCO
5	**Accessibility, Climate, Local Resources, Infrastructure and Physiography**	Alger St-Jean	Thomas Zwirz	AUSENCO
6	**History**	Alger St-Jean	Thomas Zwirz	AUSENCO
7	**Geological Setting**	Alger St-Jean	Chelsey Protulipac	SRK
8	**Deposit Types**	Alger St-Jean	Chelsey Protulipac	SRK
9	**Exploration**	Alger St-Jean	Chelsey Protulipac	SRK
10	**Drilling**	Alger St-Jean	Chelsey Protulipac	SRK
11	**Sample Preparation, Analyses and Security**	Alger St-Jean	Chelsey Protulipac	SRK
12	**Data Verification**	Chelsey Protulipac	Chelsey Protulipac	SRK
13	**Mineral Processing and Metallurgical Testing**	Johnna Muinonen	Paul Staples	AUSENCO
14	**Mineral Resource Estimates**	Chelsey Protulipac	Chelsey Protulipac	SRK
15	**Mineral Reserve Estimates**	Dave Penswick	Dave Penswick	DP
16	**Mining Methods**			
	16.1.1 Hydrology	Joao Paulo Lutti	Vu Tran	WOOD
	16.1.2 Hydrogeology	Michel Mailloux	Michel Mailloux	GOLDER

	16.2.1,16.2.2,16.2.3 Geotechnical Design Criteria – Rock	Bruce Murphy	Bruce Murphy	SRK
	16.2.4 Geotechnical Design Criteria – Soil	Cam Scott	Cam Scott	SRK
	16.3 Open Pit Mine Plan	Dave Penswick	Dave Penswick	DP
	16.4 Mining Process	Dave Penswick	Dave Penswick	DP
17	**Recovery Methods**	Genevieve Clayton	Paul Staples	AUSENCO
18	**Project Infrastructure**	Thomas Zwirz	Thomas Zwirz/ Dave Penswick/ Vu Tran /JP Lutti	AUSENCO/DP/WOOD
19	**Market Studies and Contracts**	Johnna Muinonen	Thomas Zwirz	AUSENCO
20	**Environmental Studies, Permitting and Community Impacts**	Simon Latulippe	Simon Latulippe/ Valerie Bertrand	WSP/GOLDER
21	**Capital and Operating Costs**	Jean-Marc Lepine	Jean-Marc Lepine	AUSENCO/DP/WOOD
22	**Economic Analysis**	Dave Penswick	Dave Penswick	DP
23	**Adjacent Properties**	Alger St-Jean	Thomas Zwirz	AUSENCO
24	**Other Relevant Data and Information**	Dave Penswick	Paul Staples	AUSENCO/DP
25	**Interpretations and Conclusions**	Thomas Zwirz	Thomas Zwirz	AUSENCO
26	**Recommendations**	Thomas Zwirz	Thomas Zwirz	AUSENCO
27	**References**	Thomas Zwirz	Thomas Zwirz	AUSENCO

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

- Paul Staples of Ausenco for mineral processing and metallurgy, plant and operating costs and alternative mill study coordination. Paul visited the property on May 19, 2011 and August 8, 2012
- Thomas Zwirz of Ausenco for infrastructure, plant capital costs and study coordination. Thomas did not visit the site.
- Chelsey Protulipac of SRK for the mineral resource estimation, data verification, geology and exploration contribution. Chelsey visited the property on October 23, 2018.
- David Penswick for reserve estimation, mining, mine capital, operating costs and financial evaluation. David most recently visited the property on November 8, 2019.
- Cam Scott of SRK for mine soil geotechnical, 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.
- Vu Tran of Wood for tailings storage facility design. Vu visited the property on October 23, 2018.
- Joao Paulo Lutti of Wood for site water balance and TSF seepage collection ditches and sumps. Joao Paulo visited the property on October 23, 2018.
- Michel Mailloux of Golder for Mine hydrogeology visited the site on June 3, 2019.
- Jean-Marc Lépine of Ausenco for Economic Analysis. Jean-Marc did not visit the site.
- 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.
- Simon Latulippe of WSP for Environmental Studies, Permitting and Social/Community Impact. Simon visited the property on July 13, 2013. No subsequent visits were done.

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
- Abrasion index .... Ai
- Annum (year) .... a
- Awaruite .... Aw
- Bond ball work index .... BWi
- Bond rod work index .... RWi
- Centimetre .... cm
- Concentration by weight .... Cw
- Crusher work index .... CWi
- Cubic centimetre .... cm^3
- Cubic metre .... m^3
- Cubic metres per day .... m^3/d
- 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 .... >
- Heazlewoodite .... Hz
- Hectare (10,000 m^2) .... ha
- Horsepower .... hp
- Hour .... h
- Hours per day .... h/d
- Hydro Quebec .... HQ
- Inch .... "
- Inverse distance .... ID
- JK drop weight test .... JK DWT
- 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
- Metric tonne (1,000 kg) .... t
- 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
- Standard Test Procedure .... STP
- 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 WSP 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 12,823 (2016 Census) and is the seat of the Abitibi County Regional Municipality Figure 4-1 overleaf).

RNC advises that the Dumont property consists of 235 contiguous mineral claims totalling 9,393 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

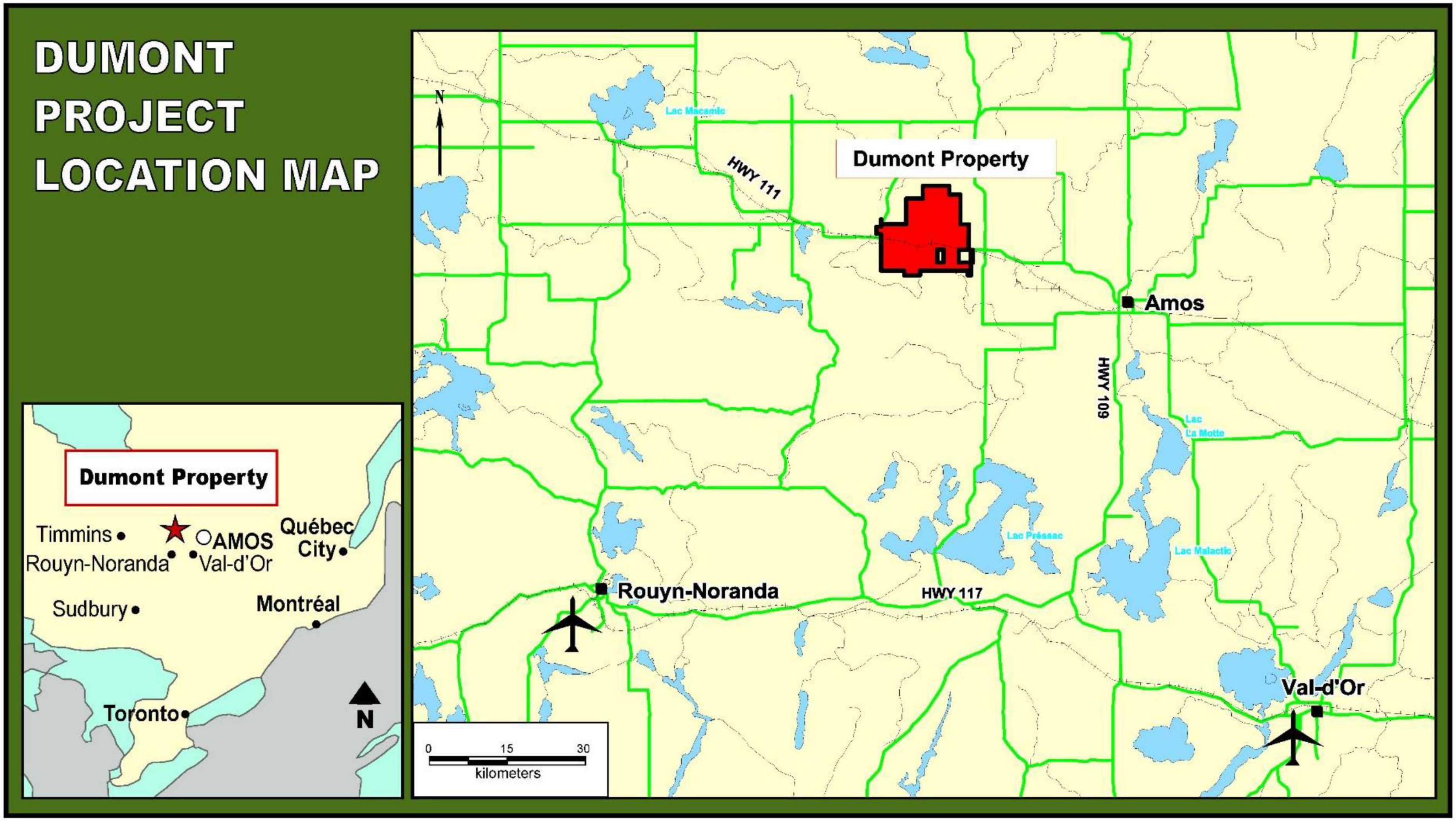
RNC advises the mineral properties comprising the Dumont property are all mineral claims. The Dumont JV holds 100% beneficial interest in seven claims; the beneficial interest in the remaining 228 claims is held 98% by the Dumont JV and 2% by Ressources Québec to secure the Ressources Québec royalty. 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 below. 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-20	43.16	2565.25	98% Magneto, 2% Ressources Québec
2025447	LAUNAY	CDC	19-Sep-20	43.12	2565.25	98% Magneto, 2% Ressources Québec
2025448	LAUNAY	CDC	19-Sep-20	43.08	2565.25	98% Magneto, 2% Ressources Québec
2025449	LAUNAY	CDC	19-Sep-20	43.05	2565.25	98% Magneto, 2% Ressources Québec
2025450	LAUNAY	CDC	19-Sep-20	43.00	2565.25	98% Magneto, 2% Ressources Québec
2025451	LAUNAY	CDC	19-Sep-20	42.97	2565.25	98% Magneto, 2% Ressources Québec
2025452	LAUNAY	CDC	19-Sep-20	42.91	2565.25	98% Magneto, 2% Ressources Québec
2025453	TRECESSON	CDC	19-Sep-20	42.82	2565.25	98% Magneto, 2% Ressources Québec
2025454	TRECESSON	CDC	19-Sep-20	42.80	2565.25	98% Magneto, 2% Ressources Québec
2025455	TRECESSON	CDC	19-Sep-20	42.59	2565.25	98% Magneto, 2% Ressources Québec
2025456	TRECESSON	CDC	19-Sep-20	42.58	2565.25	98% Magneto, 2% Ressources Québec
2025457	TRECESSON	CDC	19-Sep-20	32.69	2565.25	98% Magneto, 2% Ressources Québec
2031504	LAUNAY	CDC	6-Nov-20	47.94	2565.25	98% Magneto, 2% Ressources Québec
2031505	LAUNAY	CDC	6-Nov-20	39.80	2565.25	98% Magneto, 2% Ressources Québec
2031506	LAUNAY	CDC	6-Nov-20	39.76	2565.25	98% Magneto, 2% Ressources Québec
2031507	TRECESSON	CDC	6-Nov-20	42.60	2565.25	98% Magneto, 2% Ressources Québec
2031508	TRECESSON	CDC	6-Nov-20	42.60	2565.25	98% Magneto, 2% Ressources Québec
2031509	TRECESSON	CDC	6-Nov-20	42.58	2565.25	98% Magneto, 2% Ressources Québec
2031510	TRECESSON	CDC	6-Nov-20	42.57	2565.25	98% Magneto, 2% Ressources Québec
2031511	TRECESSON	CDC	6-Nov-20	42.56	2565.25	98% Magneto, 2% Ressources Québec
2054109	TRECESSON	CDC	8-Feb-21	42.78	2565.25	98% Magneto, 2% Ressources Québec
2054110	TRECESSON	CDC	8-Feb-21	42.75	2565.25	98% Magneto, 2% Ressources Québec
2054111	TRECESSON	CDC	8-Feb-21	42.73	2565.25	98% Magneto, 2% Ressources Québec
2054112	LAUNAY	CDC	8-Feb-21	42.63	2565.25	98% Magneto, 2% Ressources Québec
2054113	LAUNAY	CDC	8-Feb-21	42.64	2565.25	98% Magneto, 2% Ressources Québec
2054114	LAUNAY	CDC	8-Feb-21	42.63	2565.25	98% Magneto, 2% Ressources Québec
2054115	LAUNAY	CDC	8-Feb-21	42.64	2565.25	98% Magneto, 2% Ressources Québec
2054116	LAUNAY	CDC	8-Feb-21	42.63	2565.25	98% Magneto, 2% Ressources Québec
2054117	LAUNAY	CDC	8-Feb-21	42.64	2565.25	98% Magneto, 2% Ressources Québec
2054118	LAUNAY	CDC	8-Feb-21	42.65	2565.25	98% Magneto, 2% Ressources Québec
2054119	LAUNAY	CDC	8-Feb-21	42.65	2565.25	98% Magneto, 2% Ressources Québec
2054120	LAUNAY	CDC	8-Feb-21	42.65	2565.25	98% Magneto, 2% Ressources Québec
2054121	LAUNAY	CDC	8-Feb-21	42.66	2565.25	98% Magneto, 2% Ressources Québec
2054122	LAUNAY	CDC	8-Feb-21	42.67	2565.25	98% Magneto, 2% Ressources Québec
2054123	TRECESSON	CDC	8-Feb-21	42.58	2565.25	98% Magneto, 2% Ressources Québec
2054124	LAUNAY	CDC	8-Feb-21	41.80	2565.25	98% Magneto, 2% Ressources Québec
2054125	LAUNAY	CDC	8-Feb-21	41.74	2565.25	98% Magneto, 2% Ressources Québec
2054126	LAUNAY	CDC	8-Feb-21	41.69	2565.25	98% Magneto, 2% Ressources Québec
2054127	LAUNAY	CDC	8-Feb-21	41.65	2565.25	98% Magneto, 2% Ressources Québec
2054128	LAUNAY	CDC	8-Feb-21	41.59	2565.25	98% Magneto, 2% Ressources Québec
2054129	LAUNAY	CDC	8-Feb-21	41.54	2565.25	98% Magneto, 2% Ressources Québec
2054130	LAUNAY	CDC	8-Feb-21	42.39	2565.25	98% Magneto, 2% Ressources Québec
2054131	LAUNAY	CDC	8-Feb-21	42.80	2565.25	98% Magneto, 2% Ressources Québec
2054132	LAUNAY	CDC	8-Feb-21	39.72	2565.25	98% Magneto, 2% Ressources Québec
2054133	LAUNAY	CDC	8-Feb-21	39.61	2565.25	98% Magneto, 2% Ressources Québec
2054892	TRECESSON	CDC	13-Feb-21	42.71	2565.25	98% Magneto, 2% Ressources Québec
2054893	TRECESSON	CDC	13-Feb-21	42.41	2565.25	98% Magneto, 2% Ressources Québec
2054894	LAUNAY	CDC	13-Feb-21	42.41	2565.25	98% Magneto, 2% Ressources Québec
2054895	LAUNAY	CDC	13-Feb-21	42.40	2565.25	98% Magneto, 2% Ressources Québec
2054896	LAUNAY	CDC	13-Feb-21	39.69	2565.25	98% Magneto, 2% Ressources Québec
2054897	LAUNAY	CDC	13-Feb-21	42.68	2565.25	98% Magneto, 2% Ressources Québec
2054898	LAUNAY	CDC	13-Feb-21	42.73	2565.25	98% Magneto, 2% Ressources Québec
2054899	LAUNAY	CDC	13-Feb-21	43.20	2565.25	98% Magneto, 2% Ressources Québec
2054900	LAUNAY	CDC	13-Feb-21	47.82	2565.25	98% Magneto, 2% Ressources Québec
2054901	LAUNAY	CDC	13-Feb-21	38.03	2565.25	98% Magneto, 2% Ressources Québec
2054902	LAUNAY	CDC	13-Feb-21	38.74	2565.25	98% Magneto, 2% Ressources Québec
2137941	LAUNAY	CDC	4-Feb-21	42.63	1865.25	98% Magneto, 2% Ressources Québec
2137943	LAUNAY	CDC	21-Apr-21	41.84	1865.25	98% Magneto, 2% Ressources Québec

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Claim Number	Township	Type	Date Renewal Due	Area (ha)	Renewal Cost ($)	Interest
2152798	LAUNAY	CDC	19-May-20	41.89	1865.25	98% Magneto, 2% Ressources Québec
2152799	LAUNAY	CDC	19-May-20	41.95	1865.25	98% Magneto, 2% Ressources Québec
2180762	TRECESSON	CDC	12-Mar-21	29.76	1865.25	98% Magneto, 2% Ressources Québec
2180763	TRECESSON	CDC	12-Mar-21	41.68	1865.25	98% Magneto, 2% Ressources Québec
2180764	TRECESSON	CDC	12-Mar-21	41.71	1865.25	98% Magneto, 2% Ressources Québec
2180765	LAUNAY	CDC	12-Mar-21	18.67	783.25	98% Magneto, 2% Ressources Québec
2180766	LAUNAY	CDC	12-Mar-21	42.49	1865.25	98% Magneto, 2% Ressources Québec
2180767	LAUNAY	CDC	12-Mar-21	42.50	1865.25	98% Magneto, 2% Ressources Québec
2180768	LAUNAY	CDC	12-Mar-21	42.48	1865.25	98% Magneto, 2% Ressources Québec
2180769	LAUNAY	CDC	12-Mar-21	42.50	1865.25	98% Magneto, 2% Ressources Québec
2180770	LAUNAY	CDC	12-Mar-21	42.48	1865.25	98% Magneto, 2% Ressources Québec
2180771	LAUNAY	CDC	12-Mar-21	42.49	1865.25	98% Magneto, 2% Ressources Québec
2180772	LAUNAY	CDC	12-Mar-21	42.49	1865.25	98% Magneto, 2% Ressources Québec
2180773	LAUNAY	CDC	12-Mar-21	42.49	1865.25	98% Magneto, 2% Ressources Québec
2180774	LAUNAY	CDC	12-Mar-21	42.48	1865.25	98% Magneto, 2% Ressources Québec
2180775	LAUNAY	CDC	12-Mar-21	42.47	1865.25	98% Magneto, 2% Ressources Québec
2180776	LAUNAY	CDC	12-Mar-21	42.48	1865.25	98% Magneto, 2% Ressources Québec
2180777	LAUNAY	CDC	12-Mar-21	42.48	1865.25	98% Magneto, 2% Ressources Québec
2180778	LAUNAY	CDC	12-Mar-21	42.48	1865.25	98% Magneto, 2% Ressources Québec
2180779	LAUNAY	CDC	12-Mar-21	42.46	1865.25	98% Magneto, 2% Ressources Québec
2180780	LAUNAY	CDC	12-Mar-21	42.48	1865.25	98% Magneto, 2% Ressources Québec
2180781	LAUNAY	CDC	12-Mar-21	42.48	1865.25	98% Magneto, 2% Ressources Québec
2180782	LAUNAY	CDC	12-Mar-21	42.46	1865.25	98% Magneto, 2% Ressources Québec
2180783	LAUNAY	CDC	12-Mar-21	35.60	1865.25	98% Magneto, 2% Ressources Québec
2180784	LAUNAY	CDC	12-Mar-21	19.53	783.25	98% Magneto, 2% Ressources Québec
2180785	LAUNAY	CDC	12-Mar-21	42.61	1865.25	98% Magneto, 2% Ressources Québec
2180786	LAUNAY	CDC	12-Mar-21	56.93	1865.25	98% Magneto, 2% Ressources Québec
2180787	LAUNAY	CDC	12-Mar-21	56.93	1865.25	98% Magneto, 2% Ressources Québec
2180788	LAUNAY	CDC	12-Mar-21	56.93	1865.25	98% Magneto, 2% Ressources Québec
2180789	LAUNAY	CDC	12-Mar-21	56.93	1865.25	98% Magneto, 2% Ressources Québec
2180790	LAUNAY	CDC	12-Mar-21	56.93	1865.25	98% Magneto, 2% Ressources Québec
2180791	LAUNAY	CDC	12-Mar-21	56.92	1865.25	98% Magneto, 2% Ressources Québec
2180792	LAUNAY	CDC	12-Mar-21	56.92	1865.25	98% Magneto, 2% Ressources Québec
2180793	LAUNAY	CDC	12-Mar-21	56.92	1865.25	98% Magneto, 2% Ressources Québec
2180794	LAUNAY	CDC	12-Mar-21	56.92	1865.25	98% Magneto, 2% Ressources Québec
2180795	LAUNAY	CDC	12-Mar-21	56.92	1865.25	98% Magneto, 2% Ressources Québec
2180796	LAUNAY	CDC	12-Mar-21	56.91	1865.25	98% Magneto, 2% Ressources Québec
2180797	LAUNAY	CDC	12-Mar-21	56.91	1865.25	98% Magneto, 2% Ressources Québec
2180798	LAUNAY	CDC	12-Mar-21	56.91	1865.25	98% Magneto, 2% Ressources Québec
2180799	LAUNAY	CDC	12-Mar-21	56.91	1865.25	98% Magneto, 2% Ressources Québec
2180800	LAUNAY	CDC	12-Mar-21	51.74	1865.25	98% Magneto, 2% Ressources Québec
2180801	LAUNAY	CDC	12-Mar-21	56.90	1865.25	98% Magneto, 2% Ressources Québec
2180802	LAUNAY	CDC	12-Mar-21	56.90	1865.25	98% Magneto, 2% Ressources Québec
2180803	LAUNAY	CDC	12-Mar-21	56.90	1865.25	98% Magneto, 2% Ressources Québec
2180804	LAUNAY	CDC	12-Mar-21	56.90	1865.25	98% Magneto, 2% Ressources Québec
2180805	LAUNAY	CDC	12-Mar-21	43.32	1865.25	98% Magneto, 2% Ressources Québec
2180806	LAUNAY	CDC	12-Mar-21	24.54	783.25	98% Magneto, 2% Ressources Québec
2180807	LAUNAY	CDC	12-Mar-21	21.50	783.25	98% Magneto, 2% Ressources Québec
2180808	LAUNAY	CDC	12-Mar-21	21.10	783.25	98% Magneto, 2% Ressources Québec
2180809	LAUNAY	CDC	12-Mar-21	20.68	783.25	98% Magneto, 2% Ressources Québec
2180810	LAUNAY	CDC	12-Mar-21	15.48	783.25	98% Magneto, 2% Ressources Québec
2194108	TRECESSON	CDC	9-Nov-19	39.26	1865.25	100% Magneto
2194109	TRECESSON	CDC	9-Nov-19	39.26	1865.25	100% Magneto
2194110	TRECESSON	CDC	9-Nov-19	39.27	1865.25	100% Magneto
2194115	TRECESSON	CDC	9-Nov-19	38.73	1865.25	100% Magneto
2204674	TRECESSON	CDC	7-Feb-20	39.12	1865.25	98% Magneto, 2% Ressources Québec
2204675	TRECESSON	CDC	7-Feb-20	39.13	1865.25	98% Magneto, 2% Ressources Québec
2204676	LAUNAY	CDC	7-Feb-20	38.82	1865.25	98% Magneto, 2% Ressources Québec
2204677	LAUNAY	CDC	7-Feb-20	38.90	1865.25	98% Magneto, 2% Ressources Québec
2204678	LAUNAY	CDC	7-Feb-20	38.91	1865.25	98% Magneto, 2% Ressources Québec

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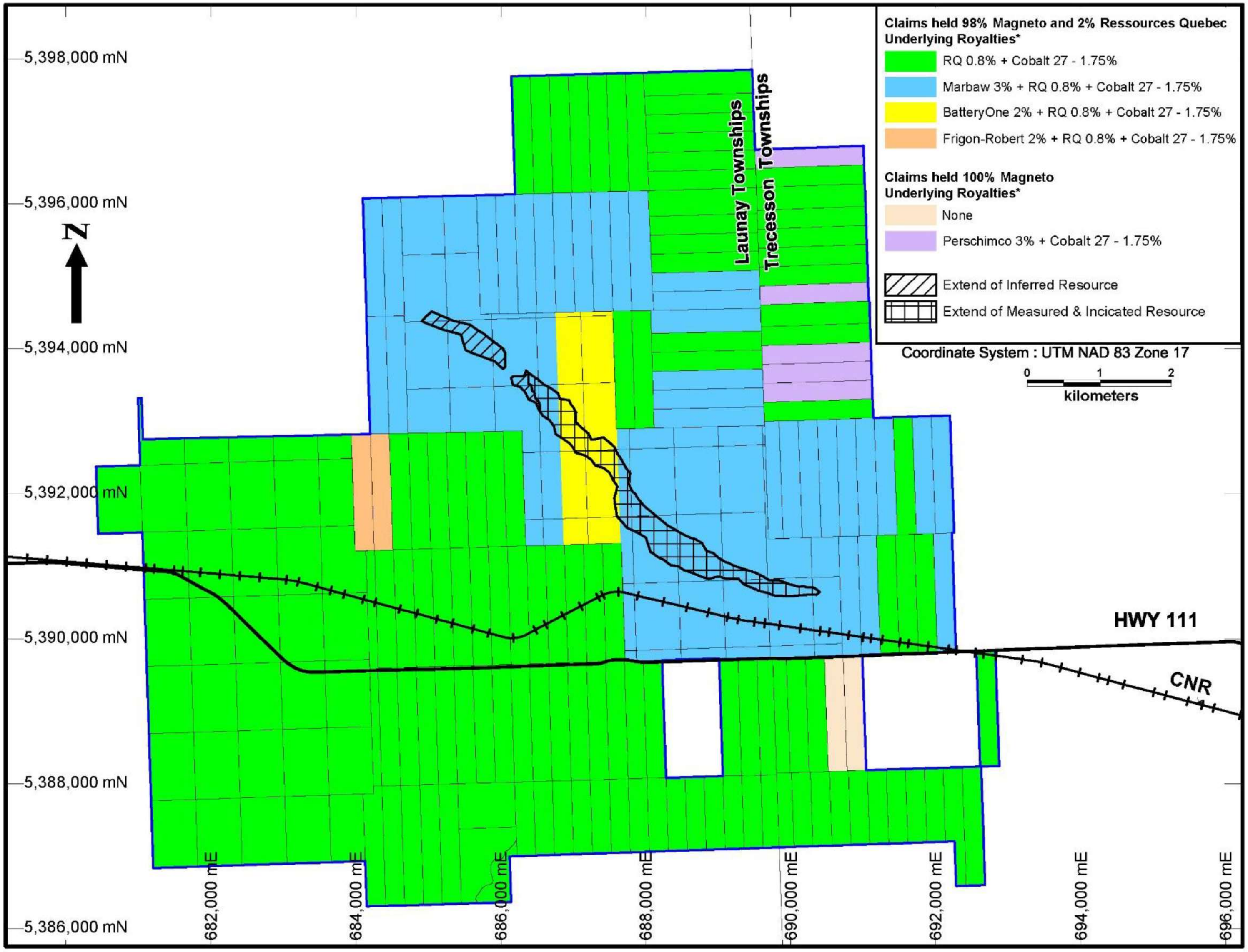
Claim Number	Township	Type	Date Renewal Due	Area (ha)	Renewal Cost ($)	Interest
2204679	LAUNAY	CDC	7-Feb-20	53.04	1865.25	98% Magneto, 2% Ressources Québec
2220724	TRECESSON	CDC	25-Apr-20	39.12	1865.25	100% Magneto
2224811	LAUNAY	CDC	29-Apr-20	42.67	1865.25	98% Magneto, 2% Ressources Québec
2224812	LAUNAY	CDC	29-Apr-20	42.68	1865.25	98% Magneto, 2% Ressources Québec
2224813	LAUNAY	CDC	29-Apr-20	42.67	1865.25	98% Magneto, 2% Ressources Québec
2224814	LAUNAY	CDC	29-Apr-20	42.68	1865.25	98% Magneto, 2% Ressources Québec
2224815	LAUNAY	CDC	29-Apr-20	42.90	1865.25	98% Magneto, 2% Ressources Québec
2229201	TRECESSON	CDC	4-May-20	39.22	1865.25	98% Magneto, 2% Ressources Québec
2229202	LAUNAY	CDC	4-May-20	38.86	1865.25	98% Magneto, 2% Ressources Québec
2229203	LAUNAY	CDC	4-May-20	38.81	1865.25	98% Magneto, 2% Ressources Québec
2235659	LAUNAY	CDC	12-Mar-21	56.94	1865.25	98% Magneto, 2% Ressources Québec
2247681	LAUNAY	CDC	26-Aug-20	42.68	1865.25	98% Magneto, 2% Ressources Québec
2247682	LAUNAY	CDC	26-Aug-20	42.68	1865.25	98% Magneto, 2% Ressources Québec
2249118	TRECESSON	CDC	8-Sep-20	39.24	1865.25	98% Magneto, 2% Ressources Québec
2251083	LAUNAY	CDC	23-Sep-20	41.78	1865.25	98% Magneto, 2% Ressources Québec
2255617	TRECESSON	CDC	24-Oct-20	42.91	1865.25	98% Magneto, 2% Ressources Québec
2255618	LAUNAY	CDC	24-Oct-20	43.30	1865.25	98% Magneto, 2% Ressources Québec
2255619	LAUNAY	CDC	24-Oct-20	43.33	1865.25	98% Magneto, 2% Ressources Québec
2255620	LAUNAY	CDC	24-Oct-20	43.30	1865.25	98% Magneto, 2% Ressources Québec
2255621	LAUNAY	CDC	24-Oct-20	43.31	1865.25	98% Magneto, 2% Ressources Québec
2255622	LAUNAY	CDC	24-Oct-20	52.65	1865.25	98% Magneto, 2% Ressources Québec
2255623	LAUNAY	CDC	24-Oct-20	48.65	1865.25	98% Magneto, 2% Ressources Québec
2255624	TRECESSON	CDC	24-Oct-20	41.92	1865.25	98% Magneto, 2% Ressources Québec
2255625	TRECESSON	CDC	24-Oct-20	39.09	1865.25	98% Magneto, 2% Ressources Québec
2255626	TRECESSON	CDC	24-Oct-20	47.12	1865.25	98% Magneto, 2% Ressources Québec
2255627	TRECESSON	CDC	24-Oct-20	39.19	1865.25	98% Magneto, 2% Ressources Québec
2255628	LAUNAY	CDC	24-Oct-20	38.91	1865.25	98% Magneto, 2% Ressources Québec
2255629	LAUNAY	CDC	24-Oct-20	39.05	1865.25	98% Magneto, 2% Ressources Québec
2255630	LAUNAY	CDC	24-Oct-20	39.16	1865.25	98% Magneto, 2% Ressources Québec
2255631	LAUNAY	CDC	24-Oct-20	48.20	1865.25	98% Magneto, 2% Ressources Québec
2255632	LAUNAY	CDC	24-Oct-20	56.94	1865.25	98% Magneto, 2% Ressources Québec
2255633	LAUNAY	CDC	24-Oct-20	56.94	1865.25	98% Magneto, 2% Ressources Québec
2255634	LAUNAY	CDC	24-Oct-20	56.94	1865.25	98% Magneto, 2% Ressources Québec
2255635	LAUNAY	CDC	24-Oct-20	56.94	1865.25	98% Magneto, 2% Ressources Québec
2255636	LAUNAY	CDC	24-Oct-20	56.93	1865.25	98% Magneto, 2% Ressources Québec
2255637	LAUNAY	CDC	24-Oct-20	56.93	1865.25	98% Magneto, 2% Ressources Québec
2255638	LAUNAY	CDC	24-Oct-20	56.93	1865.25	98% Magneto, 2% Ressources Québec
2255639	LAUNAY	CDC	24-Oct-20	56.94	1865.25	98% Magneto, 2% Ressources Québec
2255640	LAUNAY	CDC	24-Oct-20	43.32	1865.25	98% Magneto, 2% Ressources Québec
2255641	LAUNAY	CDC	24-Oct-20	22.12	783.25	98% Magneto, 2% Ressources Québec
2255642	LAUNAY	CDC	24-Oct-20	26.69	1865.25	98% Magneto, 2% Ressources Québec
2255643	LAUNAY	CDC	24-Oct-20	26.66	1865.25	98% Magneto, 2% Ressources Québec
2255644	LAUNAY	CDC	24-Oct-20	26.63	1865.25	98% Magneto, 2% Ressources Québec
2255645	LAUNAY	CDC	24-Oct-20	26.60	1865.25	98% Magneto, 2% Ressources Québec
2255646	LAUNAY	CDC	24-Oct-20	26.56	1865.25	98% Magneto, 2% Ressources Québec
2255647	LAUNAY	CDC	24-Oct-20	26.54	1865.25	98% Magneto, 2% Ressources Québec
2255648	LAUNAY	CDC	24-Oct-20	26.50	1865.25	98% Magneto, 2% Ressources Québec
2255649	LAUNAY	CDC	24-Oct-20	26.48	1865.25	98% Magneto, 2% Ressources Québec
2255650	LAUNAY	CDC	24-Oct-20	26.43	1865.25	98% Magneto, 2% Ressources Québec
2255651	LAUNAY	CDC	24-Oct-20	26.41	1865.25	98% Magneto, 2% Ressources Québec
2255652	LAUNAY	CDC	24-Oct-20	26.37	1865.25	98% Magneto, 2% Ressources Québec
2255653	LAUNAY	CDC	24-Oct-20	26.34	1865.25	98% Magneto, 2% Ressources Québec
2255654	LAUNAY	CDC	24-Oct-20	26.30	1865.25	98% Magneto, 2% Ressources Québec
2255655	LAUNAY	CDC	24-Oct-20	22.36	783.25	98% Magneto, 2% Ressources Québec
2255656	TRECESSON	CDC	24-Oct-20	20.80	783.25	98% Magneto, 2% Ressources Québec
2255657	TRECESSON	CDC	24-Oct-20	26.82	1865.25	98% Magneto, 2% Ressources Québec
2255658	TRECESSON	CDC	24-Oct-20	26.81	1865.25	98% Magneto, 2% Ressources Québec
2255659	TRECESSON	CDC	24-Oct-20	26.79	1865.25	98% Magneto, 2% Ressources Québec
2255660	TRECESSON	CDC	24-Oct-20	26.79	1865.25	98% Magneto, 2% Ressources Québec
2255661	TRECESSON	CDC	24-Oct-20	26.78	1865.25	98% Magneto, 2% Ressources Québec

Claim Number	Township	Type	Date Renewal Due	Area (ha)	Renewal Cost ($)	Interest
2255662	TRECESSON	CDC	24-Oct-20	26.76	1865.25	98% Magneto, 2% Ressources Québec
2255663	TRECESSON	CDC	24-Oct-20	26.76	1865.25	98% Magneto, 2% Ressources Québec
2255664	TRECESSON	CDC	24-Oct-20	26.69	1865.25	98% Magneto, 2% Ressources Québec
2255665	TRECESSON	CDC	24-Oct-20	35.26	1865.25	98% Magneto, 2% Ressources Québec
2267113	LAUNAY	CDC	11-Jan-21	56.90	1865.25	98% Magneto, 2% Ressources Québec
2276187	TRECESSON	CDC	8-Mar-21	39.29	1865.25	98% Magneto, 2% Ressources Québec
2276188	TRECESSON	CDC	8-Mar-21	45.83	1865.25	98% Magneto, 2% Ressources Québec
2377418	LAUNAY	CDC	13-Jan-20	56.92	2565.25	98% Magneto, 2% Ressources Québec
2377419	LAUNAY	CDC	13-Jan-20	56.92	2565.25	98% Magneto, 2% Ressources Québec
2377420	LAUNAY	CDC	13-Jan-20	56.92	2565.25	98% Magneto, 2% Ressources Québec
2377421	TRECESSON	CDC	13-Jan-20	56.92	2565.25	98% Magneto, 2% Ressources Québec
2377422	LAUNAY	CDC	13-Jan-20	56.91	2565.25	98% Magneto, 2% Ressources Québec
2377423	LAUNAY	CDC	13-Jan-20	56.91	2565.25	98% Magneto, 2% Ressources Québec
2377424	LAUNAY	CDC	13-Jan-20	56.91	2565.25	98% Magneto, 2% Ressources Québec
2377425	LAUNAY	CDC	13-Jan-20	56.90	2565.25	98% Magneto, 2% Ressources Québec
2377426	LAUNAY	CDC	13-Jan-20	56.90	2565.25	98% Magneto, 2% Ressources Québec
2377427	LAUNAY	CDC	13-Jan-20	56.90	2565.25	98% Magneto, 2% Ressources Québec
2377428	LAUNAY	CDC	13-Jan-20	56.90	2565.25	98% Magneto, 2% Ressources Québec
2377429	LAUNAY	CDC	13-Jan-20	56.90	2565.25	98% Magneto, 2% Ressources Québec
2377430	LAUNAY	CDC	13-Jan-20	56.89	2565.25	98% Magneto, 2% Ressources Québec
2377431	LAUNAY	CDC	13-Jan-20	56.88	2565.25	98% Magneto, 2% Ressources Québec
2377432	LAUNAY	CDC	13-Jan-20	56.88	2565.25	98% Magneto, 2% Ressources Québec
2377433	LAUNAY	CDC	13-Jan-20	56.88	2565.25	98% Magneto, 2% Ressources Québec
2377434	LAUNAY	CDC	13-Jan-20	36.08	2565.25	98% Magneto, 2% Ressources Québec
2377435	LAUNAY	CDC	13-Jan-20	54.69	2565.25	98% Magneto, 2% Ressources Québec
2377436	LAUNAY	CDC	13-Jan-20	54.41	2565.25	98% Magneto, 2% Ressources Québec
2377437	LAUNAY	CDC	13-Jan-20	46.65	2565.25	98% Magneto, 2% Ressources Québec
2377438	LAUNAY	CDC	13-Jan-20	37.90	2565.25	98% Magneto, 2% Ressources Québec
2377439	LAUNAY	CDC	13-Jan-20	43.69	2565.25	98% Magneto, 2% Ressources Québec
2377440	LAUNAY	CDC	13-Jan-20	36.43	2565.25	98% Magneto, 2% Ressources Québec
2377441	LAUNAY	CDC	13-Jan-20	9.06	1033.25	98% Magneto, 2% Ressources Québec
2377442	LAUNAY	CDC	13-Jan-20	23.21	1033.25	98% Magneto, 2% Ressources Québec
2377443	LAUNAY	CDC	13-Jan-20	45.83	2565.25	98% Magneto, 2% Ressources Québec
2377444	LAUNAY	CDC	13-Jan-20	4.39	1033.25	98% Magneto, 2% Ressources Québec
2377445	LAUNAY	CDC	13-Jan-20	22.27	1033.25	98% Magneto, 2% Ressources Québec
2377446	LAUNAY	CDC	13-Jan-20	3.95	1033.25	98% Magneto, 2% Ressources Québec
2377447	LAUNAY	CDC	13-Jan-20	2.28	1033.25	98% Magneto, 2% Ressources Québec
2377448	LAUNAY	CDC	13-Jan-20	14.85	1033.25	98% Magneto, 2% Ressources Québec
2377449	LAUNAY	CDC	13-Jan-20	31.37	2565.25	98% Magneto, 2% Ressources Québec
2377450	LAUNAY	CDC	13-Jan-20	45.79	2565.25	98% Magneto, 2% Ressources Québec
2377451	LAUNAY	CDC	13-Jan-20	40.94	2565.25	98% Magneto, 2% Ressources Québec
2377452	LAUNAY	CDC	13-Jan-20	2.57	1033.25	98% Magneto, 2% Ressources Québec
2377453	LAUNAY	CDC	13-Jan-20	8.83	1033.25	98% Magneto, 2% Ressources Québec
2377454	LAUNAY	CDC	13-Jan-20	17.22	1033.25	98% Magneto, 2% Ressources Québec
2377455	LAUNAY	CDC	13-Jan-20	9.02	1033.25	98% Magneto, 2% Ressources Québec
2377456	LAUNAY	CDC	13-Jan-20	16.77	1033.25	98% Magneto, 2% Ressources Québec
2377457	LAUNAY	CDC	13-Jan-20	9.21	1033.25	98% Magneto, 2% Ressources Québec
2377458	LAUNAY	CDC	13-Jan-20	16.32	1033.25	98% Magneto, 2% Ressources Québec
2377459	TRECESSON	CDC	13-Jan-20	10.18	1033.25	98% Magneto, 2% Ressources Québec
2377460	TRECESSON	CDC	13-Jan-20	35.03	2565.25	98% Magneto, 2% Ressources Québec
2377461	LAUNAY	CDC	13-Jan-20	2.88	1033.25	98% Magneto, 2% Ressources Québec
2377462	LAUNAY	CDC	13-Jan-20	0.81	1033.25	98% Magneto, 2% Ressources Québec
2377463	TRECESSON	CDC	13-Jan-20	6.39	1033.25	98% Magneto, 2% Ressources Québec
2377464	TRECESSON	CDC	13-Jan-20	35.71	2565.25	98% Magneto, 2% Ressources Québec
2377465	TRECESSON	CDC	13-Jan-20	21.18	1033.25	98% Magneto, 2% Ressources Québec
2487714	TRECESSON	CDC	23-Mar-21	41.73	1265.25	100% Magneto
2487715	TRECESSON	CDC	23-Mar-21	41.75	1265.25	100% Magneto

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 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. RNC acquired a 100% interest in the claims 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 satisfaction of certain conditions (such conditions, other than the receipt by RNC of a notice from Marbaw requesting that these shares be issued, have been satisfied); and (2) payment of $1,250,000 to Marbaw on 8 March 2008 (This amount was paid).

RNC also committed to incurring a minimum expenditure of $8,000,000 on the property prior to ceasing operations. This commitment was satisfied in 2008. The Marbaw property is subject to a 3% NSR royalty payable (now by Magneto Investments Limited Partnership) to Marbaw. Half of this 3% NSR may be re-purchased at any time for $10,000,000.

This property is also subject to the Ressources Québec royalty and Cobalt 27 royalty.

4.2.3.2 BatteryOne 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.

Following the exercise of the Coyle-Roby Option, the property is subject to a 2% NSR royalty payable to Terrence Coyle (1%) and Michel Roby (1%). On Jan. 22, 2019, BatteryOne Royalty Corp. (BatteryOne) announced that the had purchased this royalty from Coyle-Roby. Half (50%) of this 2% NSR may be repurchased for $1,000,000 at any time. An advance royalty of $5,000 per year is also payable to beginning in 2011. Scheduled royalty payments have been made annually in October since 2011.

These claims are also subject to the Ressources Québec royalty and Cobalt 27 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%). Half (50%) of this 2% NSR may be repurchased for $1,000,000 at any time.

These claims are also subject to the Ressources Québec royalty and Cobalt 27 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. This NSR may be bought back in tranches 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 also subject to the Cobalt 27 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. The Dumont JV 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 Cobalt 27 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.

On July 8th, 2015, Royal Nickel closed a royalty and private placement transaction with Orion Mine Finance ("Orion"). RNC received gross proceeds of US$10 million from Orion in exchange for a 0.75% net smelter return royalty in the Dumont Project and 10 million RNC common shares (issued at $0.395 per share. Half (50%) of the royalty (0.375%) may be repurchased for a cash payment of US$15 million on the 3rd, 4th or 5th anniversary of closing.

On Feb. 22, 2018 Cobalt 27 Capital Corp. ("Cobalt 27") announced that it had agreed to acquire these existing royalties totalling 1.75% Net Smelter Return ("NSR") royalty on all future production over all metals from the Dumont Nickel-Cobalt Project ("Dumont").Consequently, Cobalt 27 now holds an aggregate 1.75% NSR royalty that contains a US$15 million buyback right to the Dumont joint venture to repurchase 0.375% of the 1.75% NSR ("Repurchase Option"), which if exercised would result in a 1.375% remaining NSR. The one-time Repurchase Option is only exercisable on the third, fourth or fifth anniversary of the original royalty agreement dated July 8, 2015. The Cobalt 27 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 Energy and Natural Resources (MERN) 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 the Dumont JV is conducted under the terms of access agreements between the Dumont

JV and individual landowners. Stream crossings have been constructed under permits issued variously or jointly by the MERN, CPTAQ, and the Quebec Ministry of Environment (MELCC). 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 initiated discussions with the local Algonquin Conseil de la Première nation Abitibiwinni ("PNA") and on 5 April 2013 entered into a memorandum of understanding for cooperation regarding the development of the Dumont Nickel project. On the basis of this MOU, negotiations with PNA to establish an Impact and Benefits Agreement ("IBA") were completed and on May 2, 2017 the Company and the PNA announced the signing of an IBA for the Dumont Project. The IBA serves as a framework to govern the relationship with PNA and lays out the commitments of the parties regarding the impacts and benefits of the Dumont Project. RNC's interest in the is agreement was assigned to the Dumont JV at the time of the joint venture transaction. Consequently, the parties to the IBA are PNA and the Dumont JV.

4.4 Mineral and Surface 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. Of these privately held surface rights approximately 1,438 hectares are required for the development of the Dumont project. The Dumont JV has purchased approximately 660 ha (46%) of these private surface rights and holds options to purchase on the remainder 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. A portion of the surface rights over the Dumont Project claims were previously classified as an agricultural zone within the meaning of the Act respecting the preservation of agricultural land and agricultural activities, RSQ, c P-41.1. Exclusion of these lands from the agricultural zone, which is required to conduct mining activity on these lands, was granted by the CPTAQ in February 2013 with minor additional lands being excluded in May 2015.

Use of surface rights for mining and associated activities under the terms of a mining lease is subject to environmental permitting. The Dumont JV has obtained the main provincial and federal environmental authorizations as noted in Chapter 20. Access to surface rights for private lands is obtained through purchase of these lands from private surface rights holders as noted above. Access to surface rights for public lands would be obtained through the mining lease and surface lease processes with the MERN. 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. The closure plan for the Dumont Project is outlined in Chapter 20. 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.

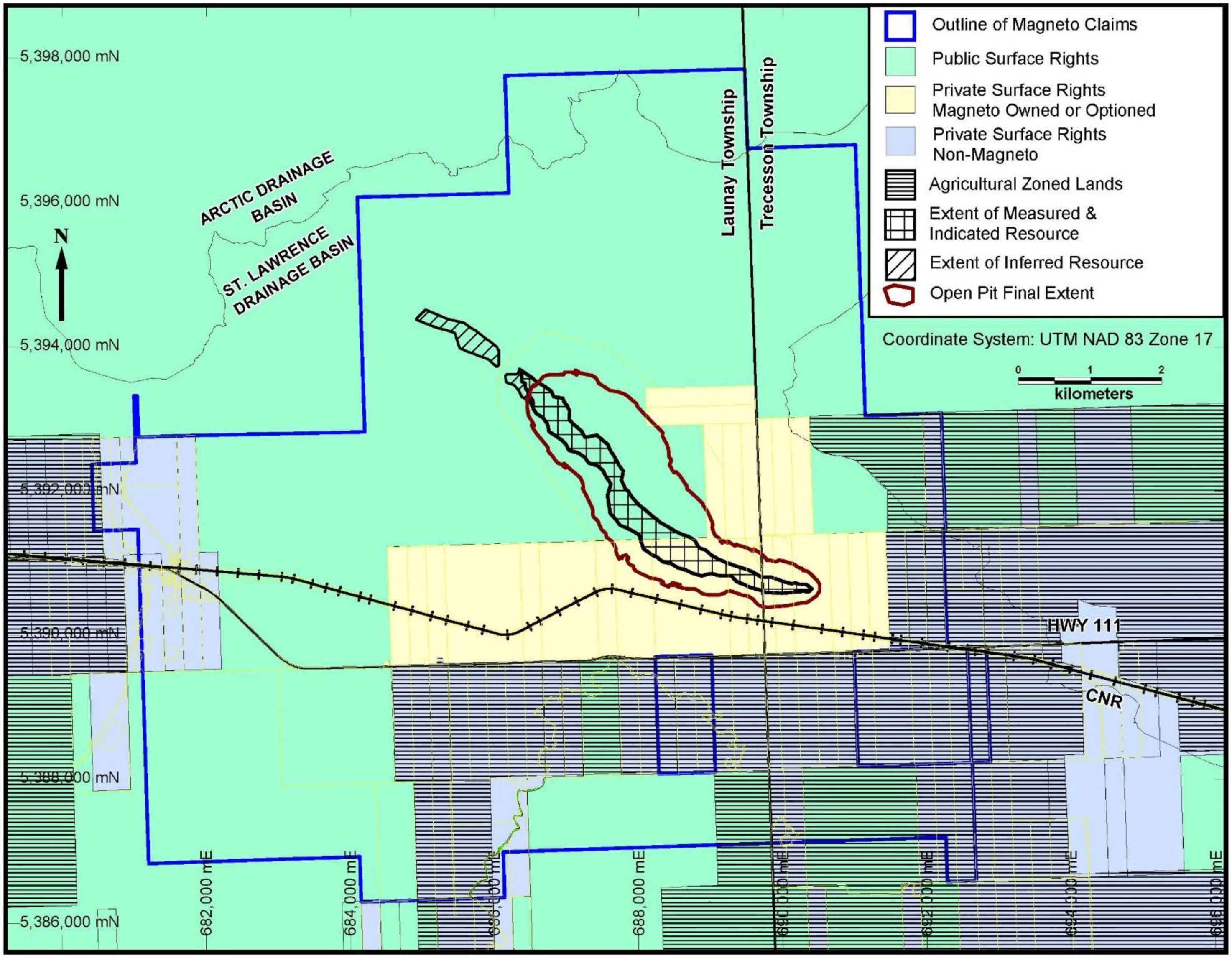
Ausenco

4.5 Environmental Liabilities

Neither Ausenco nor the Dumont JV 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



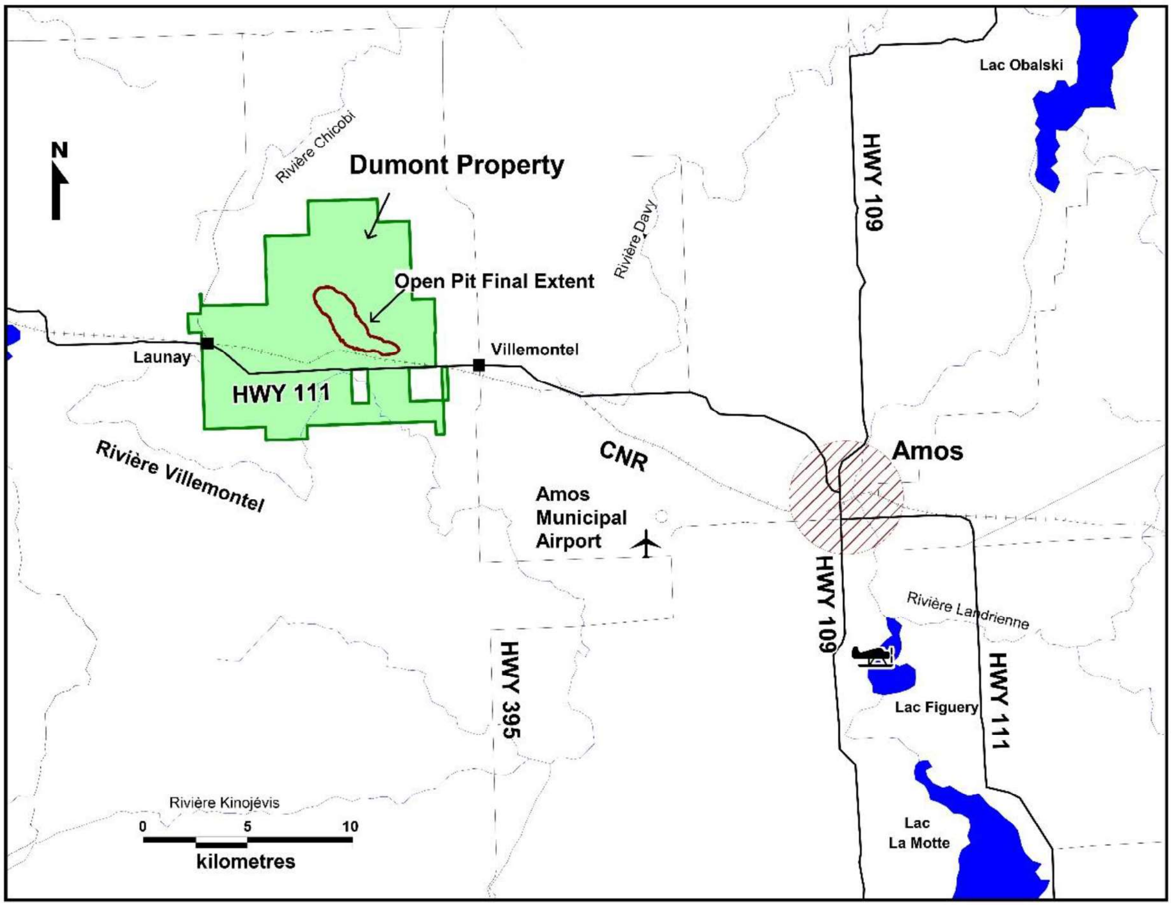
Source: RNC.

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 locally are agriculture and forestry; and Amos serves as a regional services hub. 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 (2016 Census population 42,334), which is 120 km by road to the southwest, and Val d'Or (2016 Census population 33,871), 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 120 kV 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 quarry and, if required under exceptional circumstances, from the Villemontel River, at a point located approximately 5 km from the planned site for the mill.
- Gas **–** Although the use of propane gas delivered by tanker truck is considered for heating buildings in this study, an existing natural gas pipeline extends to within approximately 25 km from the south edge of the property which could be considered for future requirements.

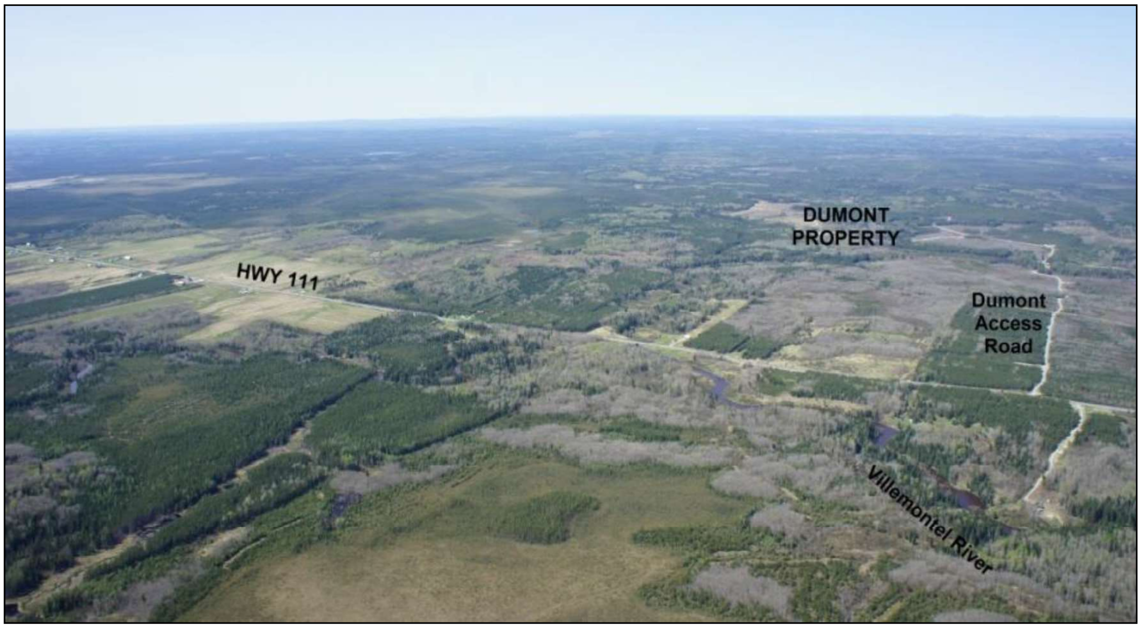
5.3 Climate

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 the same as 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.

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 test work (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 test work 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 test work 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 test work 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

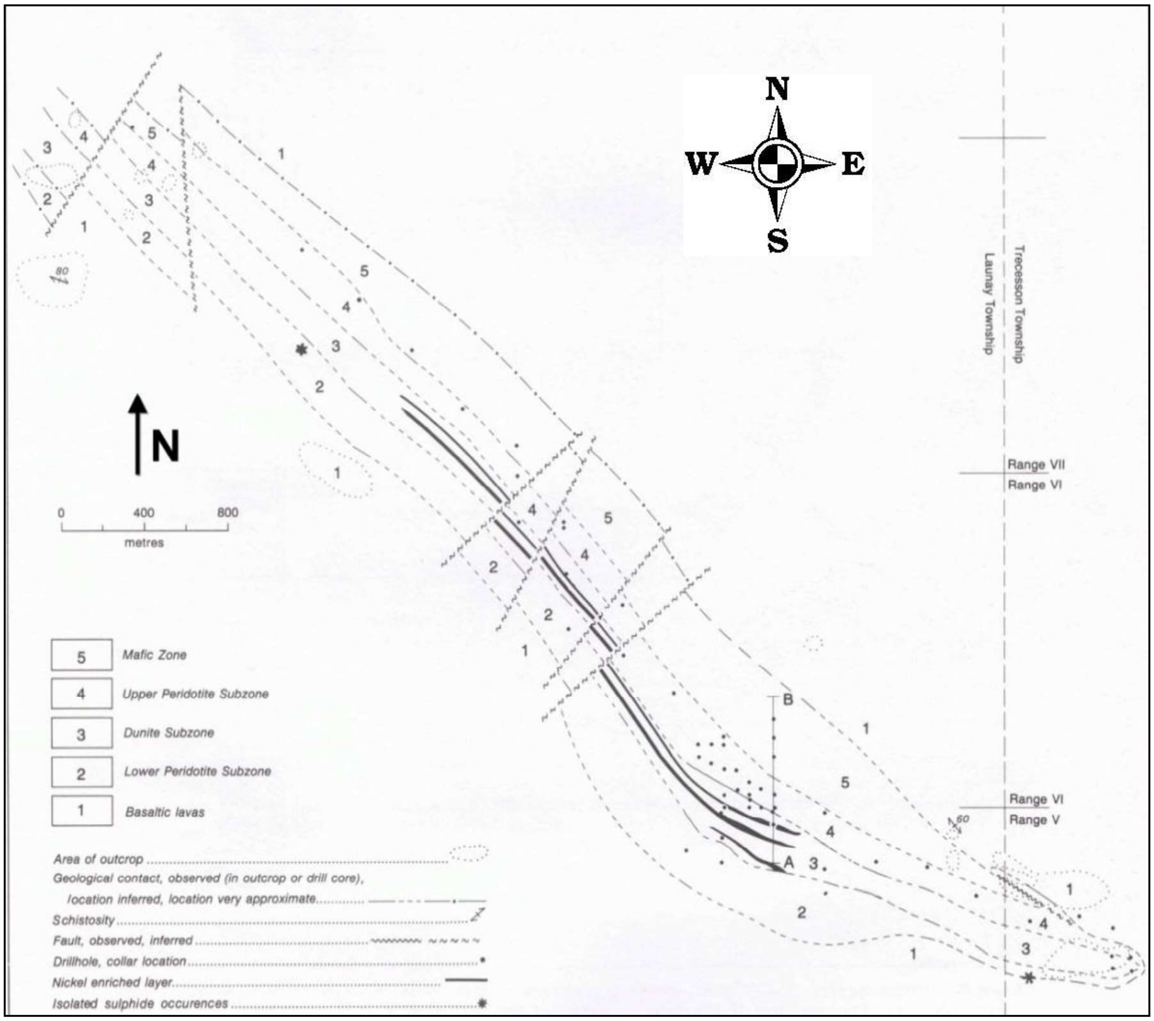
In 1982, exploration resumed on the property and four percussion 15.2 cm (6") diameter holes were drilled and cuttings recovered to prepare a bulk sample.

In 1986, CRM conducted, for the account of Magnitec, a H_2SO_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 H_2SO_3 leach test and indicated that the tailings from an operation on the Dumont property would give a low extraction rate of the SO_2 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 2008 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 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 test work, 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 2010 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 2011 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 by-product, additional recovery optimization and use of inpit crushing or trolley system were identified for further study in the PFS.

6.1.5.4 2012 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 entitled, "Technical Report on the Dumont project, Launay and Trécesson Townships, Quebec, Canada" (22 June 2012) (Ausenco, 2012). 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™ 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 test work, 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 by-product, additional recovery optimization and optimization of the trolley system configuration were identified for further study in the feasibility study.

6.1.5.5 2013 RNC Feasibility Study

Following the positive results of the revised pre-feasibility study Ausenco was commissioned by RNC in May 2012 to prepare the feasibility study and NI 43-101 compliant technical report on the project entitled, “Technical Report on the Dumont project, Launay and Trécesson Townships, Quebec, Canada” (25 July 2013) (Ausenco, 2013). 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.

Highlights of the 2013 Feasibility Study included:

- $1.1 billion after-tax NPV8%
- 15% after-tax internal rate of return
- C1 cash costs2 of $4.01/lb ($8,840/t) during initial phase and $4.31/lb ($9,502/t) over life-of-project (low 2nd quartile of cash cost curve)
- Estimated annual average of $427 million EBITDA and $238 million free cash flow over the 20-year mine life
- Minimal increase in initial capital expenditure estimate to $1.2 billion compared to 2012 revised pre-feasibility Study
- De-risked feasibility study capex increased by only 7% compared to 2012 revised pre-feasibility study (which used Q4 2010 basis for costing)

- 11% increase in ore reserves compared to 2012 revised pre-feasibility study to 1.2 billion tonnes containing 6.9 billion pounds of nickel to support a 33-year project life including 1.3 billion pounds of proven reserve
- Established 1 million ounces PGE (platinum + palladium) reserve
- Initial nickel production of 73 million pounds (Mlbs) (33 kt) annually, expanded in year 5 to an annual average of 113 Mlbs (51 kt) for the remainder of the 20 year mine life

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 tonnes 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 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 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 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% Ni (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	1,030	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 by-products 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.

Ausenco

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.

6.3.6 2013 Mineral Resource Estimation & Mineral Reserve (RNC)

The 13 April 2012 Mineral Resource estimate contained in the June 2012 Technical Report was superseded by an updated mineral resource effective 30 April 2013 (Ausenco, 2013). This resource is described in Chapter 14 herein and also forms the basis for the updated 2019 Mineral Reserves and Feasibility Study presented in this report.

The 2013 Mineral Reserves were prepared under the direction of David A. Warren, Eng., Principle Consultant - Mining with Snowden Mining Industry Consultants, based on the 30 April 2013 mineral resource block model. 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.

The 2013 Dumont mineral reserves are summarized in Table 6-11.

Table 6-11: 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**

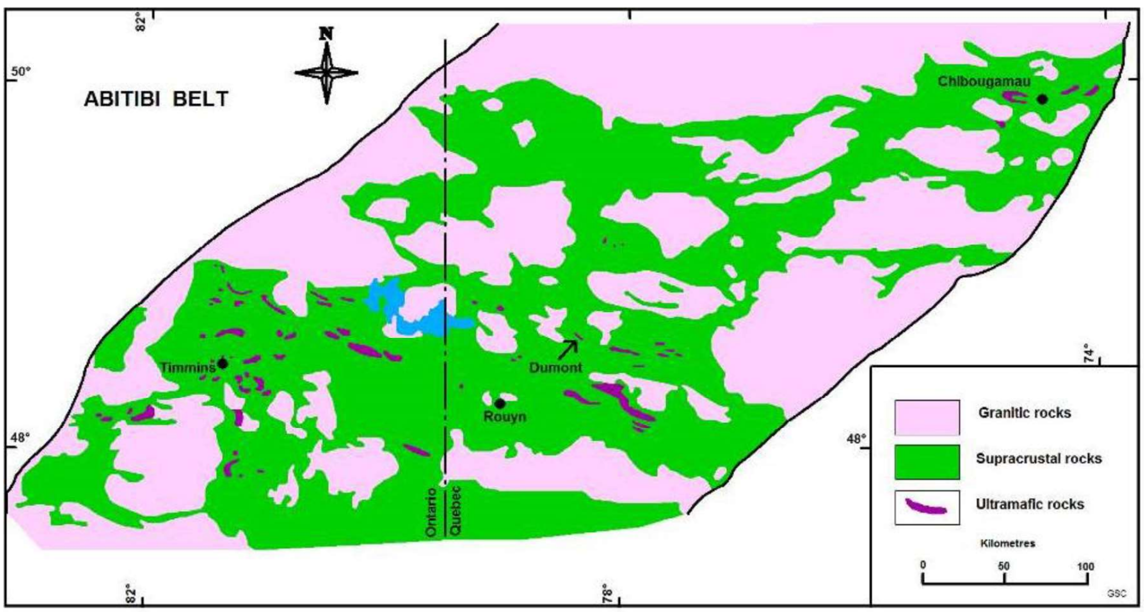
1. *Reported at a cut-off grade of 0.15% nickel inside an engineered pit design based on a Lerchs-Grossmann (LG) optimized pit shell using a nickel price of US$5.58 per pound (62% of the long-term forecast of US$9.00 per pound), average metallurgical recovery of 43%, marginal processing and G&A costs 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.

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

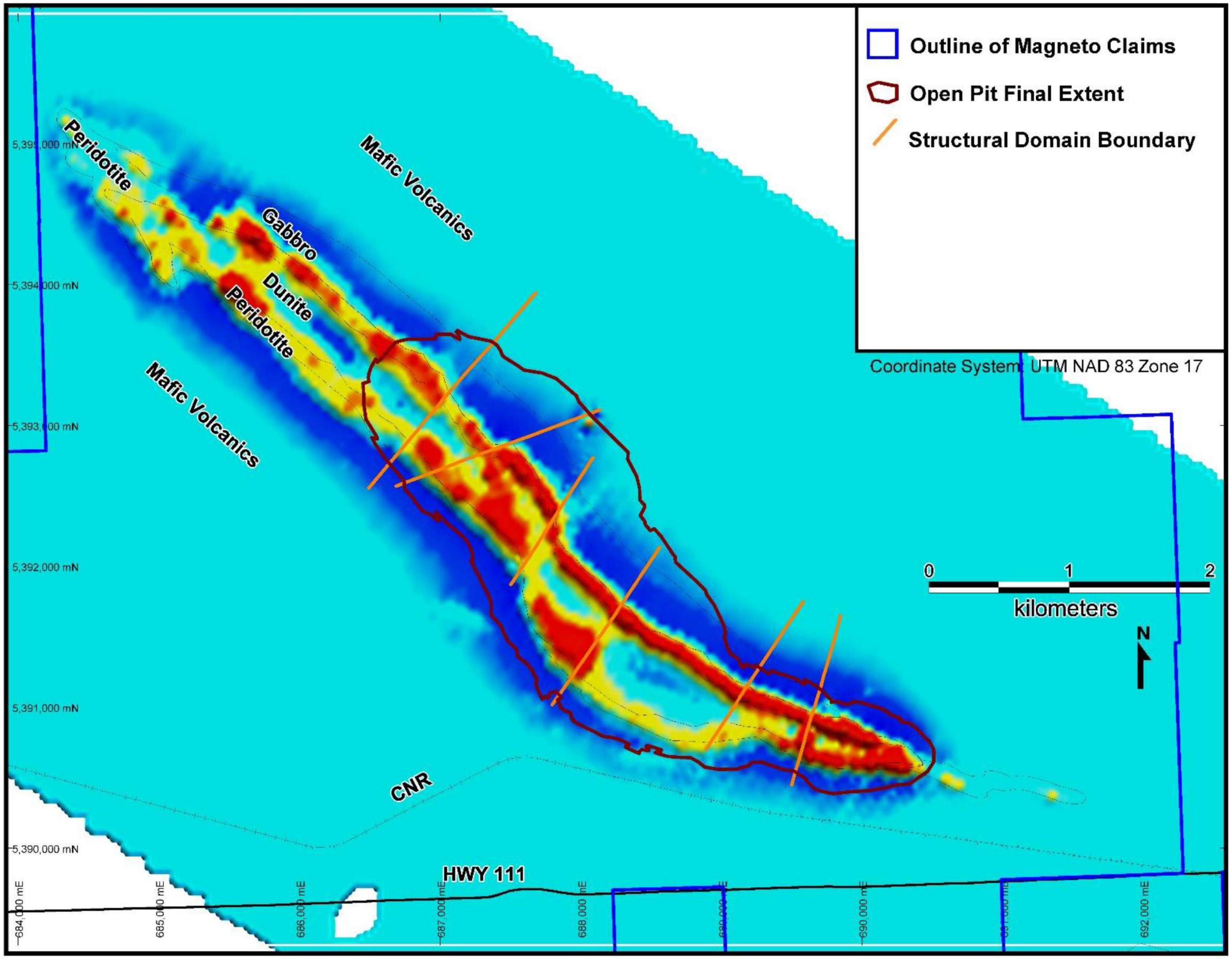
The property is covered by a layer of glacial overburden and muskeg. Mineralization subcrops approximately 30 m below the surface. Contacts between the Dumont sill and its host rocks have not been observed in outcrop but, in overall attitude, the body appears to be conformable to the layering of the 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

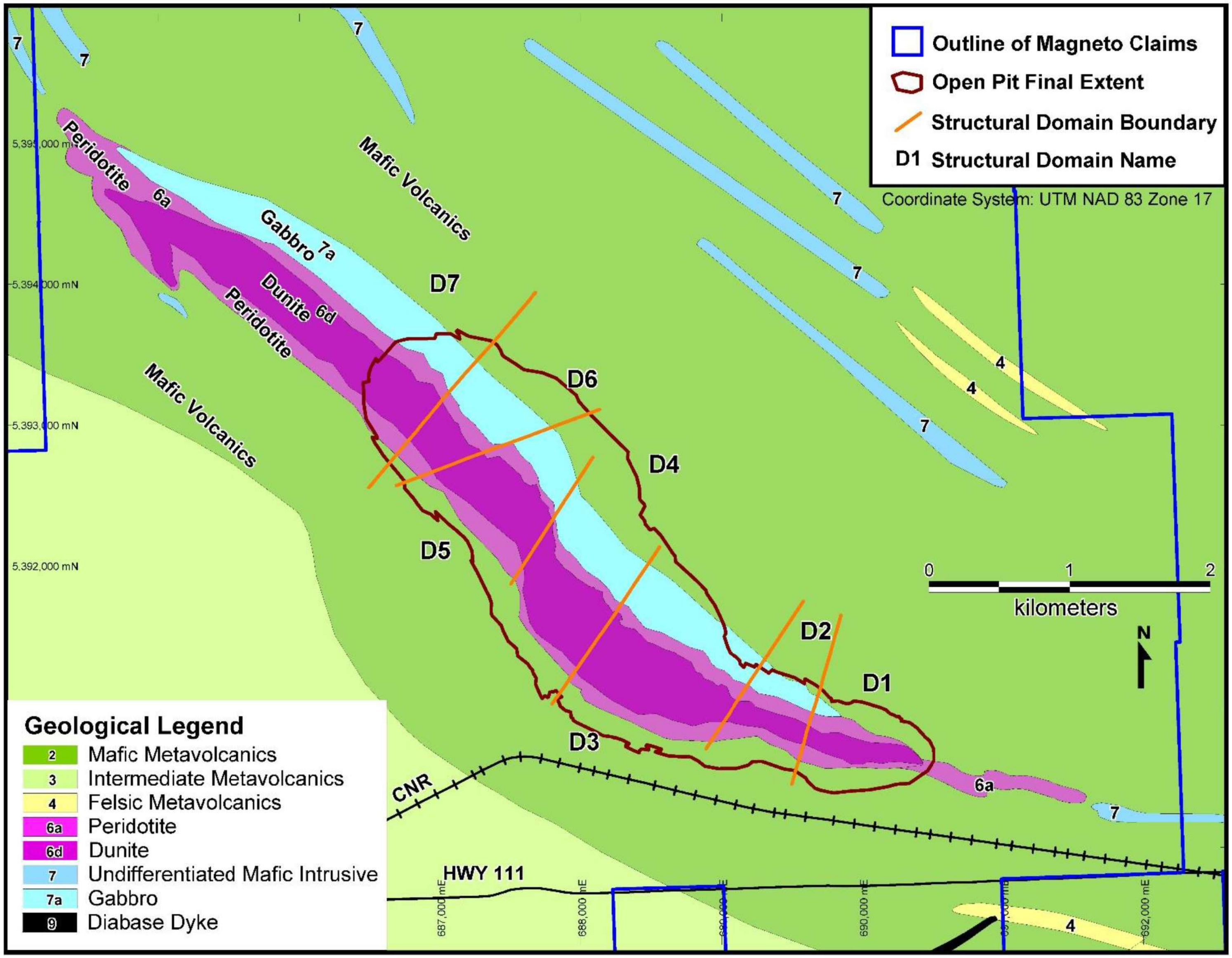
Based on the identified prominent northwest (NW) and northeast (NE) trending faults, the sill can be divided into structural blocks/domains. The true thickness of the upper mafic and lower ultramafic zone varies by location or fault block though the sill. The north-western end of the body has not been 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

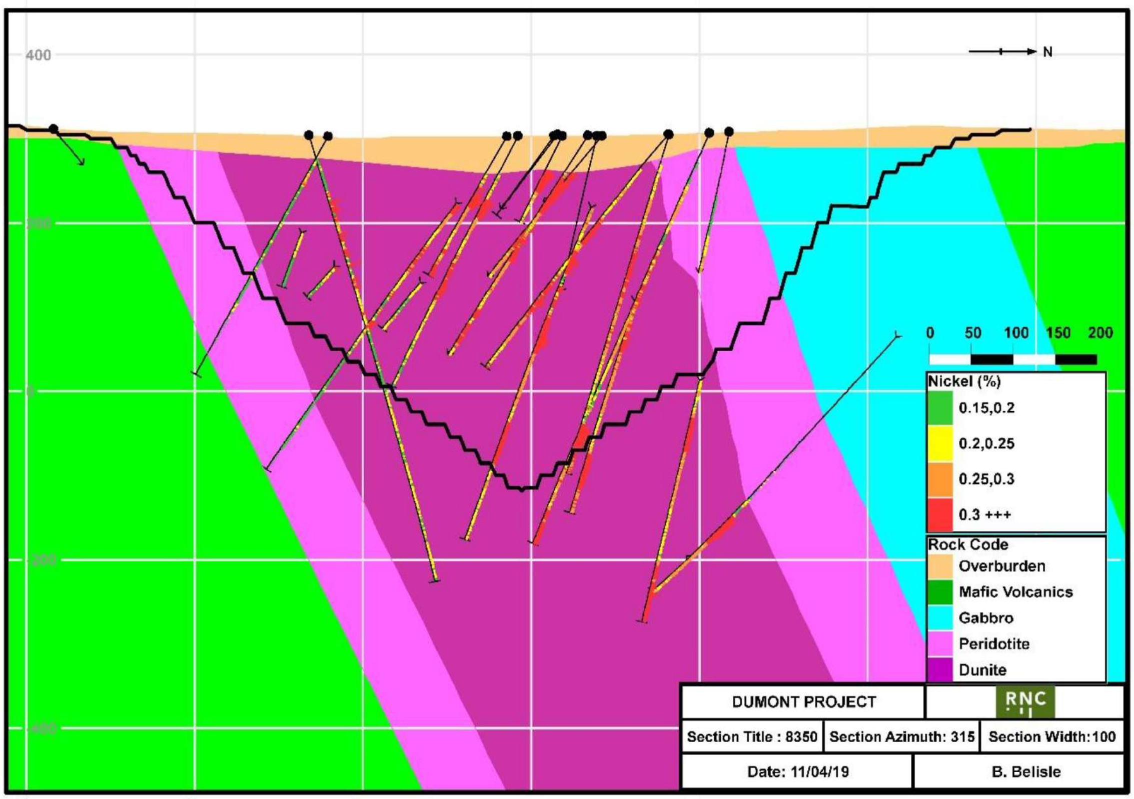
The ultramafic rocks have been serpentinized to varying degrees from partial to complete serpentinization. Along the basal contact of the sill (outside the resource envelope) serpentinization is frequently overprinted by varying degrees of talc-carbonate alteration. The predominant secondary assemblage 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).

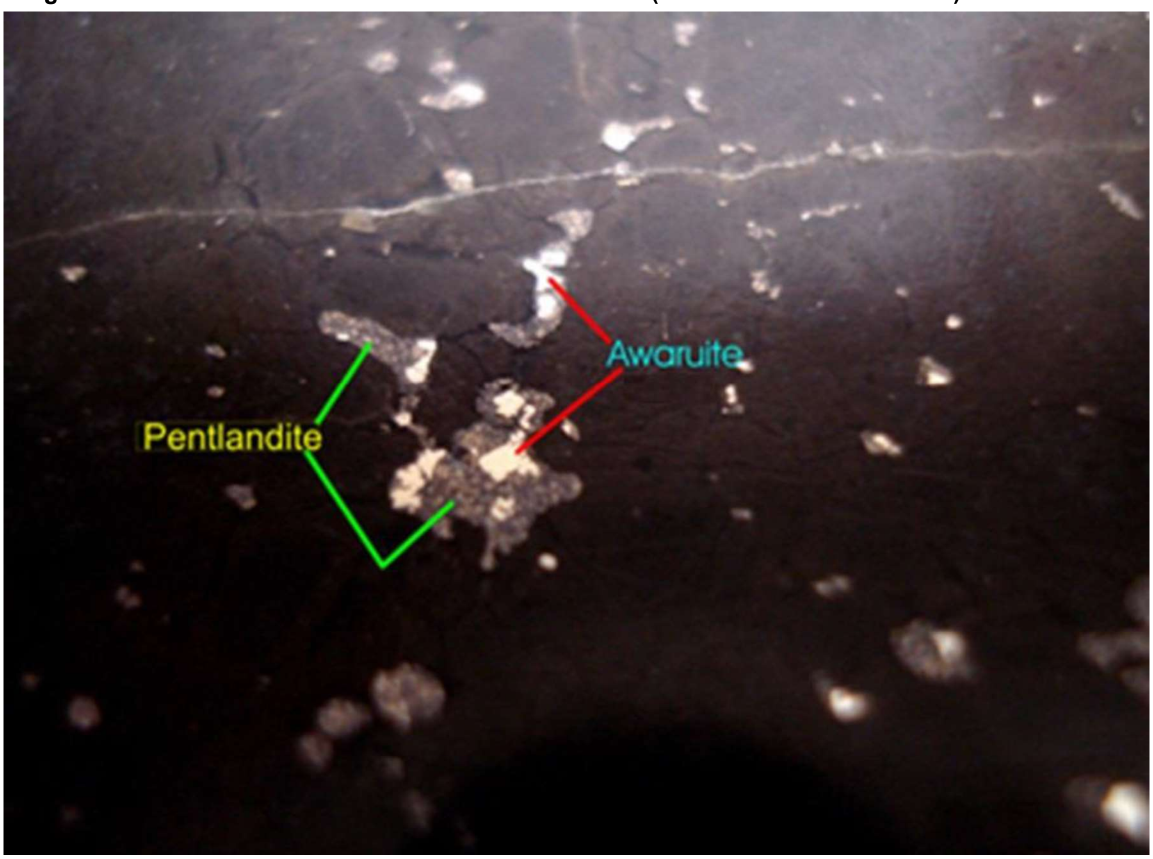
7.3 Disseminated Nickel Mineralization

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.

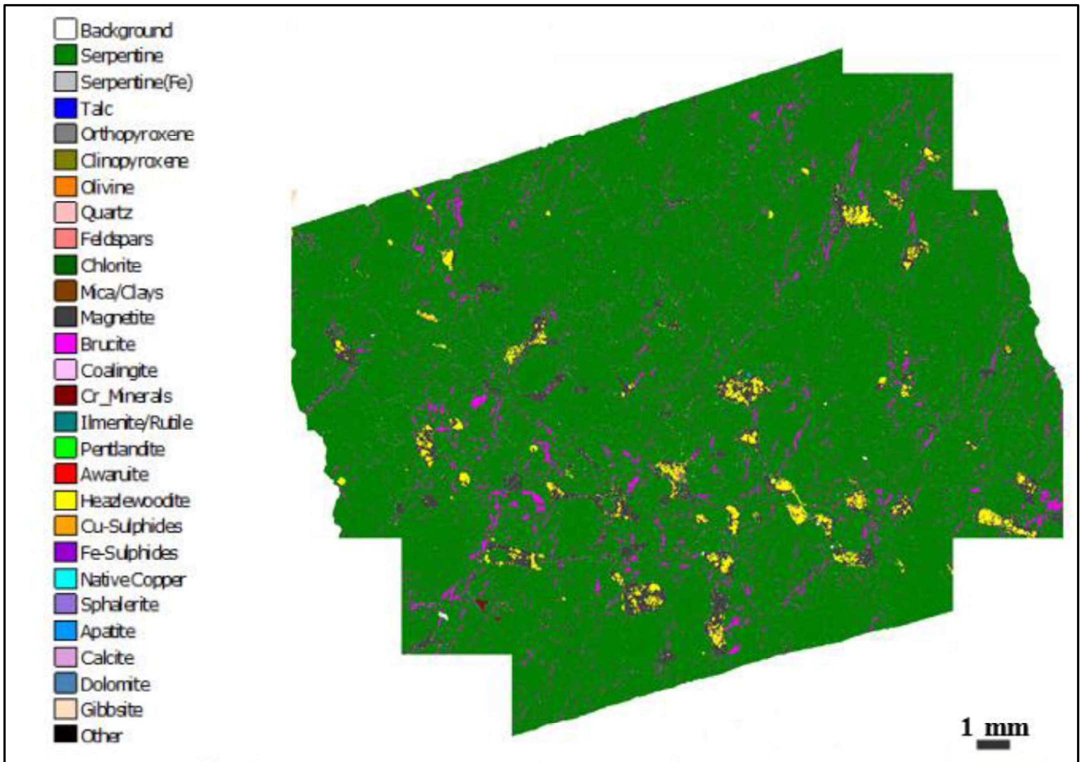
7.3.1.2 Sulphide Mineralization Assemblage

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 (Figure 7-6 and Figure 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.

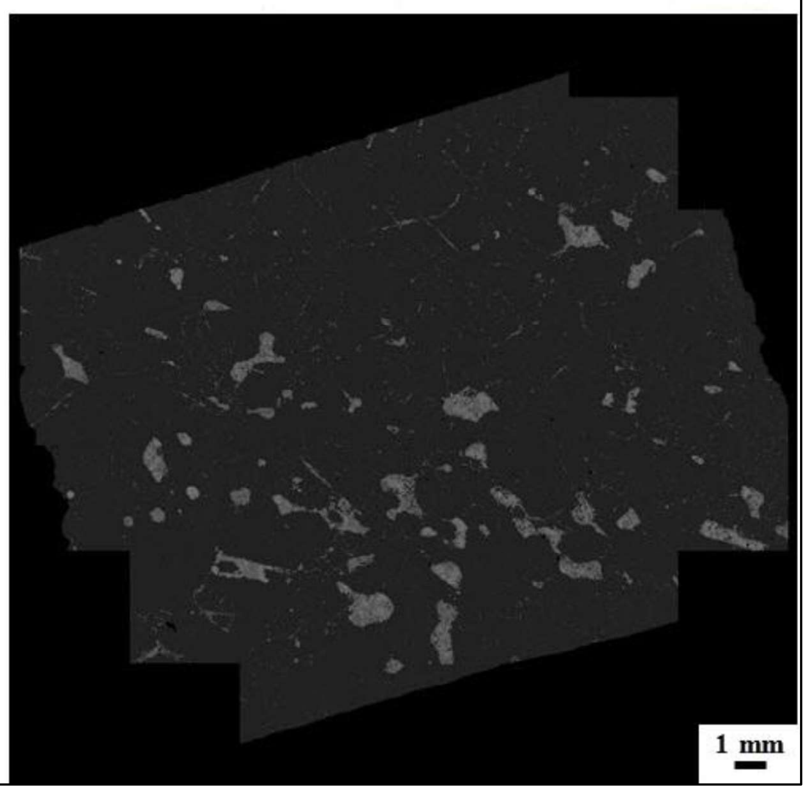
7.3.1.3 Alloy Mineralization Assemblage

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)

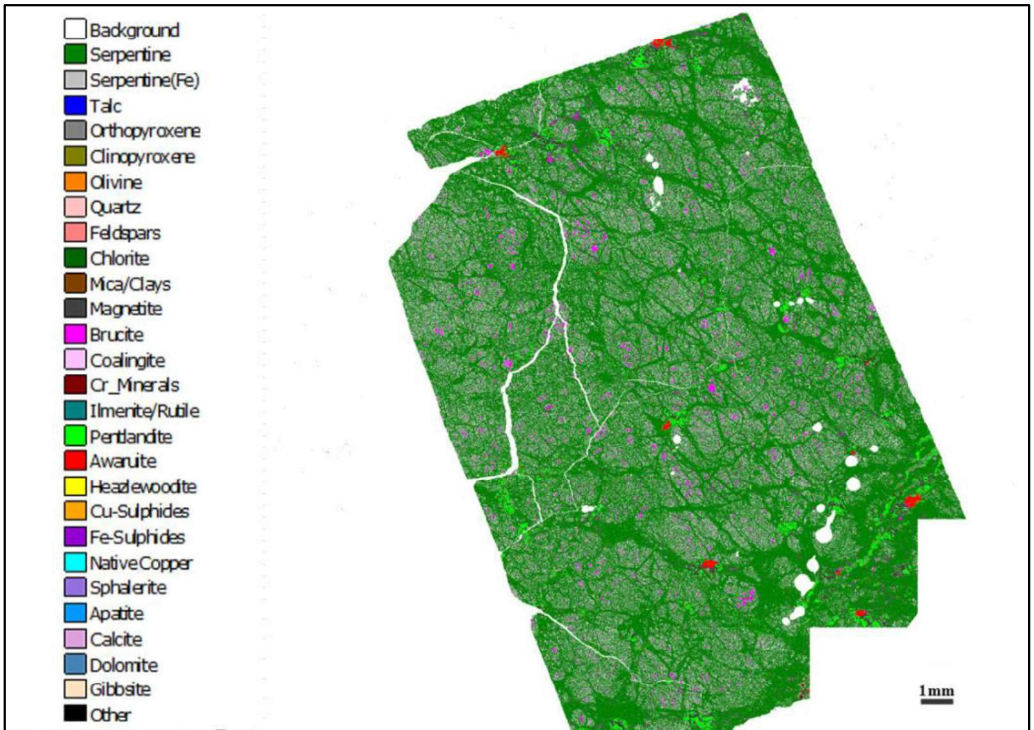


Modal Abundance	%
Serpentine (Total)	82.89
Fe-Serpentine	0.64
Talc	0.00
Orthopyroxene	0.00
Clinopyroxene	0.00
Olivine	0.04
Quartz	0.00
Feldspars	0.00
Chlorite	0.07
Mica/Clays	0.00
Magnetite	9.06
Brucite	4.65
Coalingite	0.08
Cr_Minerals	0.04
Ilmenite/Rutile	0.00
Pentlandite	0.19
Awaruite	0.02
Heazlewoodite	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

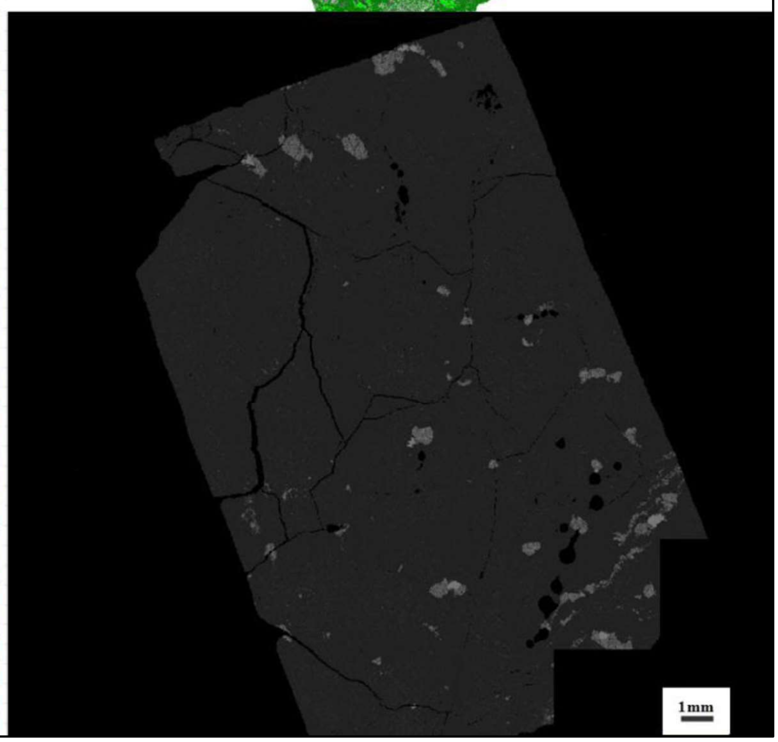


Note: Top: False-colour EXPLOMIN™ field stitch image. Bottom: Equivalent BSE image. (Heazlewoodite to pentlandite ratio 17.7). Modal Abundances as reported from EXPLOMIN™: 0.19% Pn, 2.92% Hz, 0.02% Aw, Metallic Ni 2.16% [(0.02%Aw*0.731%Ni) + (2.92%Hz*0.714Ni%) + (0.19%Pn*0.32%Ni)]. Sample contains coarse 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)

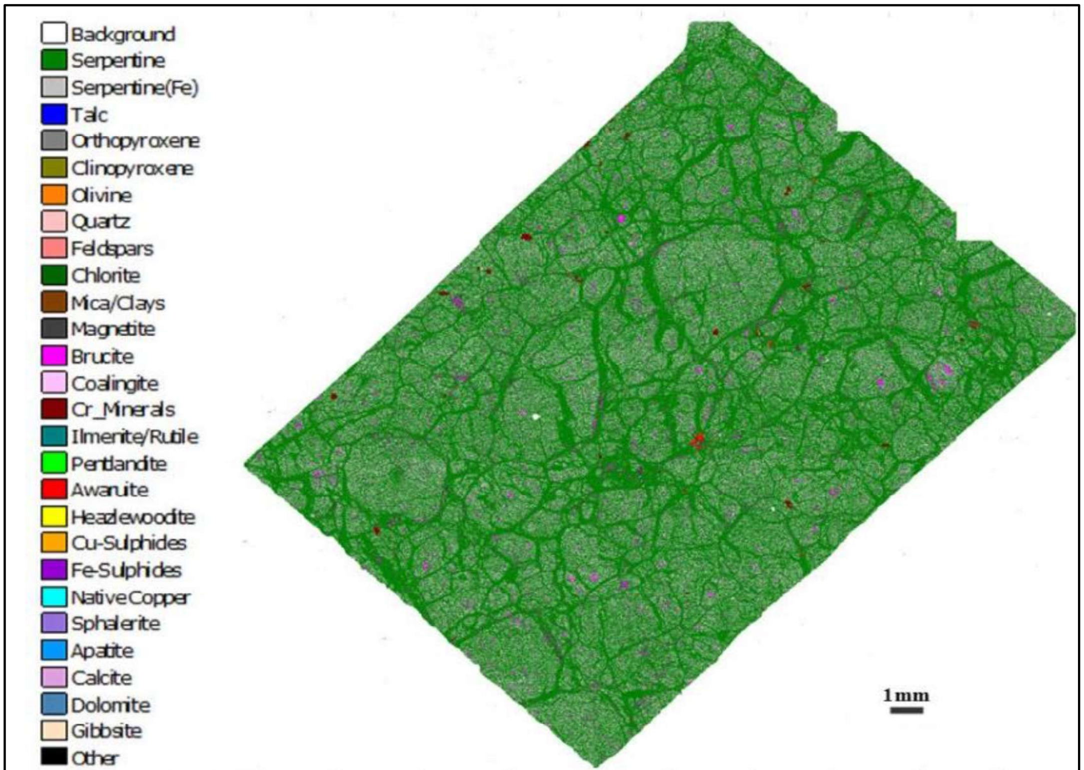


Modal Abundance	%
Serpentine (Total)	92.05
Fe-Serpentine	24.48
Talc	0.00
Orthopyroxene	0.00
Clinopyroxene	0.00
Olivine	0.00
Quartz	0.00
Feldspars	0.00
Chlorite	0.08
Mica/Clays	0.00
Magnetite	1.88
Brucite	2.15
Coalingite	0.40
Cr_Minerals	0.02
Ilmenite/Rutile	0.00
Pentlandite	2.70
Awaruite	0.68
Heazlewoodite	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

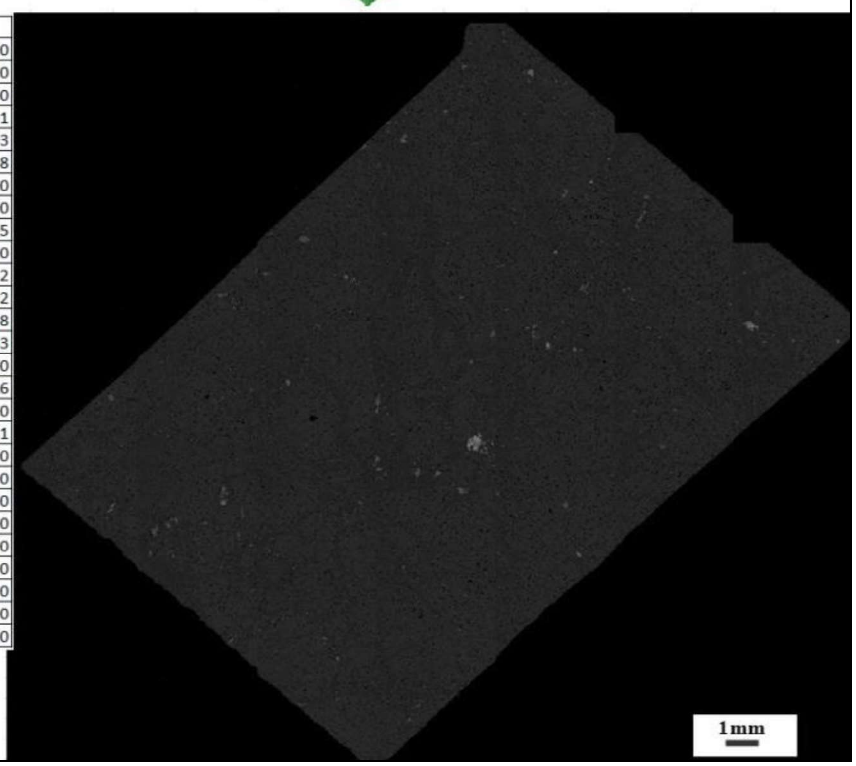


Note: Top: False colour EXPLOMIN™ field stitch image. Bottom: Equivalent BSE image. (Heazlewoodite to pentlandite ratio 0.003). Field Stitch Modal Abundances as reported from EXPLOMIN™: 2.7% Pn, 0.02% Hz, 0.68% Aw, Metallic Ni 1.38% [(0.68%Aw*0.731%Ni) + (0.02%Hz*0.714Ni%) + (2.7%Pn*0.32%Ni)]. Samples 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)

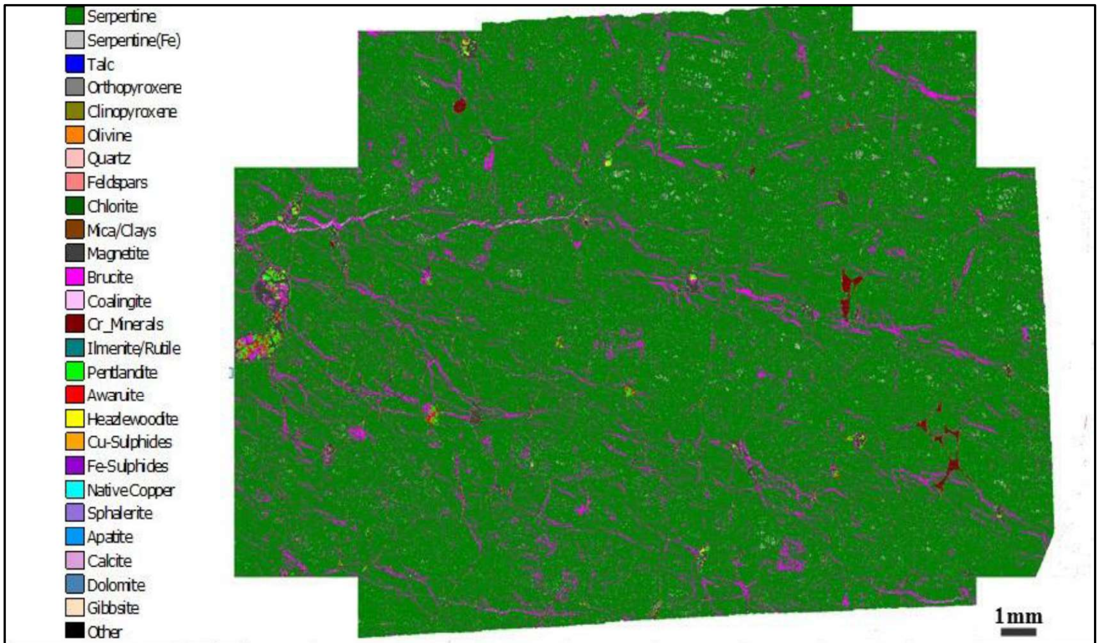


Modal Abundance	%
Serpentine (Total)	93.80
Fe-Serpentine	25.20
Talc	0.00
Orthopyroxene	0.01
Clinopyroxene	0.03
Olivine	0.58
Quartz	0.00
Feldspars	0.00
Chlorite	0.05
Mica/Clays	0.00
Magnetite	1.52
Brucite	3.02
Coalingite	0.38
Cr_Minerals	0.33
Ilmenite/Rutile	0.00
Pentlandite	0.06
Awaruite	0.20
Heazlewoodite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

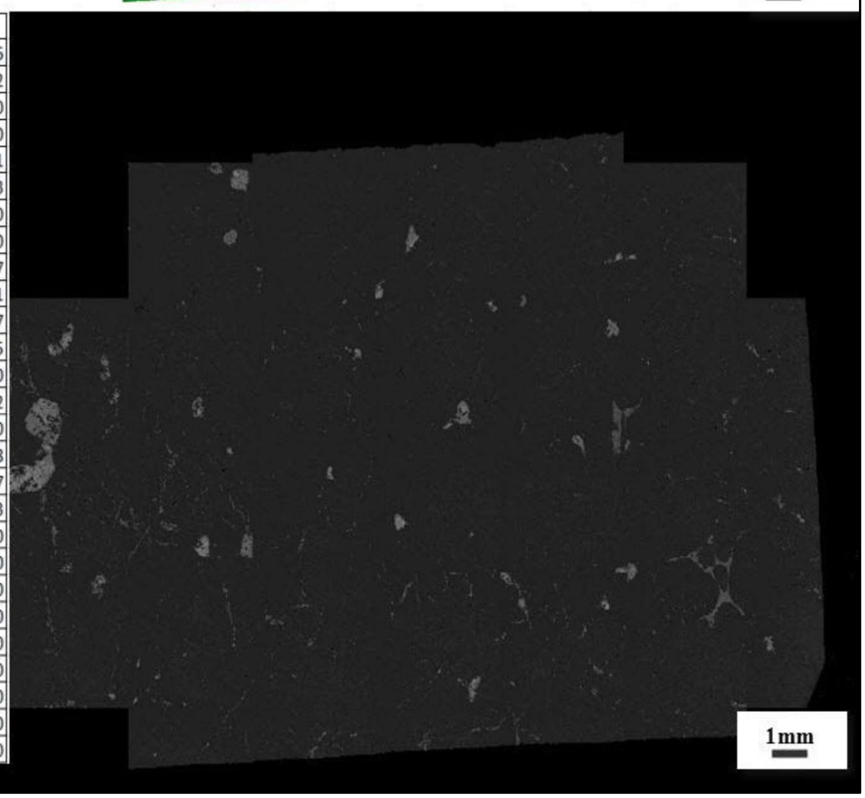


Note: Top: False colour EXPLOMIN^{TM} field stitch image. Bottom: Equivalent BSE image. Modal Abundances as reported from EXPLOMIN^{TM} :(0.06% Pn, 0.01% Hz, 0.20% Aw) Metallic Ni 0.17% [(0.2%Aw*0.731%Ni) + (0.01%Hz*0.714Ni%) + (0.06%Pn*0.32%Ni)]. Sample contains awaruite associated with magnetite and 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

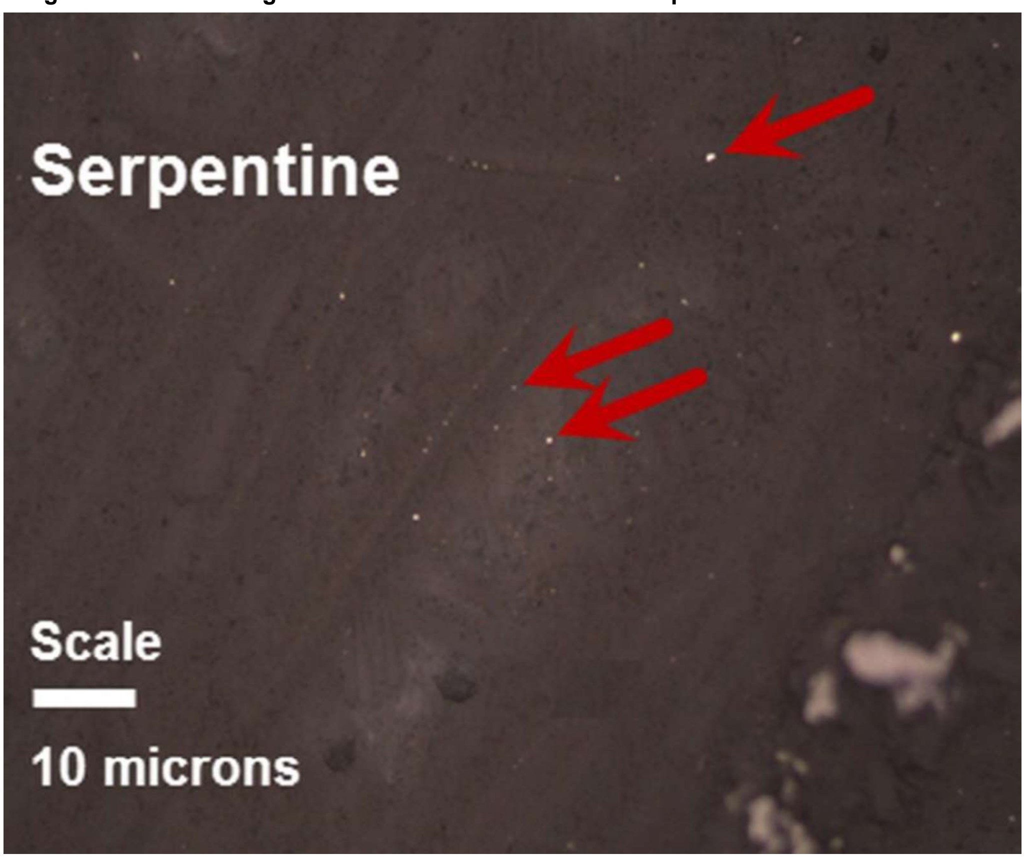
The mixed mineralization assemblage typically represents a transition from sulphide to alloy or sulphide (pentlandite) to sulphide (heazlewoodite) mineralization. The mixed mineralization assemblage contains varying amounts of sulphide (pentlandite and heazlewoodite) along with similar quantities of 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)

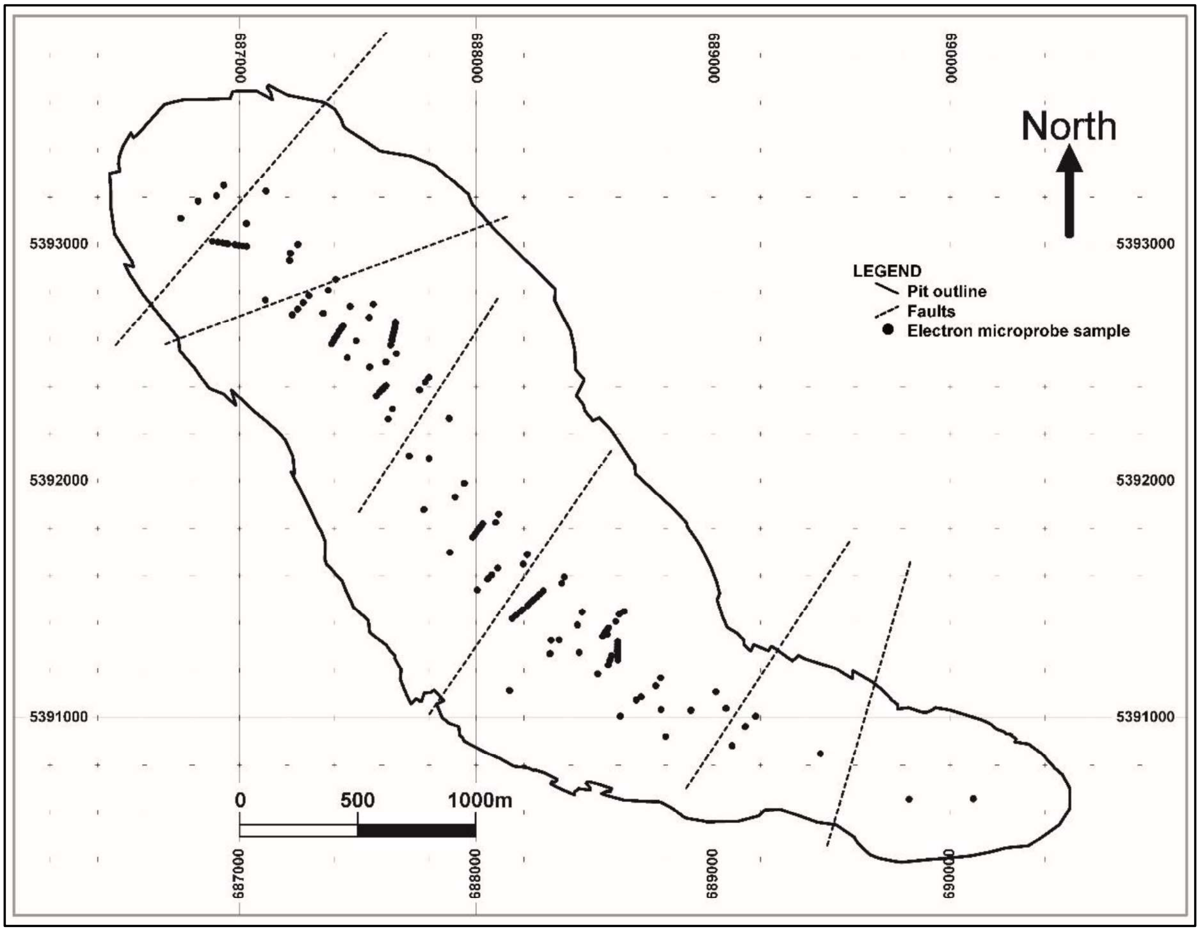
	All Samples in Domain		Sulphide Samples		Non-Sulphide Samples	
Domain	# Samples	Average Nickel in Silicates %	# Samples	Average Nickel in Silicates %	# Samples	Average Nickel in Silicates %
Heazlewoodite Dominant	521	37.3	124	15.54	397	44.06
Mixed Sulphide	162	34.1	64	16.4	98	45.8
Pentlandite Dominant	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 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 were 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

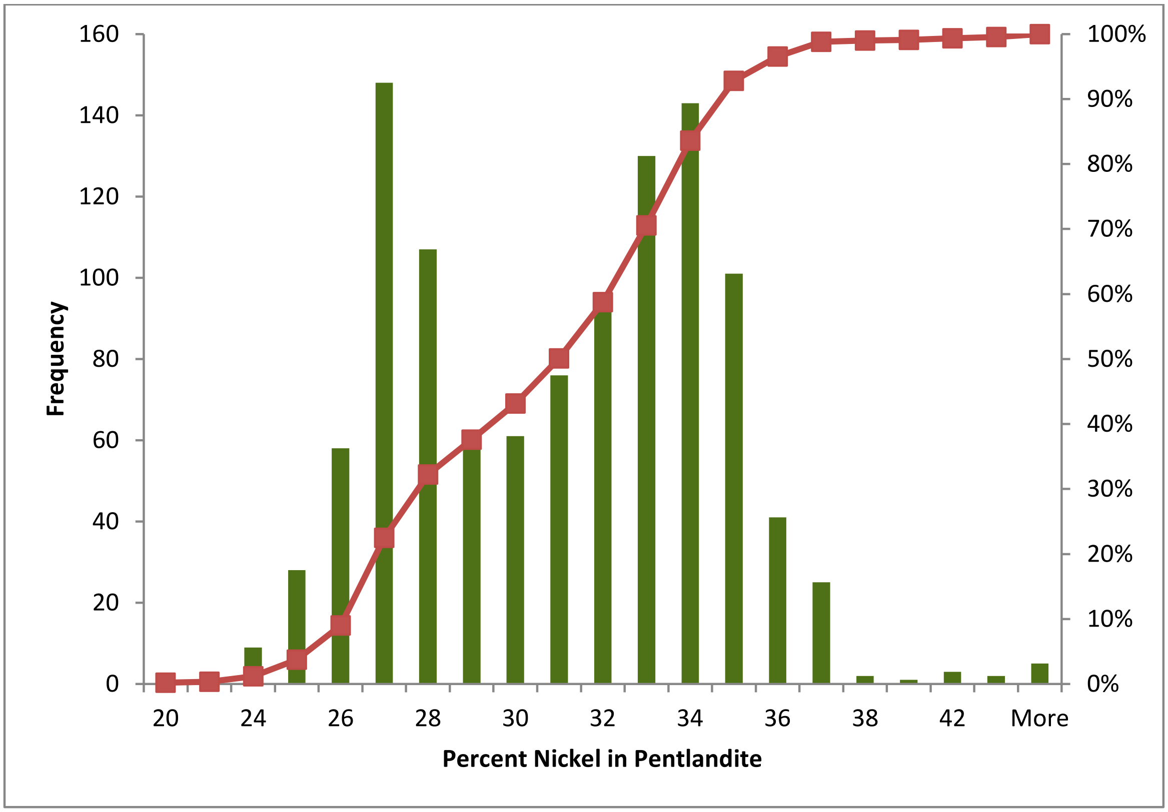
	Minimum Value (% Ni)	Maximum Value (% Ni)	Average (% Ni)	Number of Points	Standard Deviation	Number of Samples
Pentlandite	18.21	52.58	30.54	1103	3.65	117
Awaruite	59.03	89.86	72.85	699	3.10	118
Heazlewoodite	61.14	74.31	72.08	641	1.01	99
Olivine	0.124	0.4	0.29	131	0.06	7
Serpentine	0.00	1.31	0.13	917	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

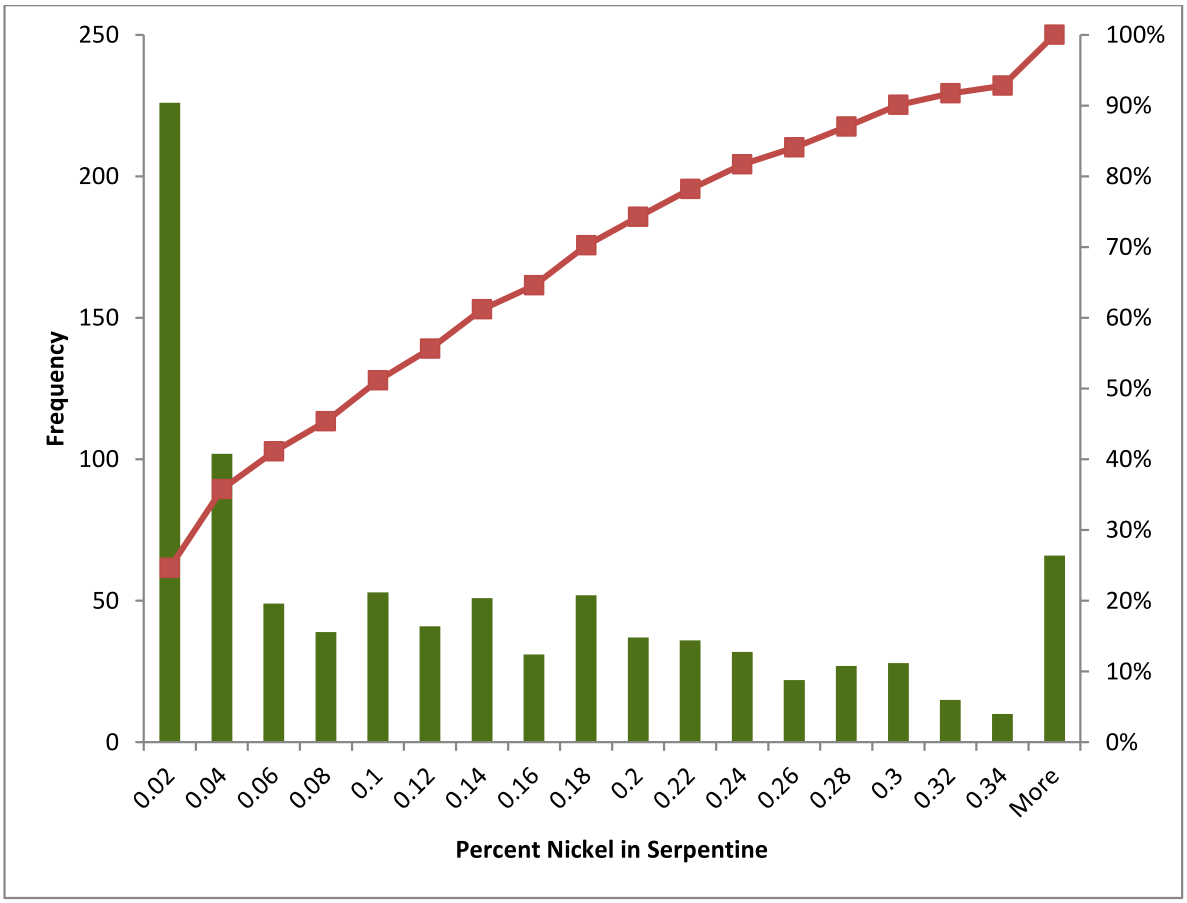
Source: RNC.

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 ($(Mg, Fe, Ni)_3Si_2O_5(OH)_4$). 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

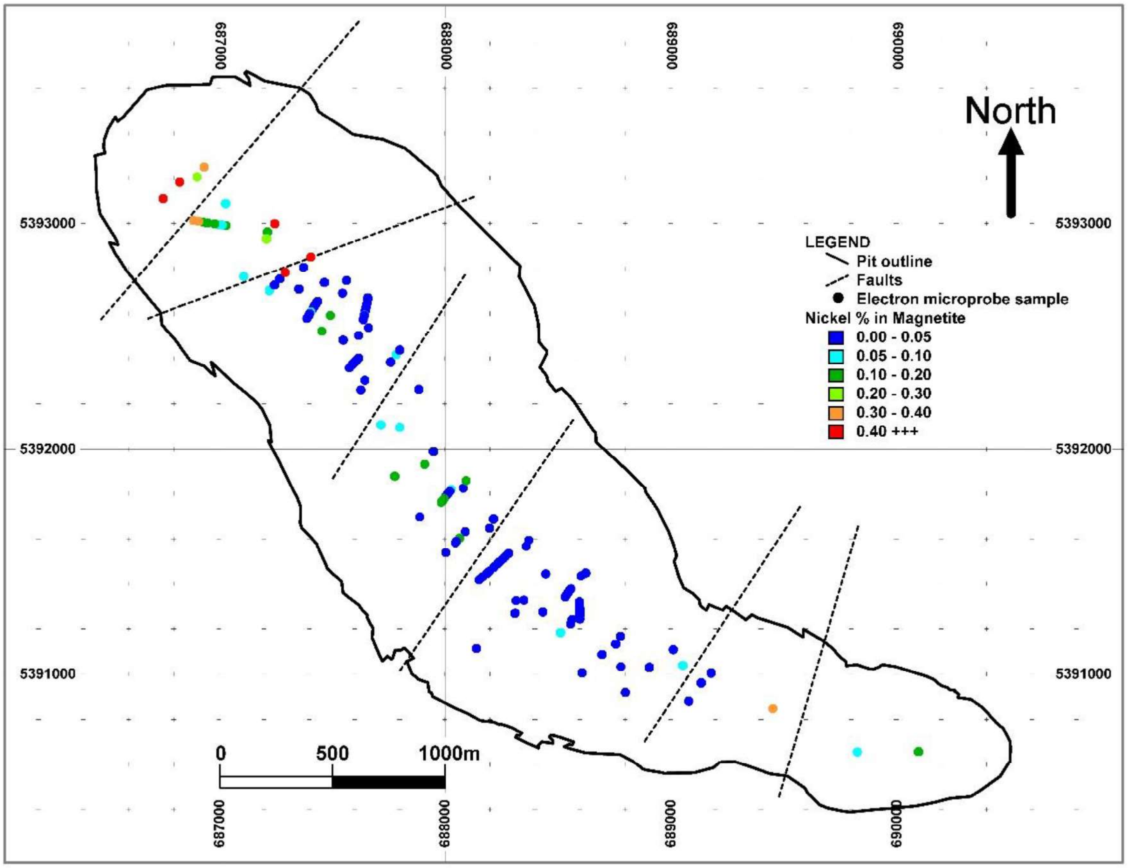


Note: 51% of data has less than 0.1% Ni in Serpentine. **Source:** RNC.

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.

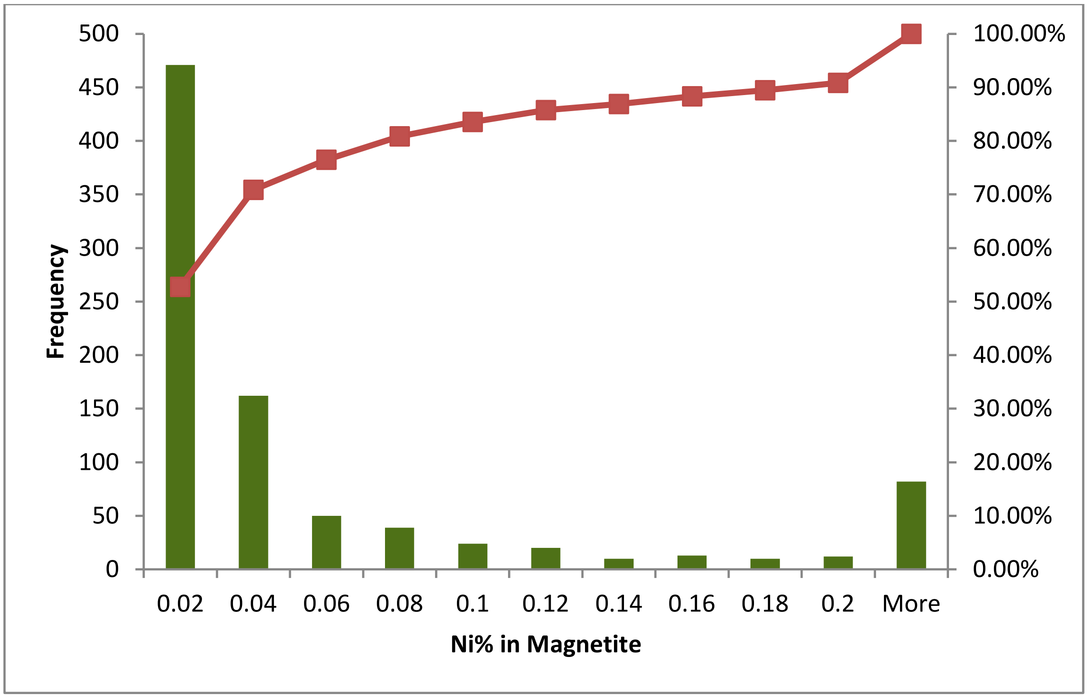
Table 7-4: Electron Microprobe Analyses for Magnetite

	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
St.Dev.	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.

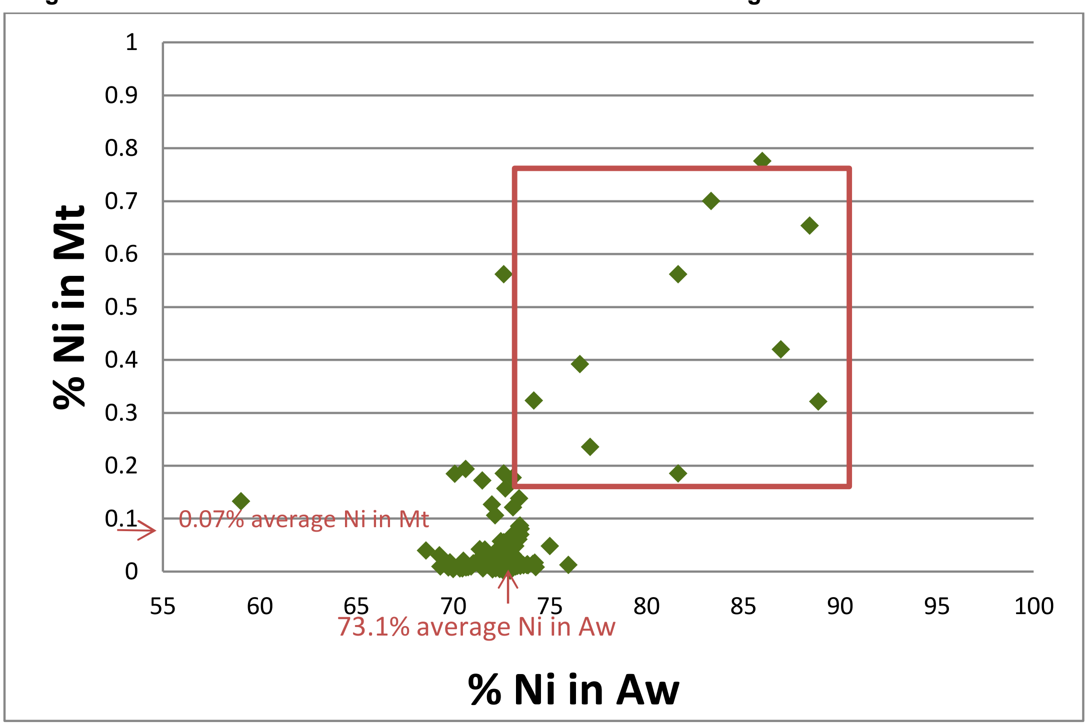
Magnetite on average contains 0.07% Ni by weight. 77% of 893 points analysed have values less than 0.06% (Figure 7-15). The group in Figure 7-15 with greater than 0.2% Ni in magnetite is associated with a zone containing Aw with higher than expected Ni (Figure 7-16).

Figure 7-15: Percent Nickel in Magnetite Distribution



Source: RNC.

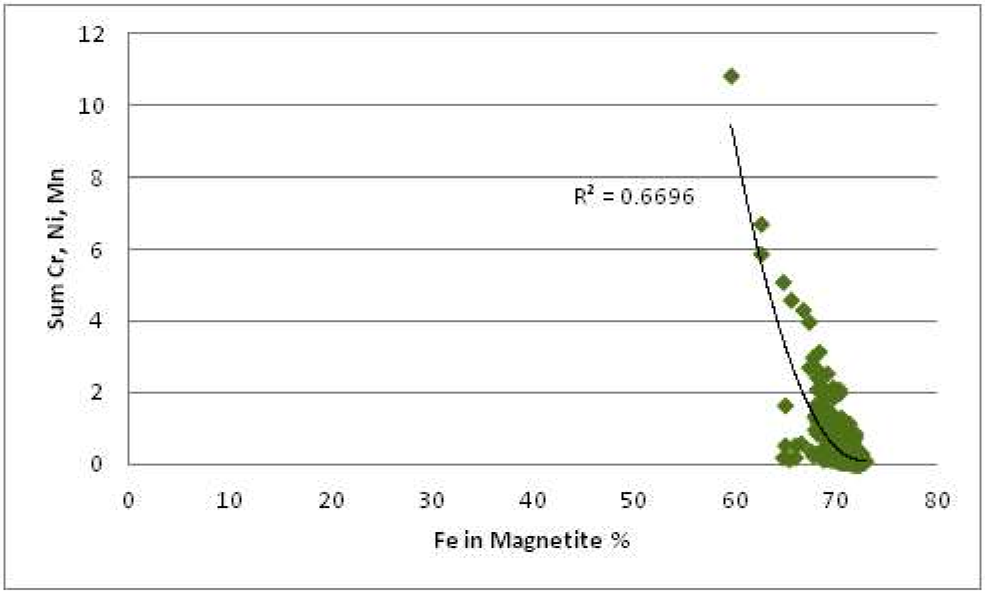
Figure 7-16: Percent Nickel in Awaruite vs. Percent Nickel in Magnetite



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/cm^3 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).

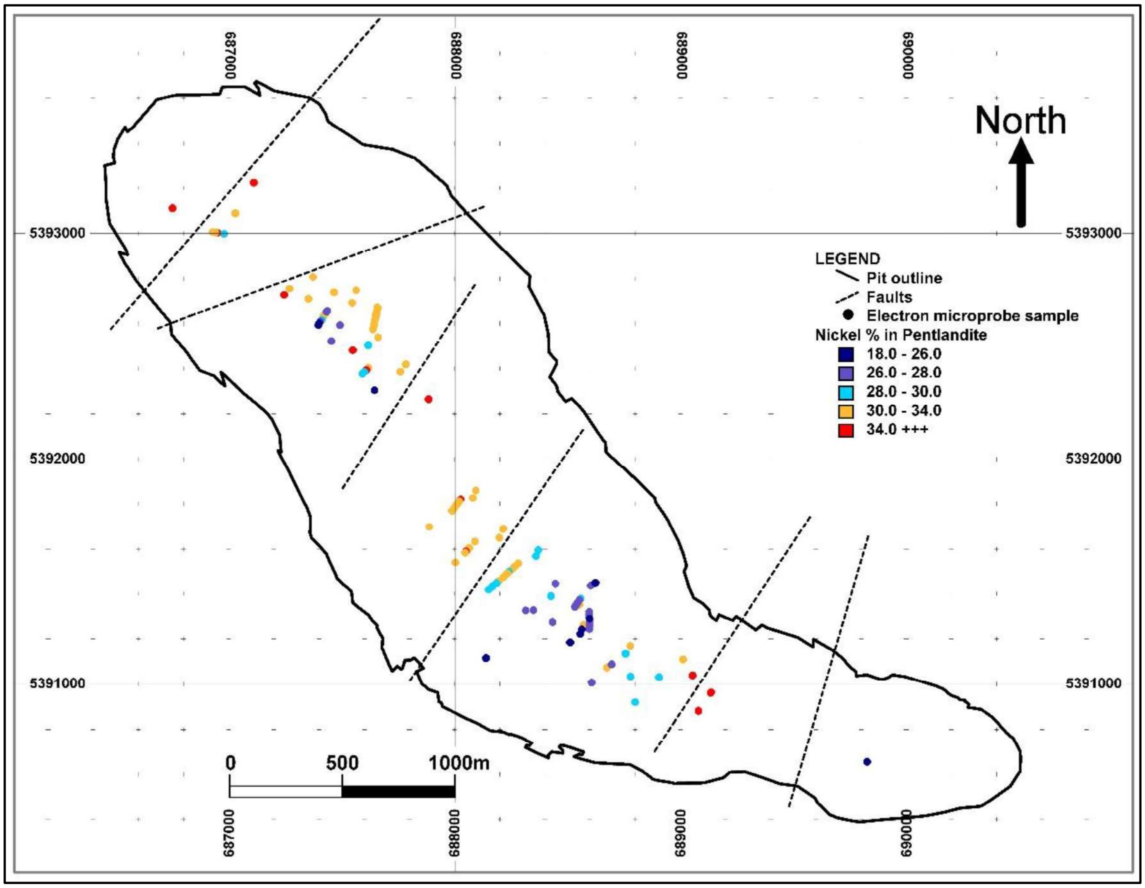
Pentlandite + Hydrogen + Water = Awaruite + Magnetite + Hydrogen Sulphide

$Ni_{4.5}Fe_{4.5}S_8 + 4H_2(aq) + 4H_2O = 1.5Ni_3Fe + Fe_3O_4 + 8H_2S(aq)$ **(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).

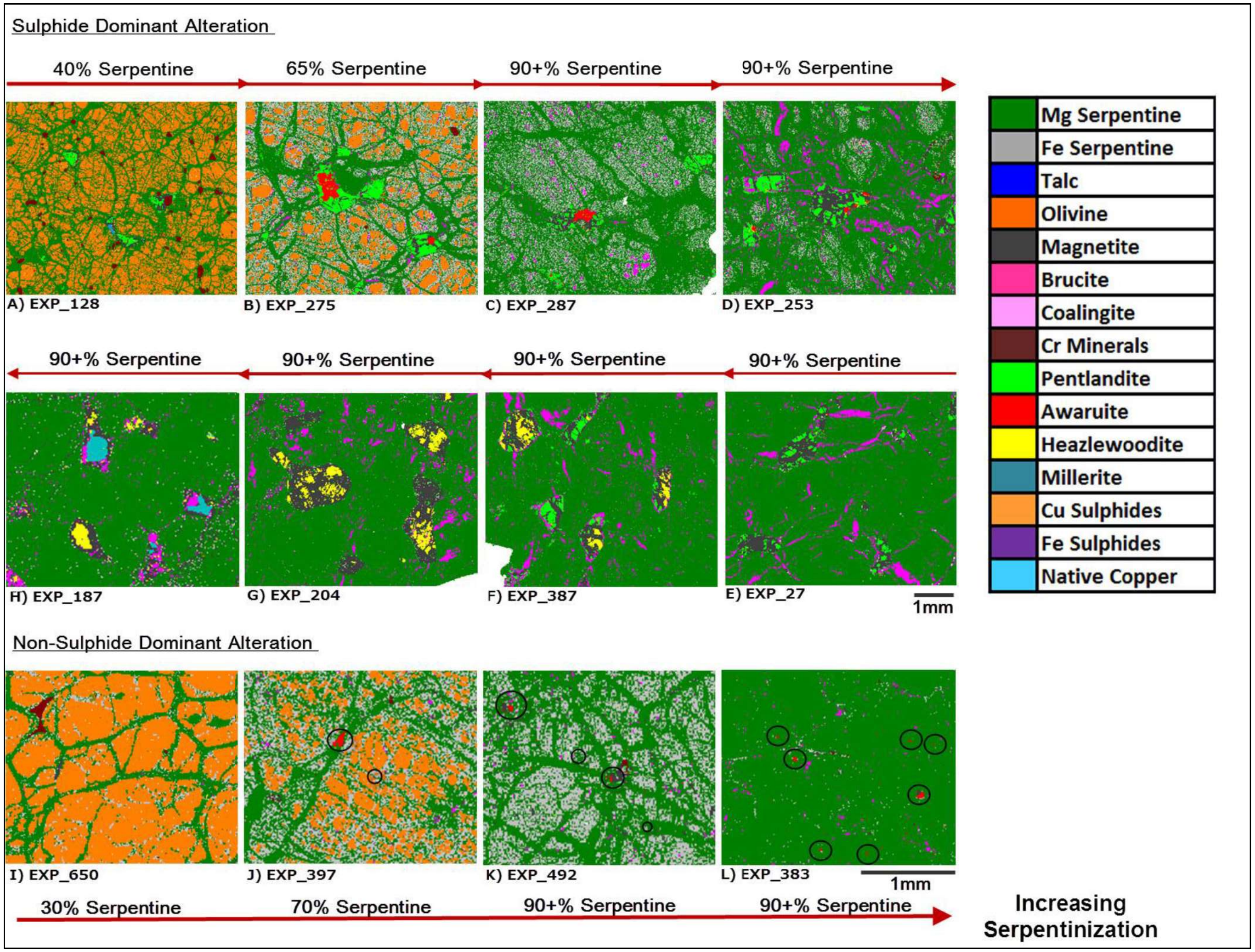
Figure 7-18: Distributions of Ni Tenor in Pentlandite



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 Figure 7-19 and Figure 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.

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

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

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

$3Ni_3Fe + 6H_2S(aq) + H_2O = Fe_3O_4 + 3Ni_3S_2 + 7H_2(aq))$ **(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 7-14 G). 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.

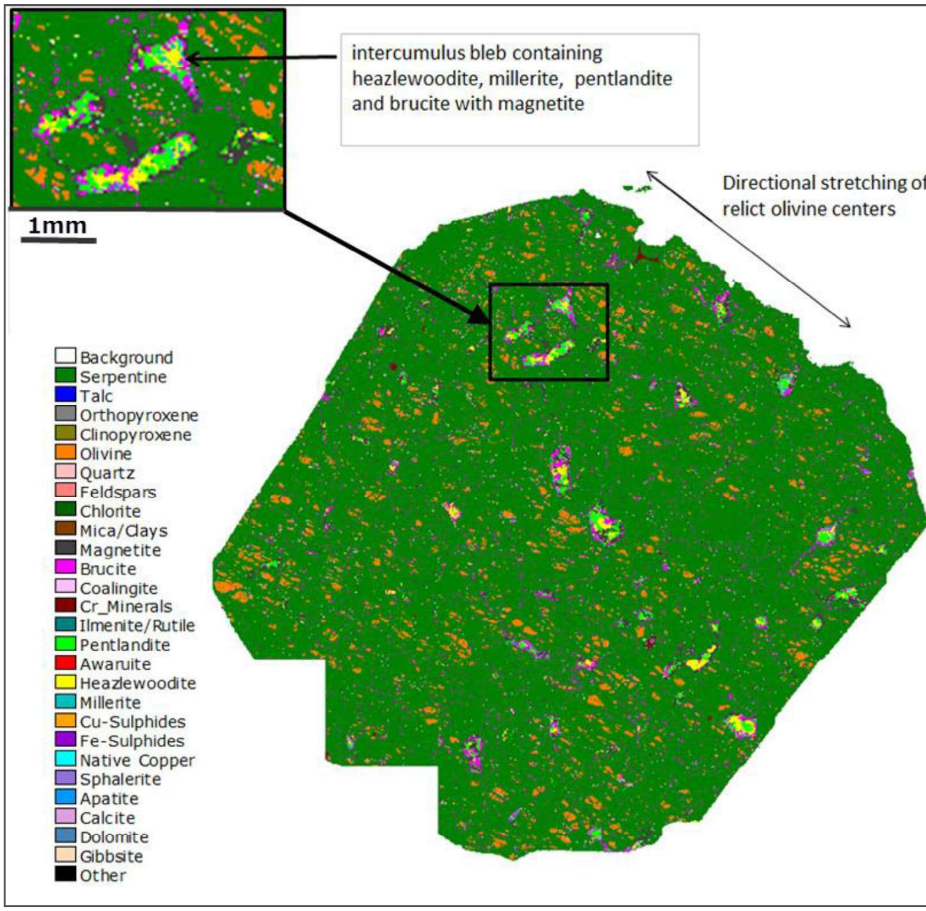
Heazlewoodite + Hydrogen Sulphide = Hydrogen + Millerite

$Ni_3S_2 + H_2S(aq) = H_2(aq)) + 3NiS$ **(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

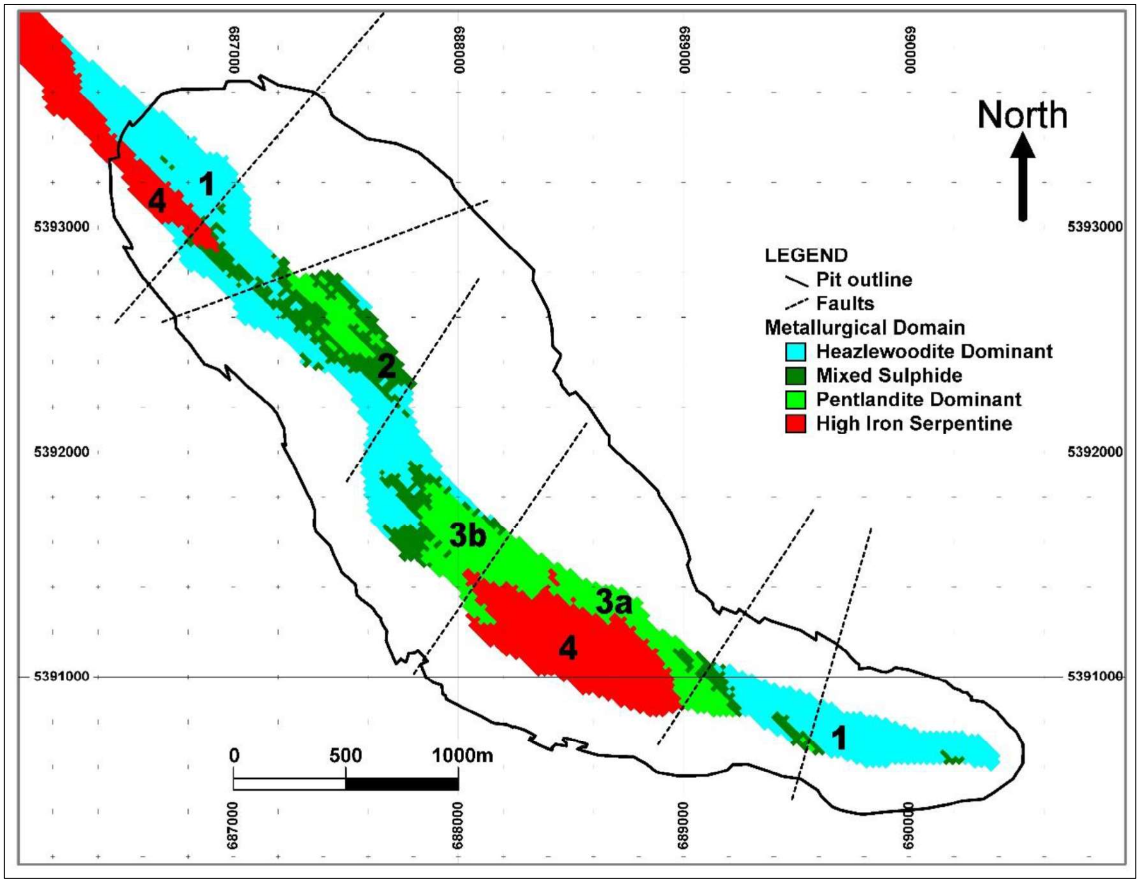


Serpentine	73.68
Serpentine(Fe)	3.96
Talc	0.00
Orthopyroxene	0.00
Clinopyroxene	0.01
Olivine	8.59
Quartz	0.00
Feldspars	0.00
Chlorite	0.06
Mica/Clays	0.00
Magnetite	5.90
Brucite	3.67
Coalingite	0.53
Cr_Minerals	0.14
Ilmenite/Rutile	0.00
Pentlandite	1.58
Awaruite	0.03
Heazlewoodite	1.47
Millerite	0.14
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™ field stitch image (EXP_214). (Heazlewoodite to pentlandite ratio 1.01). Modal Abundances as reported from EXPLOMIN™: 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.

Ausenco

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 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 correspond 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™ 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

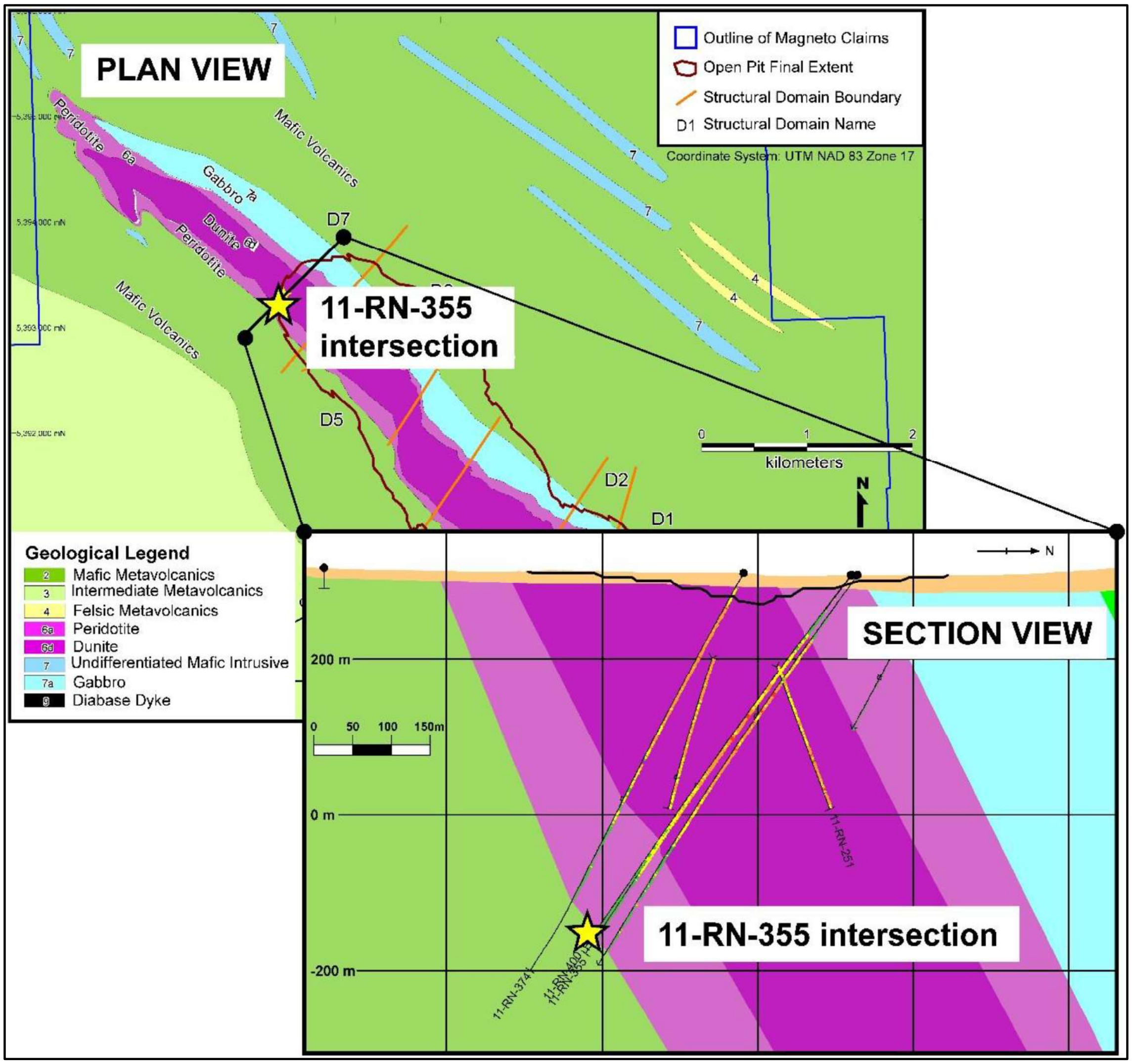
A hole drilled on section 5500E, passing through the Dumont intrusion and penetrating the footwall contact between the peridotite and the footwall mafic volcanic rock just to the northwest of the FS pit intersected a 1.25 m core-length of massive sulphide mineralization (Figure 7-22). The massive sulphide 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.

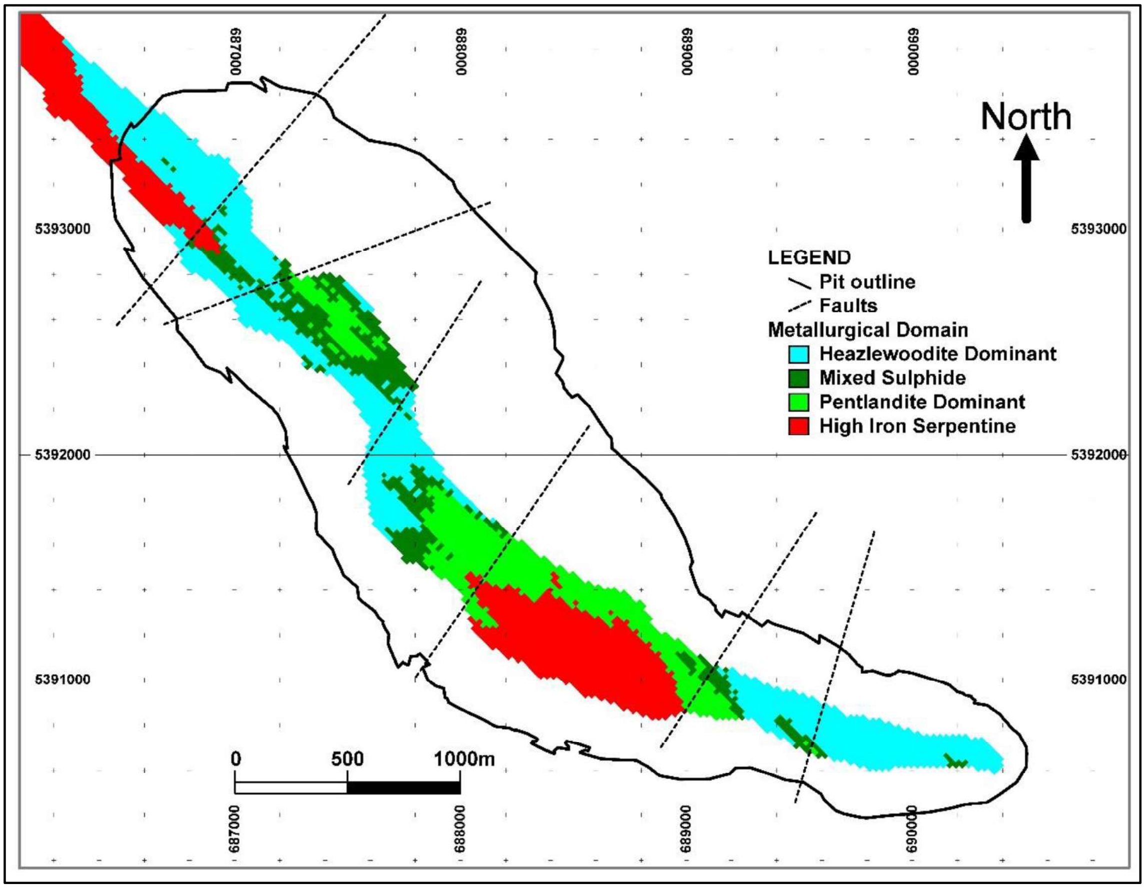
Pentlandite dominant samples (Hz/Pn<1, FESP<14) are most common making up a significant proportion of the reserve.

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. At Dumont, 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 formation of 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

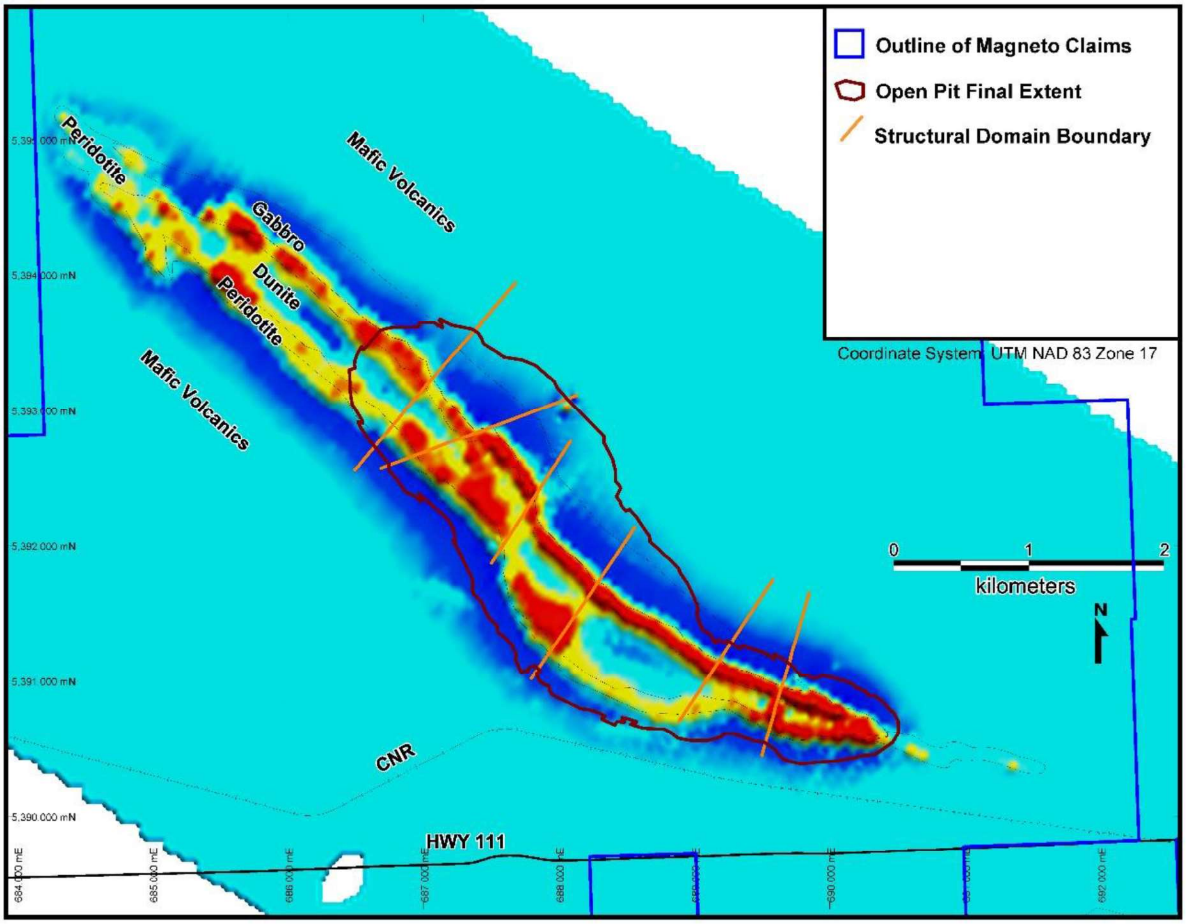
9.1.1 Airborne Geophysics

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 crosscut 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.

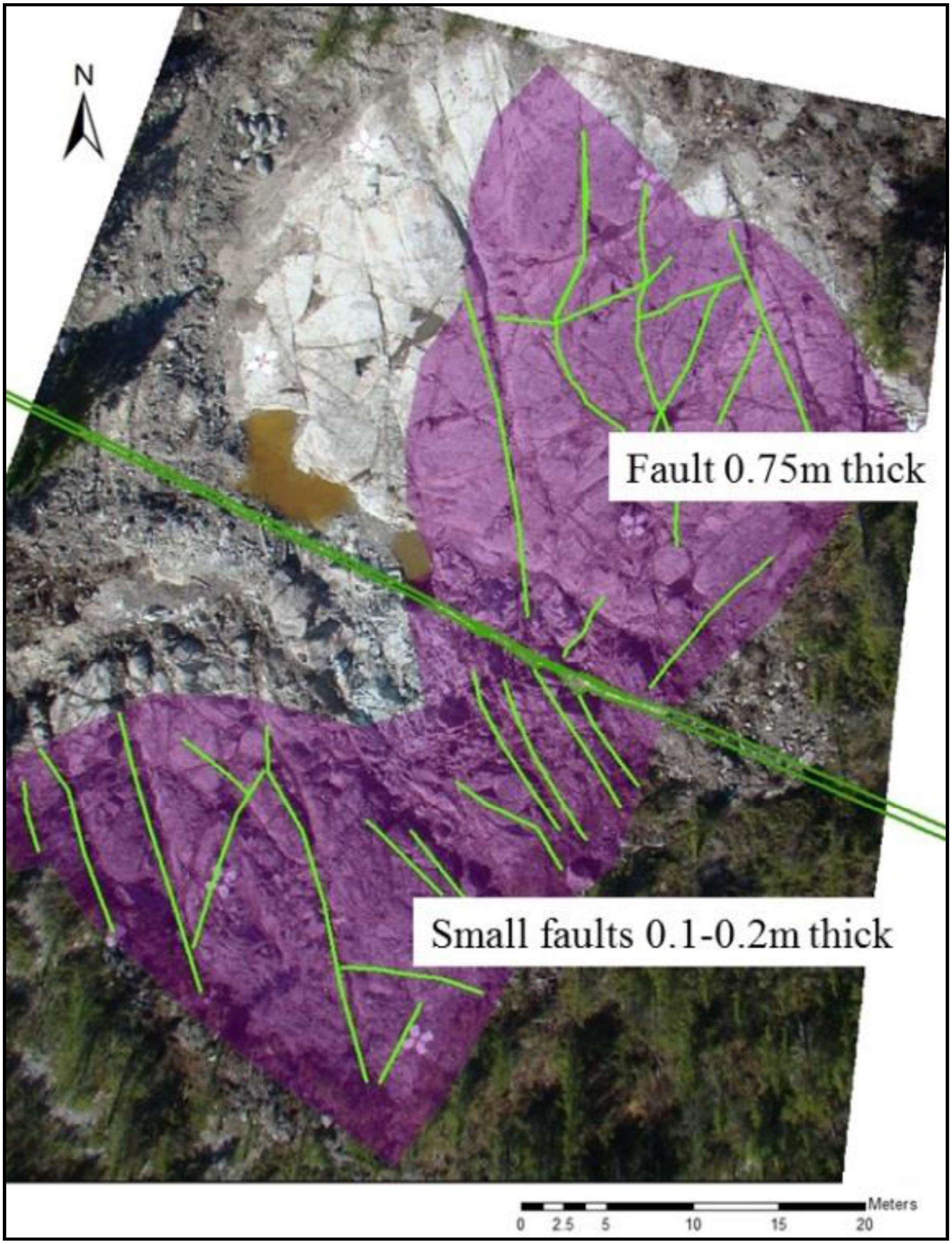
9.2 Geological Mapping

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 and to rule out the possibility of sterilizing potential mineral resources with infrastructure emplacements. 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



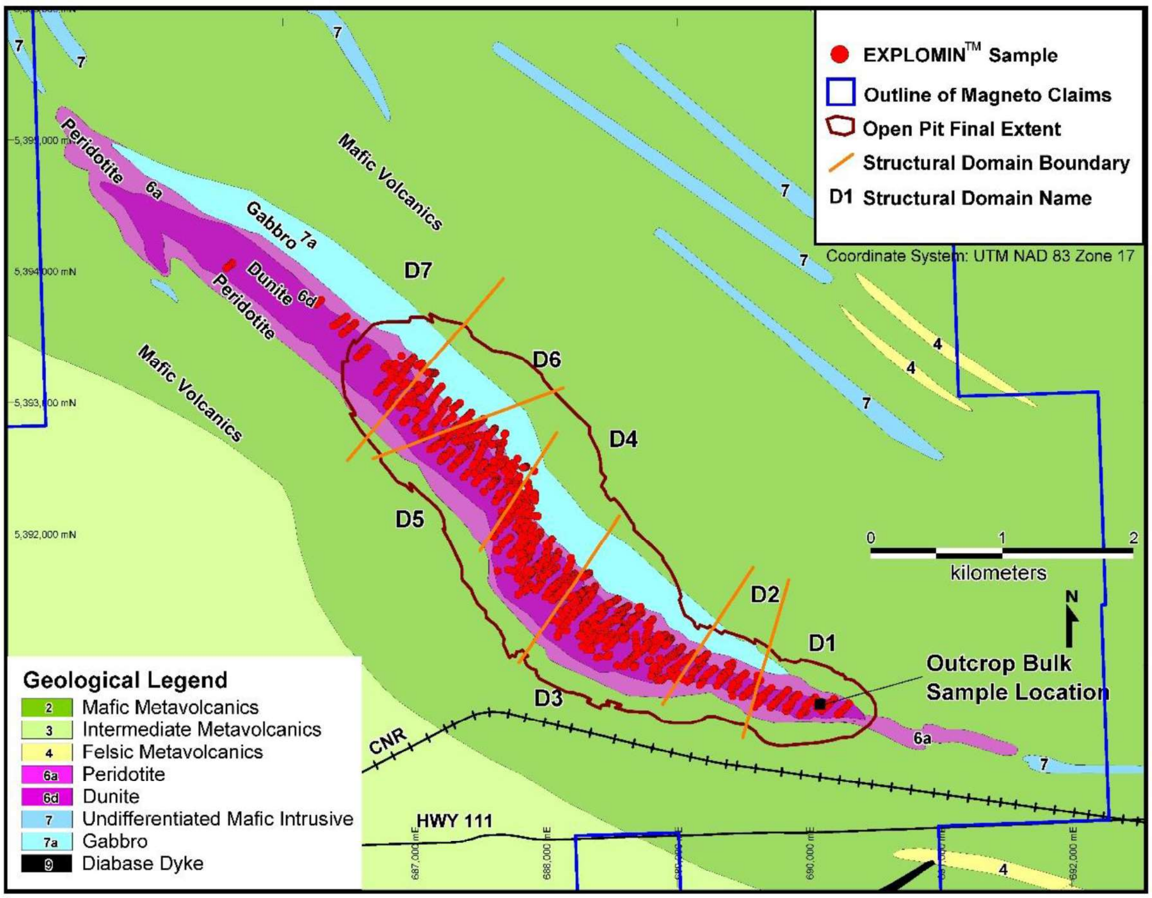
Source: Itasca Consulting. Note that the image is best fit, and not a true orthophoto, therefore there are mismatches.

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.

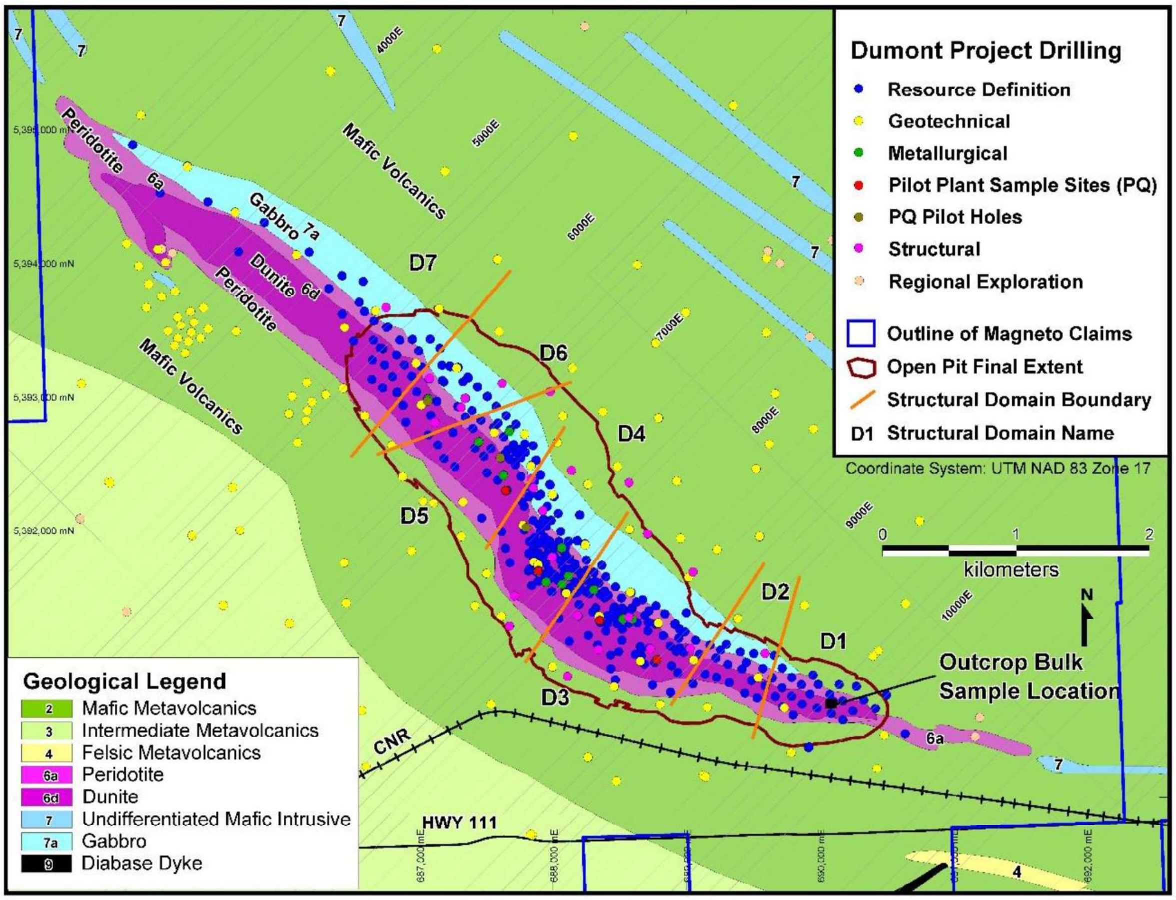
Metallurgical domain composite characterization samples were selected on an ongoing basis to represent the mineralogy of each metallurgical domain composite as defined for test work. This includes all domain composites described in Section 13, as well as all metallurgical composites defined in the mini 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.

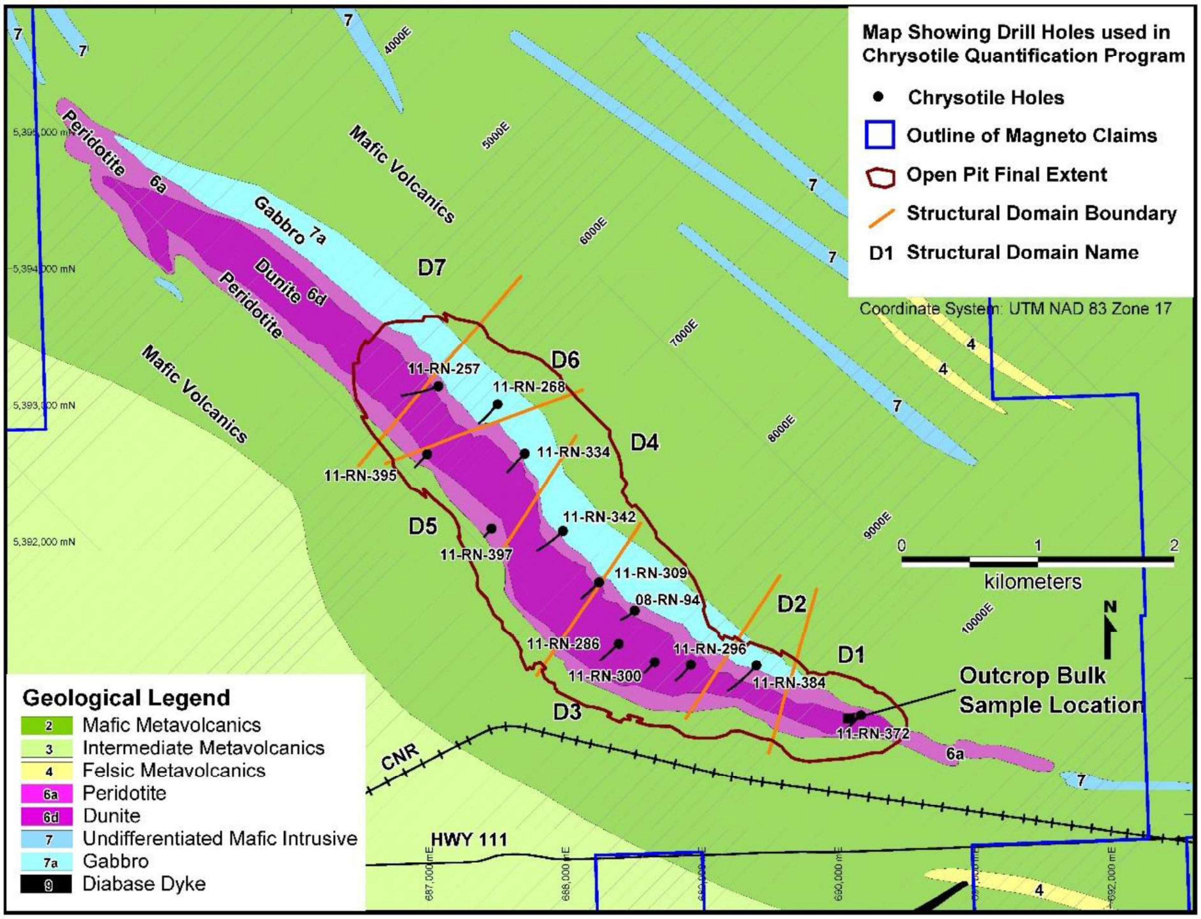
A section of the outcrop measuring approximately 40 m x 55 m was cleared of glacial overburden with an excavator and power washed. A smaller area within this was identified for sampling and subsequently drilled and blasted to a depth of approximately 1.5 m.

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 Table 9-1 and Table 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	95th 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.

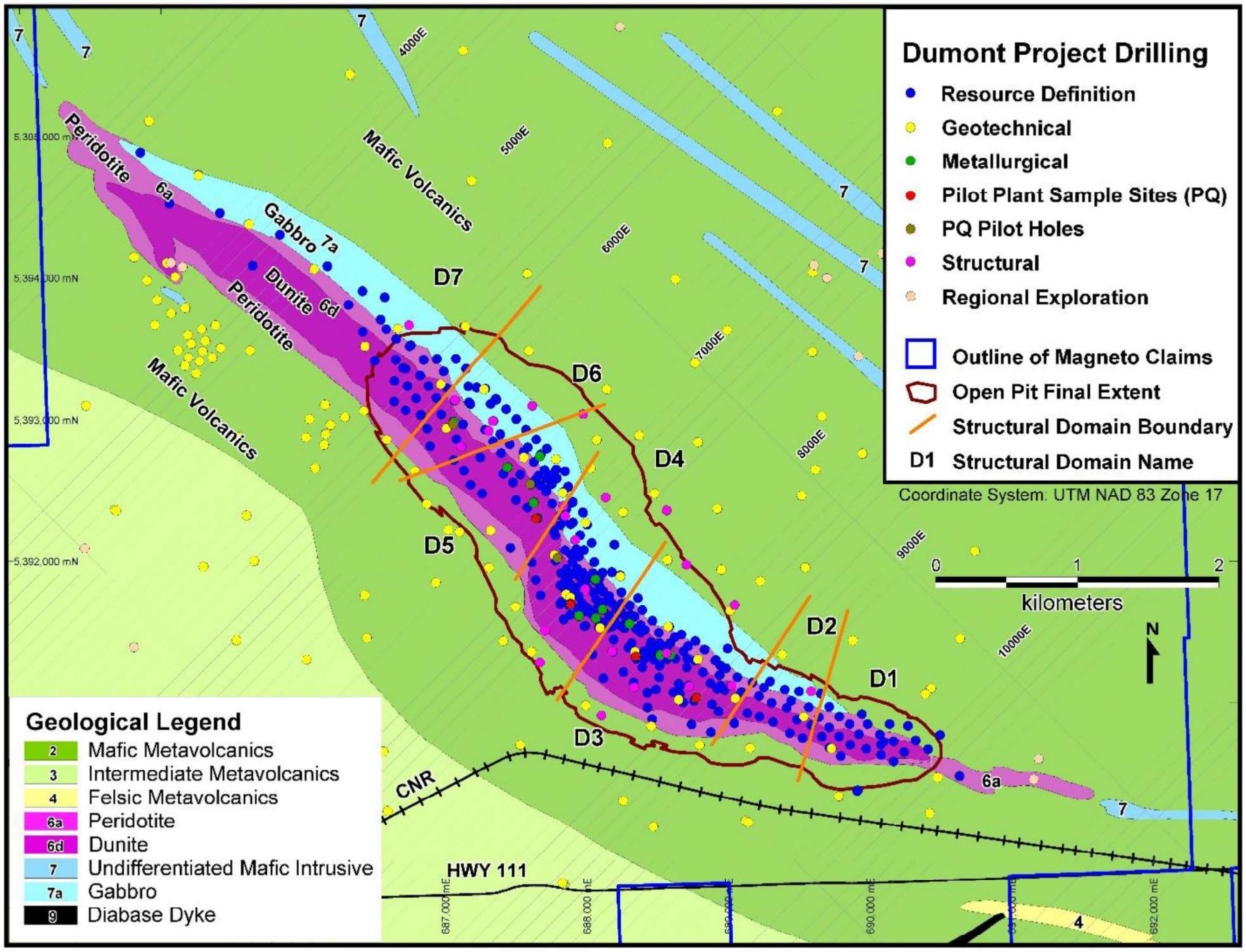
All geological, engineering and supervision portions of the drilling program were overseen by geological staff of RNC, principally Mr. John Korczak, P.Geo. (former employee), Mr. Lorne Burden, P.Geo. (former employee), and Mr. Robert Cloutier, Geo., OGQ supervised by Mr. Alger St-Jean, P.Geo., Vice-President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

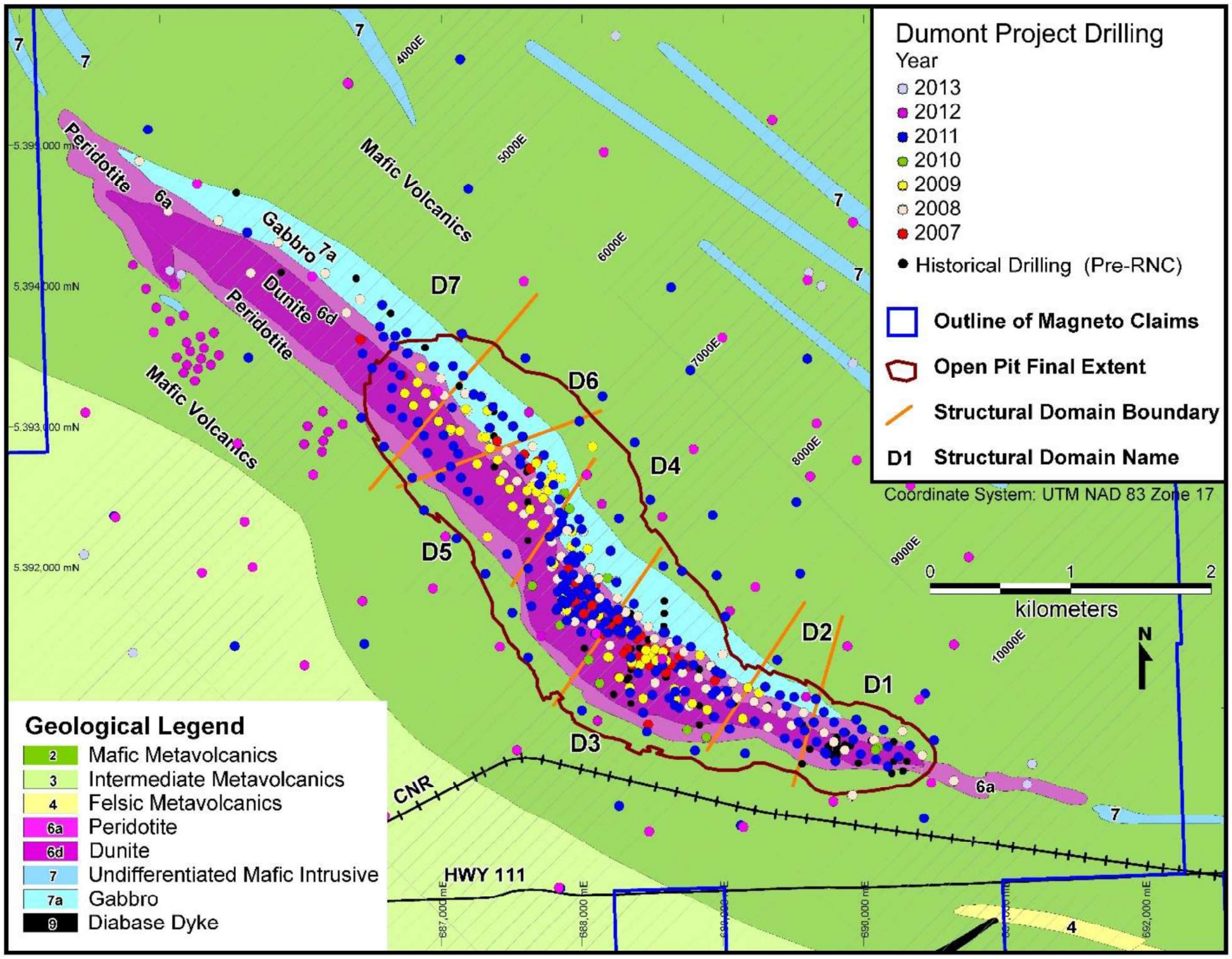
Source: RNC.

Figure 10-1: Location of Drill Holes on the Dumont Property



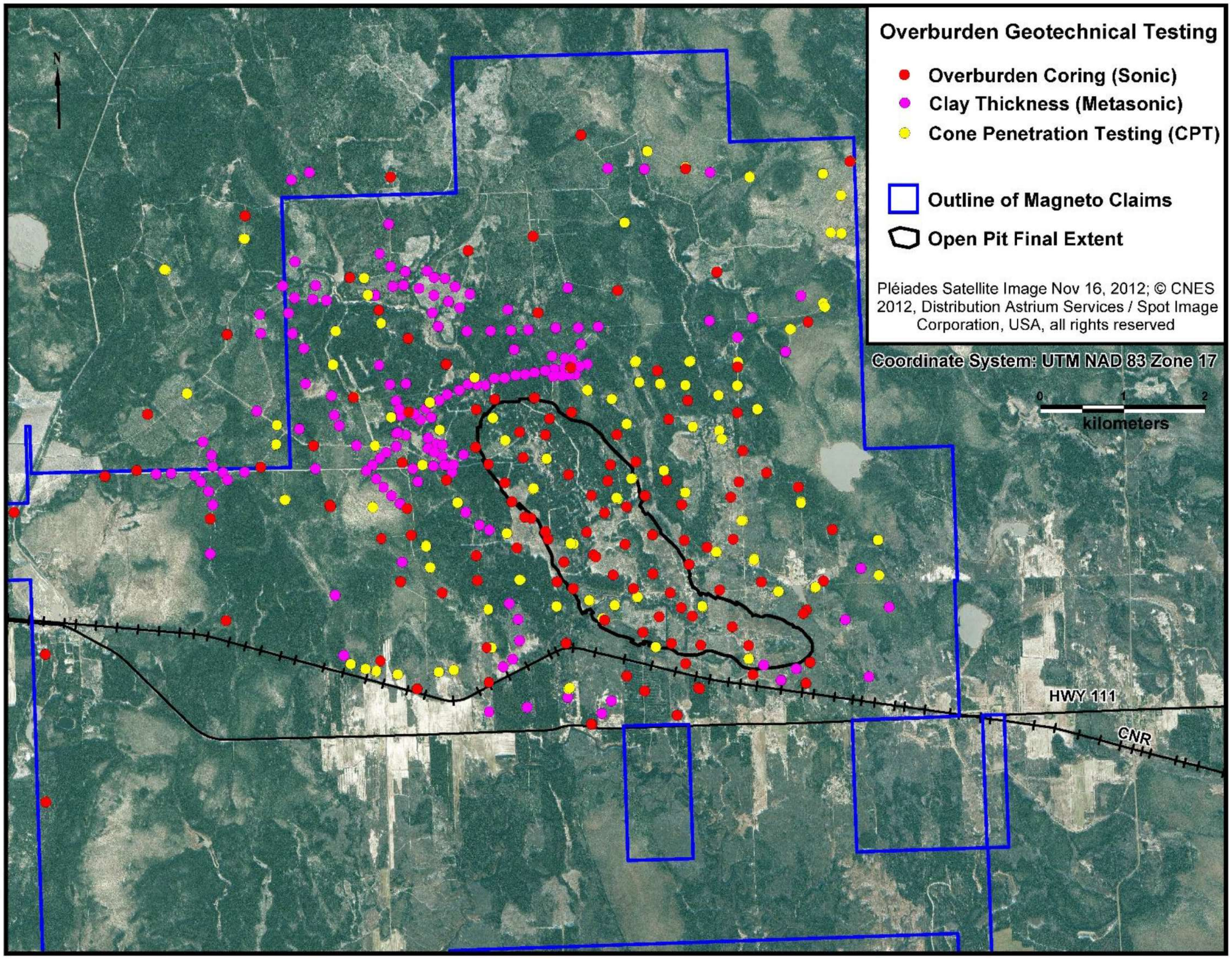
Source: RNC.

Figure 10-2: Drill Holes on the Dumont Property – Drilling Year



Source: RNC.

Figure 10-3: Overburden Drilling & Cone Penetration Test (CPT) Sites



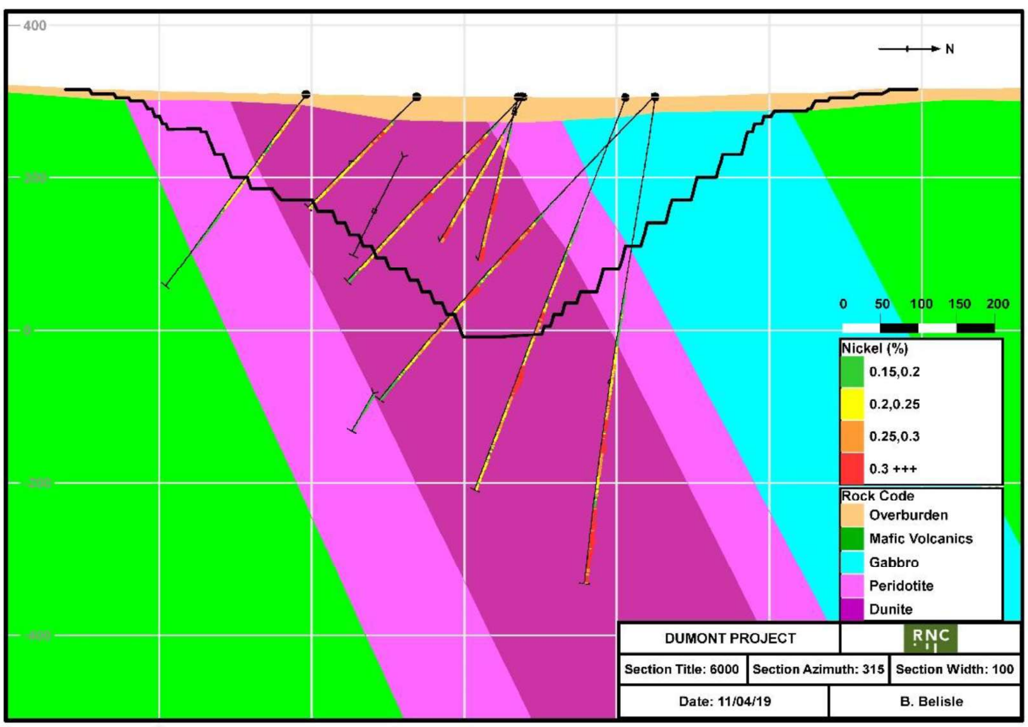
Source: RNC.

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.

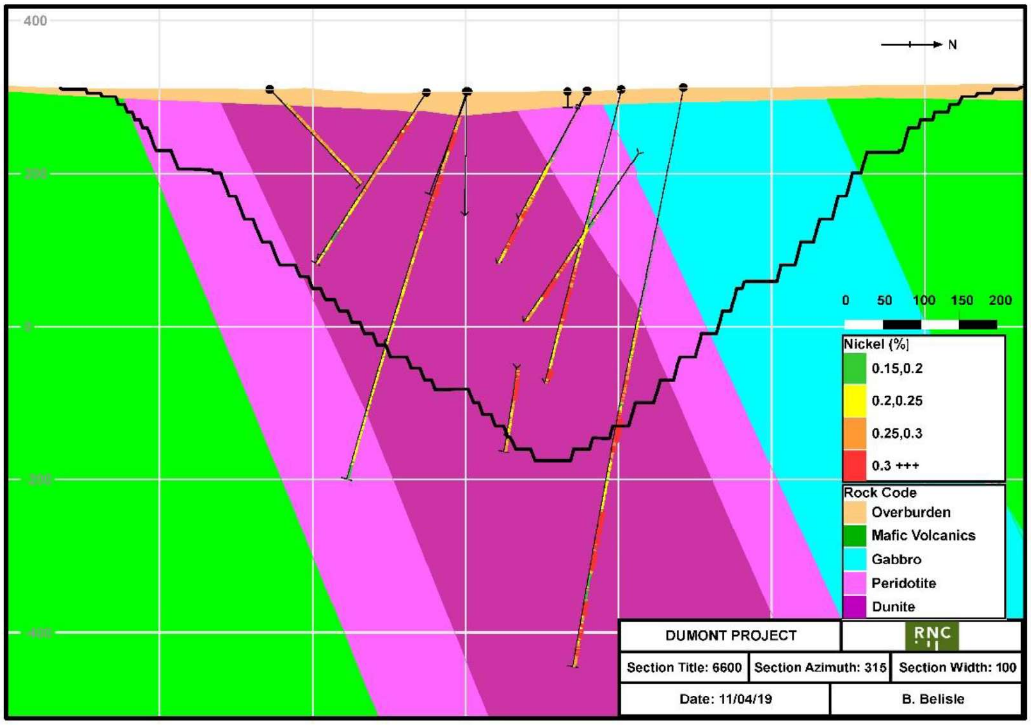
The program was designed to define mineralization down to a nominal depth of 500 m from surface (- 200 m elevation). In places, drilling has investigated mineralization down to a depth of 700 m (- 400 m elevation). Figure 10-1 illustrates the location of all holes completed during the sectional resource 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



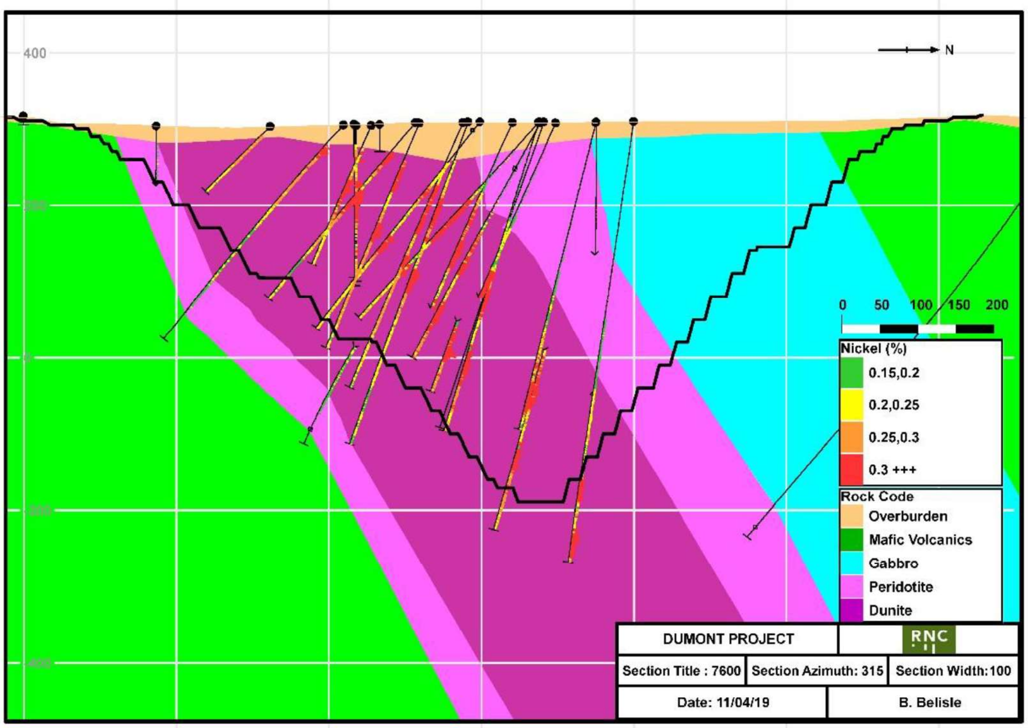
Source: RNC. Note that the scale is given in metres. Section shown is 100 m wide.

Figure 10-5: Drill Section 6600 E showing Outline of FS Pit



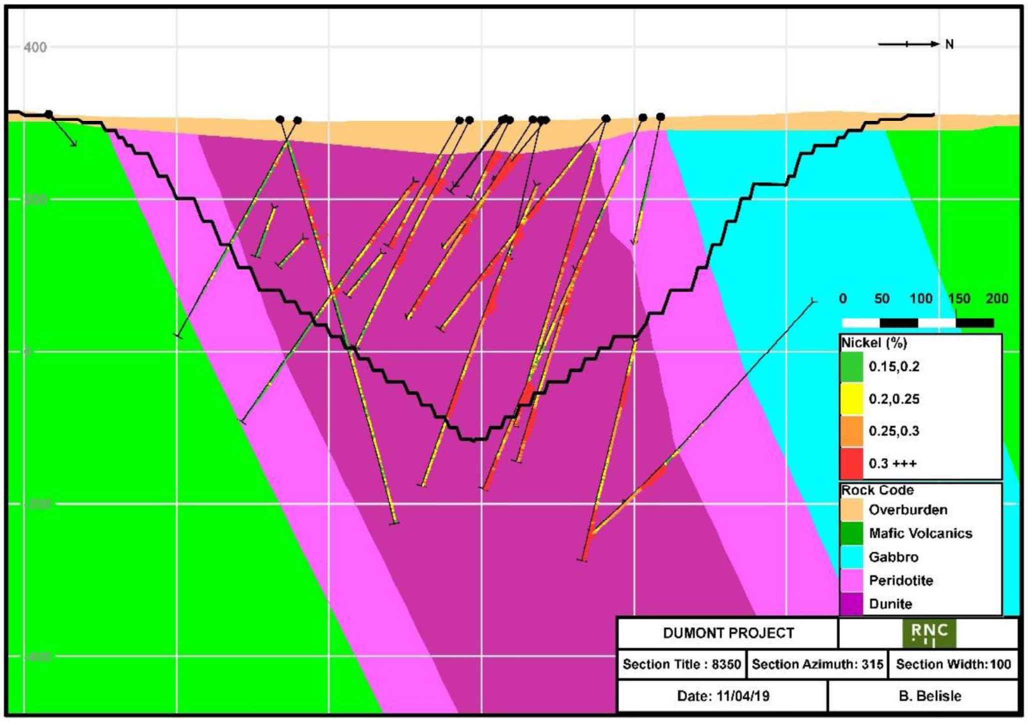
Source: RNC. Note that the scale is given in metres. Section shown is 100 m wide.

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 crosscut 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

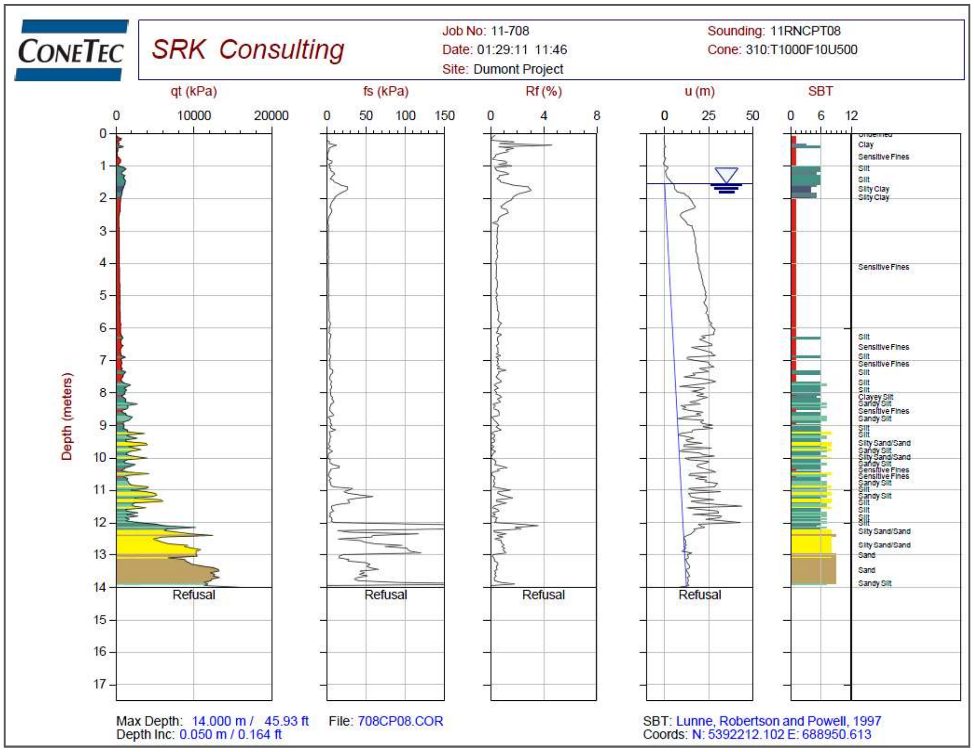
During the winter of 2011, drill holes were completed at 66 locations using a sonic drill rig which employs high frequency, resonate energy to advance the core barrel and casing into the ground. The drill hole plan and locations were strongly influenced by site accessibility, particularly in relation 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 test work. 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 test work.

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 test work 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 test work 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, 2012 and 2013 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 test work, 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" (July 2013). No new sampling has been performed on the Dumont Project since publication of the 2013 Technical Report.

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: 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™ 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: 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 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: 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 an RNC data validation sample.

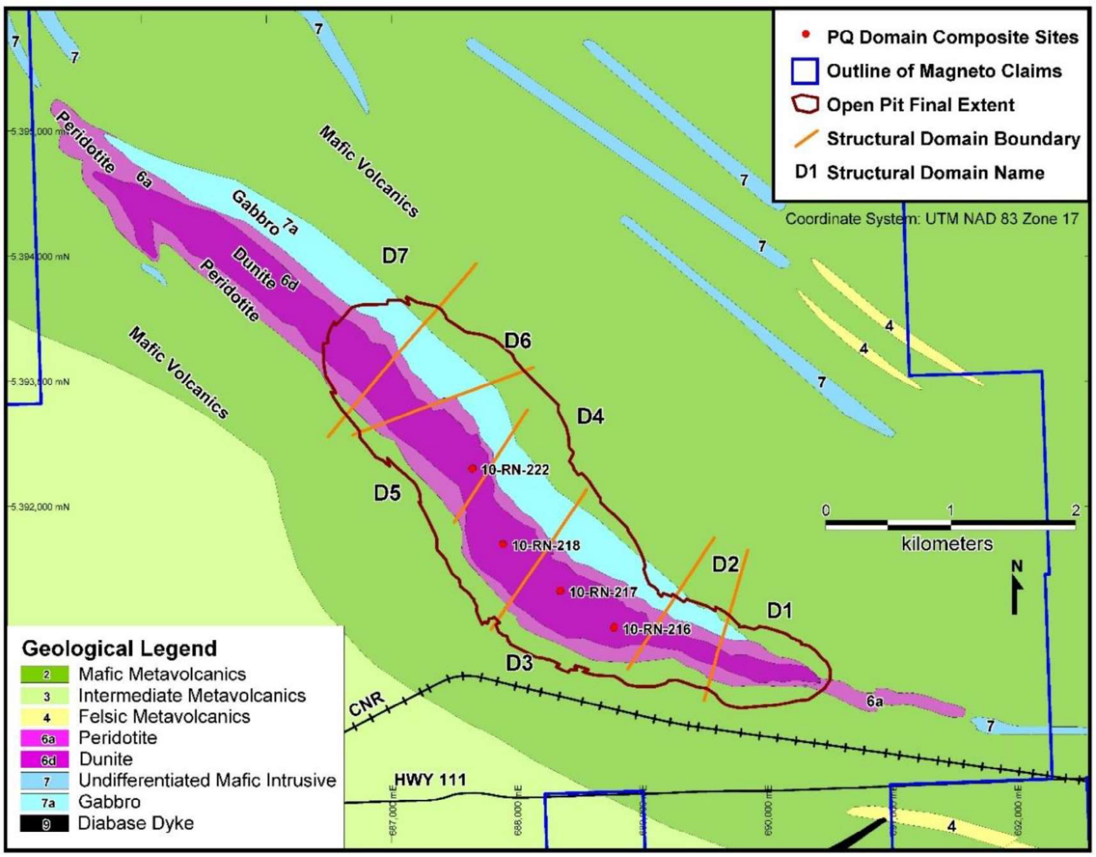
As part of RNC's QA/QC procedures for geochemical assays, a set of control samples comprised of a blank and standard reference material sample, are inserted sequentially into the sample stream. The cut mineralogical samples along with the inserted control samples are then shipped to the ALS Minerals 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 test work. 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 test work.

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.

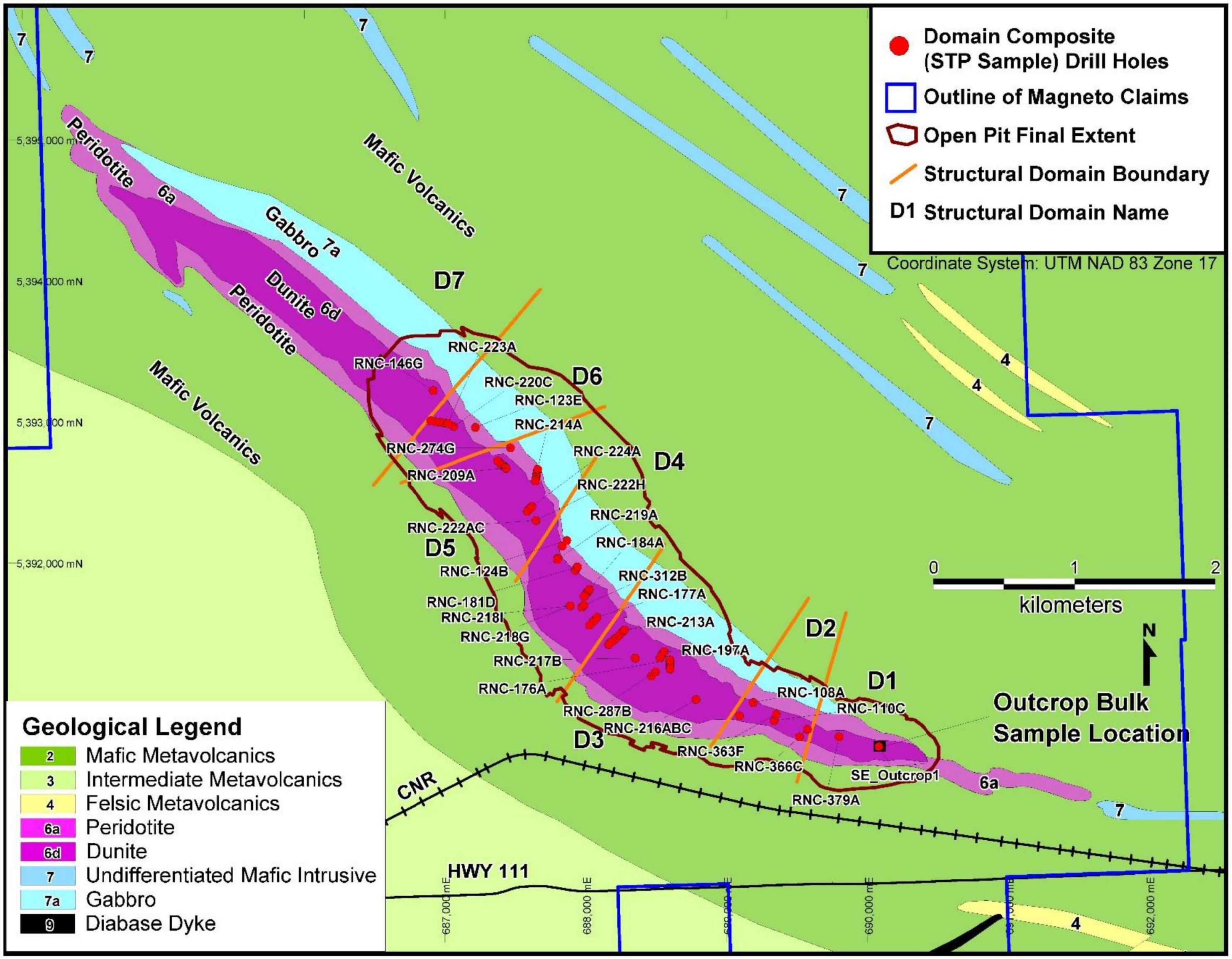
11.1.5 Metallurgical Variability Sample Selection

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. Test work was completed on 105 individual metallurgical domain composite samples. Test work includes both metallurgical lab scale recovery tests as well as mineralogical analysis by QEMSCAN and assay.

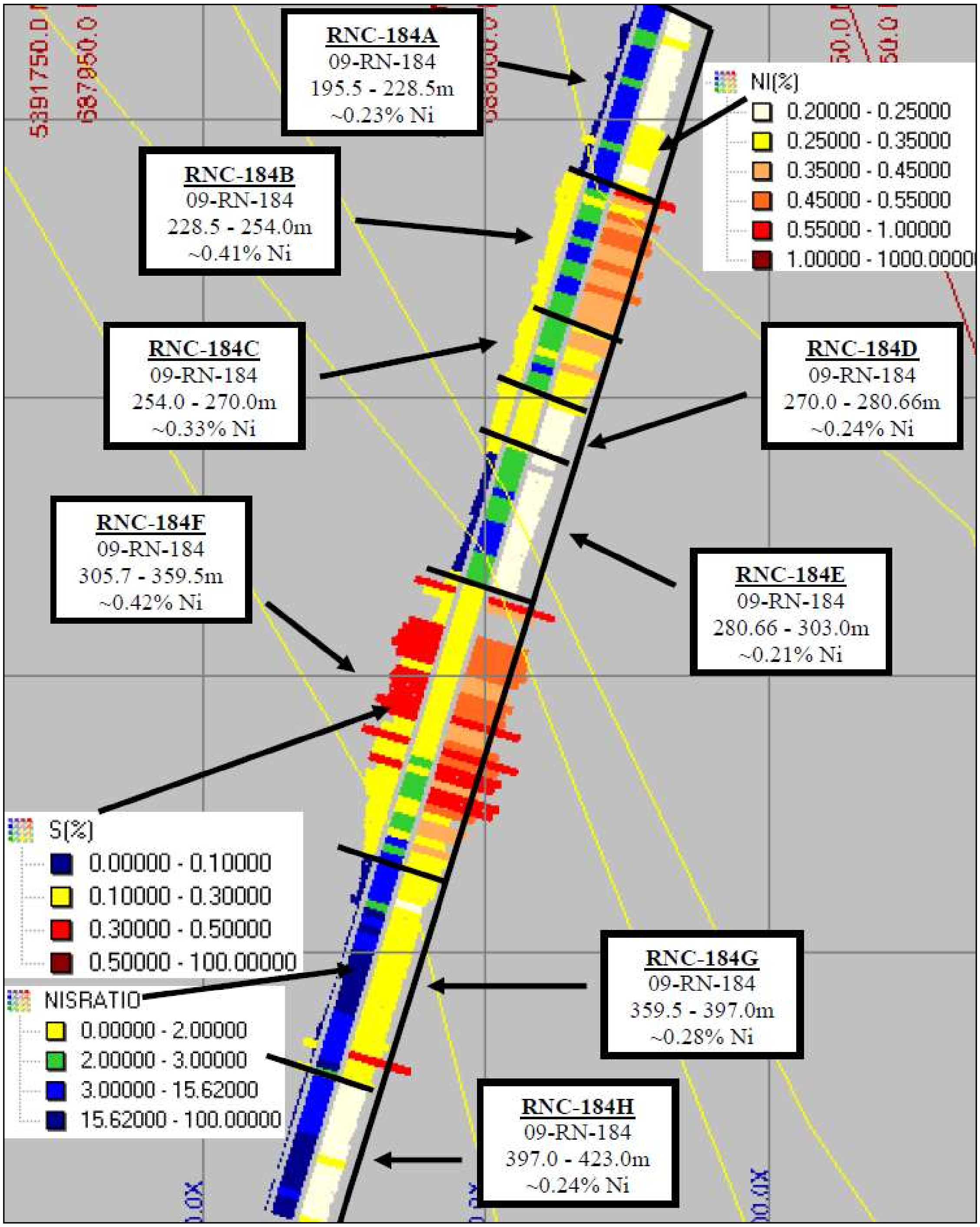
Continuous domain samples were assembled along the continuous length of the drill holes as shown in Figure 11-4. Each of the samples defined a homogeneous domain as characterized by nickel grade, nickel deportment, mineralization grain size and alteration. Any change in these characteristics led to the 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 test work, 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).

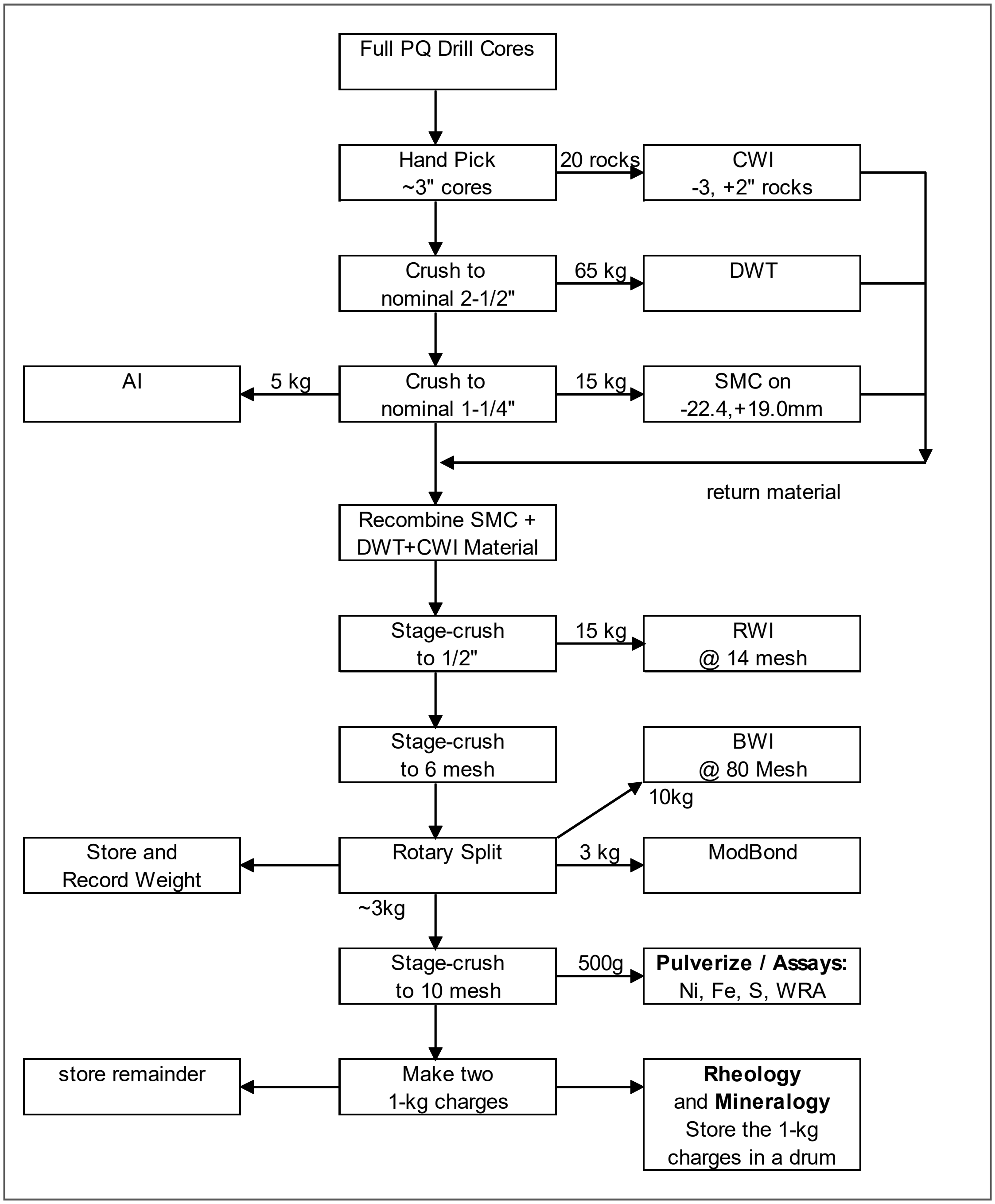
11.1.6.2 Sample Preparation

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.

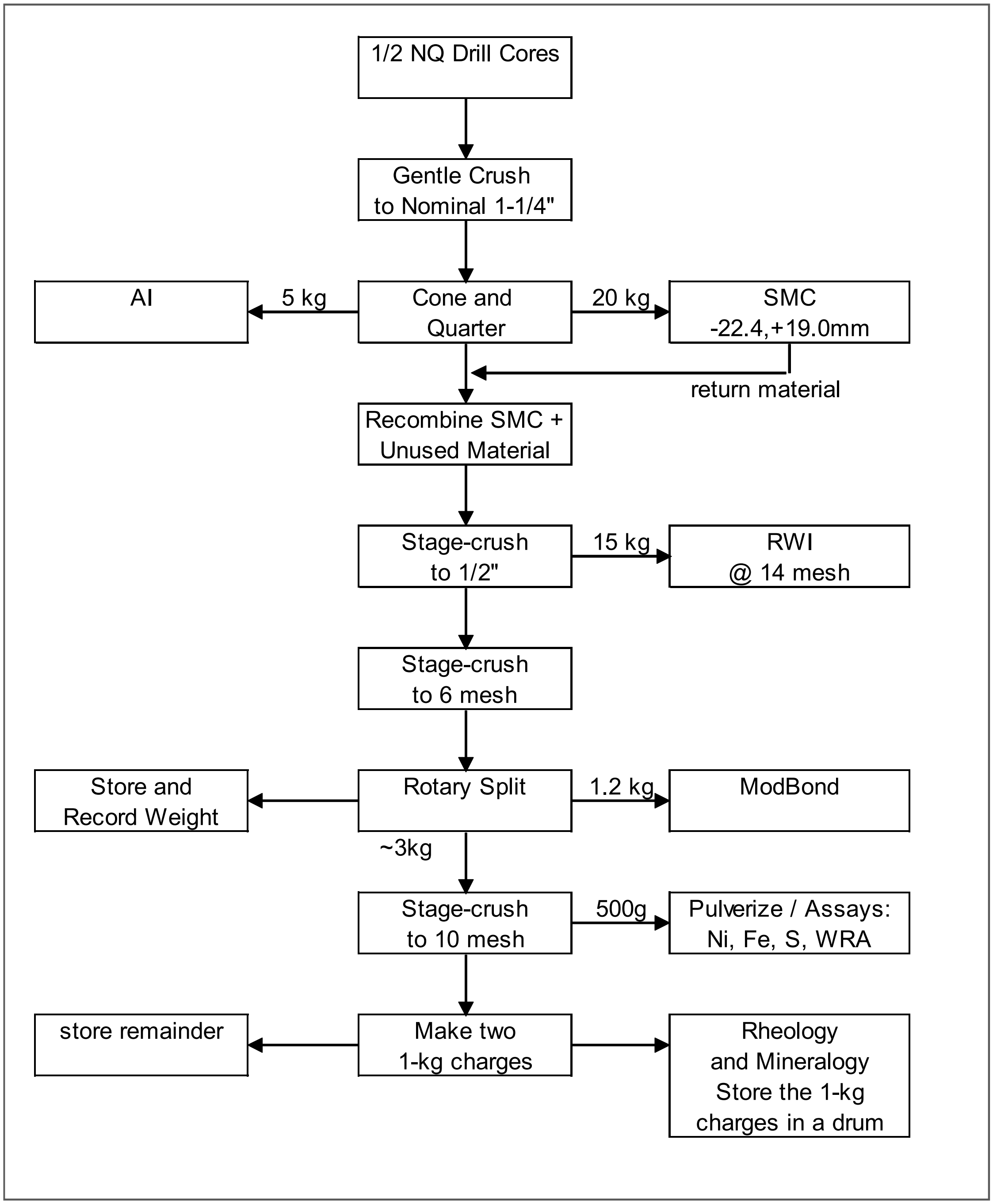
The 92 half-NQ drill core samples were submitted for the same suite of tests with the exception of the Bond low-energy impact test and the drop-weight test. The preparation of the 92 half-NQ drill core samples is shown in Figure 11-6. Three samples selected by RNC were submitted for full rheology benchmark 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



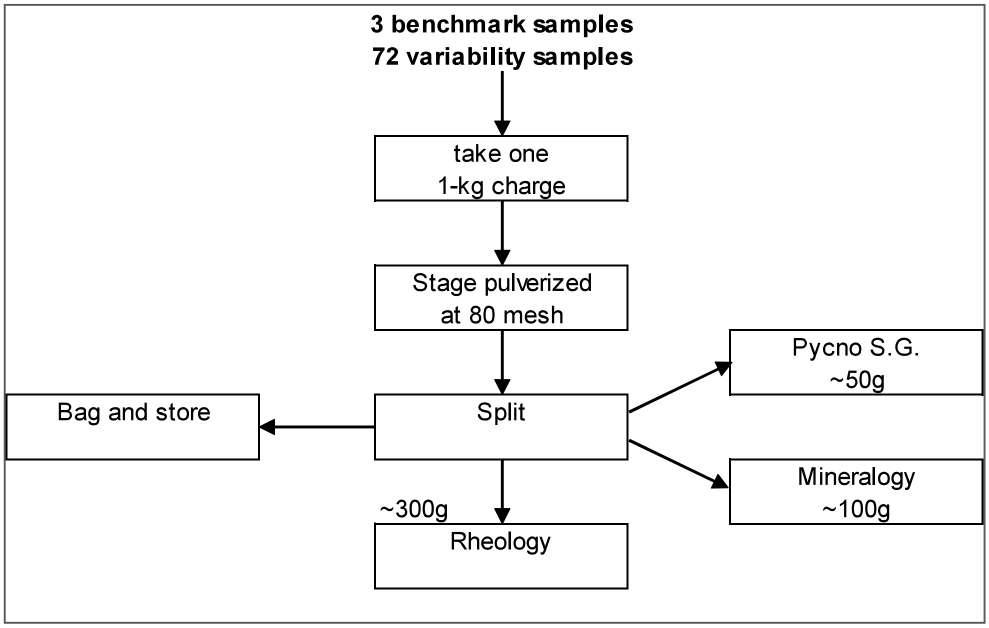
Source: SGS Minerals Services.

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 Test Work

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 inpit 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 inpit tailings deposition scenario after mining is complete.

11.1.7.2 Analytical Methods for Laboratory Test Work (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 (CaCO3) 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 (CaCO3). 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 Test Work (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, PO4, 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 mini plant, 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 mini plant 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 mini plant 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 were 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™ 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™ 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™ mineral mapping program. Satmagan results of the EXPLOMIN™ 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™ results. Satmagan was performed on 10% of the EXPLOMIN™ study.

11.3 SRK Comments

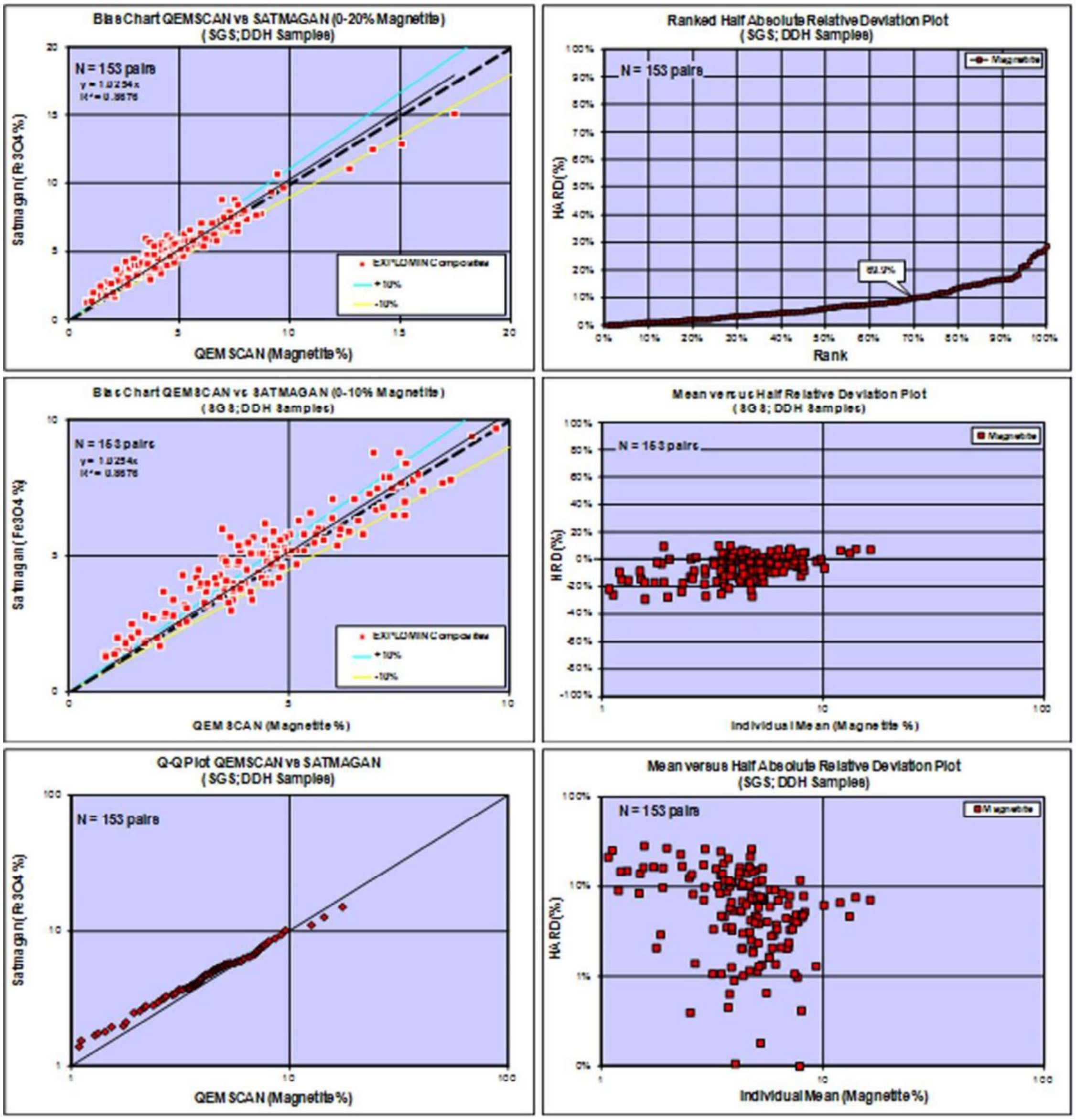
In the opinion of SRK, the sampling preparation, security and analytical procedures used by RNC are consistent with and often exceed generally accepted industry best practices.

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, OGQ 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 visits 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 EXPLOMINTM 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.

On October 23, 2018, Chelsey Protulipac, P.Geo from SRK visited the project site accompanied by Robert Cloutier, Geo., OGQ from RNC. The site visit was undertaken to confirm the exploration work completed and assess the extent of bulk sampling completed to date. During the visit, a selection of borehole collars was examined and compared to the database. No discrepancies were found between the location, identification, and orientation of the borehole collars examined.

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.

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.

The 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 shows 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 shows 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.

In 2015 a pilot plant was conducted at SGS- Lakefield on the Outcrop sample (Hz Domain only) to generate concentrate for downstream testing. The pilot plant treated ~300 tonnes of material in order to generate sufficient concentrate for roasting. Metallurgical data was gathered on the slimes and rougher circuits. The Aw circuit was not tested due to the absence of Aw in the sample. Tests for concentrate thickening, filtration and threshold moisture limits were completed on this concentrate sample.

In 2014-2016 roasting tests were performed on the Dumont concentrate. The concentrate produced from the pilot plant was able to confirm the ability to dead roast the Hz concentrate as well as provide larger samples for downstream test work by potential end users including stainless steel plants and nickel pig iron producers.

13.2 Previous Test work

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 Test Work 1971-1972

In 1971 and 1972, Centre de Recherches Minérales, Ste-Foy, Quebec (CRM) conducted a laboratory test work 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 test work 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 Test Work 2007-2008

Preliminary metallurgical test work 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. Test work 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 (STP) 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 test work.

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 STP 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 Test Work

Flotation test work 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 test work 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 Test Work

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 Table 13-1,

Table 13-2 and Table 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 test work 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 test work 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.

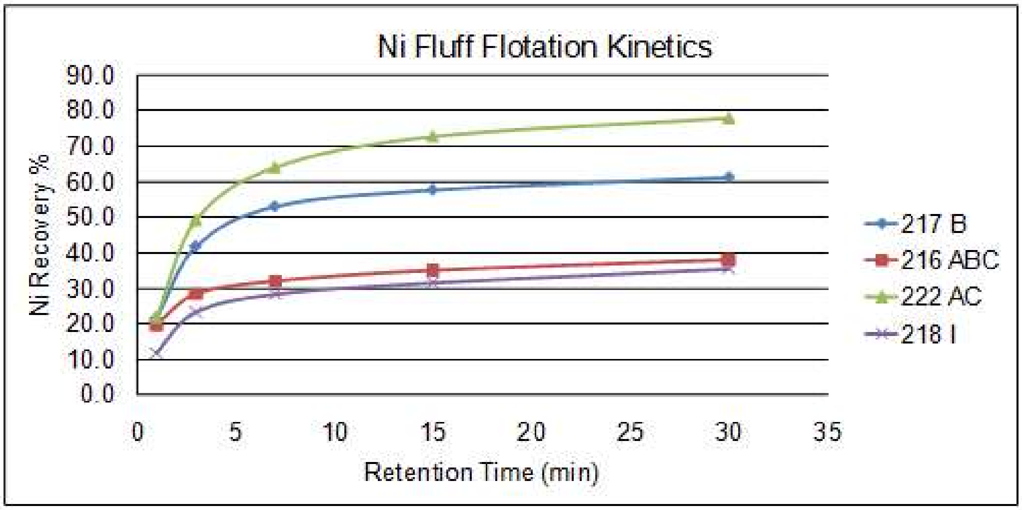
13.2.3.2 Recovery from the Fibre Portion of STP samples

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 P_{80} of the fibre product is 180 µm which is very similar to the P_{80} of the rougher grind used in the STP.

The size distribution of the fluff product (the fluff product is the material removed by air defibering) is around 180 µm, or similar to P_{80} 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 test work shown in Figure 13-1.

Production of the fluff product through defibering has been discontinued in the feasibility flowsheet as the process step was not leading to a metallurgical improvement compared with desliming. For the overall rougher nickel recovery analysis, based on the results shown below, the fluff that was produced in the first set of 70 STP tests is assumed to equal the rougher recovery for that test.

Figure 13-1: Recovery from Fluff Portion of STP



Source: RNC.

13.3 Feasibility Study 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 test work 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 Table 13-1, Table 13-5 and Table 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 Table 13-7 to Table 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				

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 (Hz). 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 test work. 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 400 mm 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 Test Work

The testing consisted of both grindability test work to characterize the competency, hardness and abrasion of the Dumont material as well as slurry rheology. The historical work had focussed on selecting samples based on assumed mineralogical impact in section 13.2.2.4. The sample selection to support this study were chosen to cover the breadth and depth of the deposit to evaluate the variability.

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 Test Work 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

	JKTech Parameters				Work Indices					
Statistics	Axb smc	DWI kWh/m³	T_{10} @ 1 kWh/t	Rel. Density	CWi kWh/t	RWi kWh/t	BWi kWh/t	Mod. kWh/t	Ai g	Ni Grade %
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 10th to 90th), 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 (10th percentile) to 43.6 (90th 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 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 was 13.5 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 (10th percentile) to 16.3 kWh/t (90th 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 (10th percentile) to 22.3 kWh/t (90th 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 (10th percentile) to 22.0 kWh/t (90th 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 test work (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		Observations	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		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		-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, rheopexy, some plug flow	-9	-0.17	58	1.056
T4	50.1	30.0	26	5	26	3	Peak, transient 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 behaviour was re-measured. This allowed the rheological behaviour to be characterized as a function of their solids density.

The benchmark samples showed a variety of rheological behaviour, 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 behaviour comparable to Benchmark A, featuring extreme rheopexy, rendering their shear yield stress behaviour non-measurable due to torque overload at the CSD. These samples displayed unsheared yield stress values ranging from 7 Pa to 83 Pa, average 61 Pa.

Seven samples displayed flow behaviour 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 behaviour 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 solid 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.

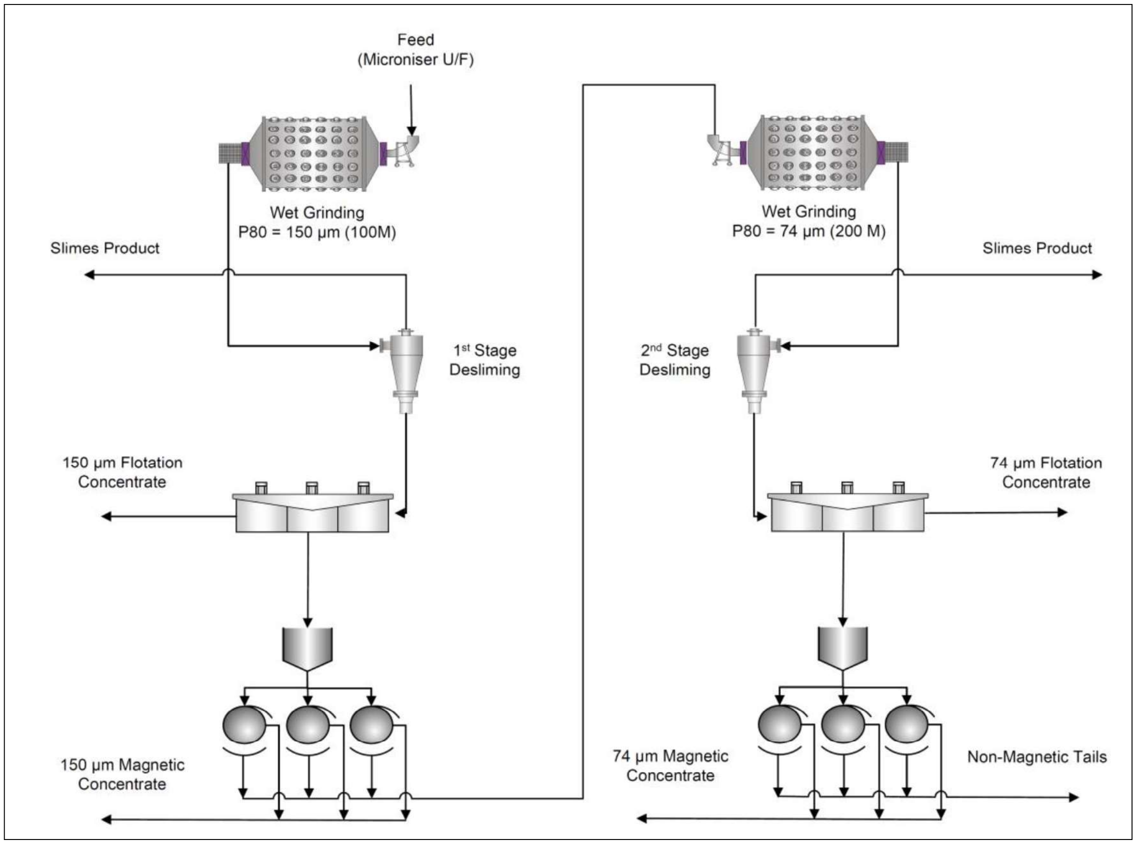
13.6.2 Variability Testing (STP Metallurgical Domain Samples)

The initial 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 was put into bags and then frozen as soon as possible;
- the air-defibered (fluff) portion 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 was processed by a hydrocyclone to deslime the pulp with approximately 5% weight going to the overflow;
- flotation separation was conducted on the hydrocyclone U/F;

- magnetic separation was conducted on the flotation rougher tails;
- the non-magnetic portion was then wet ground to an 80% minus 200 mesh;
- dispersant Calgon at 500 g/t and PAX at 100 g/t was added into the ball mill prior to wet grinding;
- second stage of grinding was followed by a second stage of wet desliming (about 5% weight loss to the overflow);
- a second stage of flotation separation was conducted on the hydrocyclone U/F;
- the flotation tails underwent a second stage of magnetic separation;
- weight assessment recorded for 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

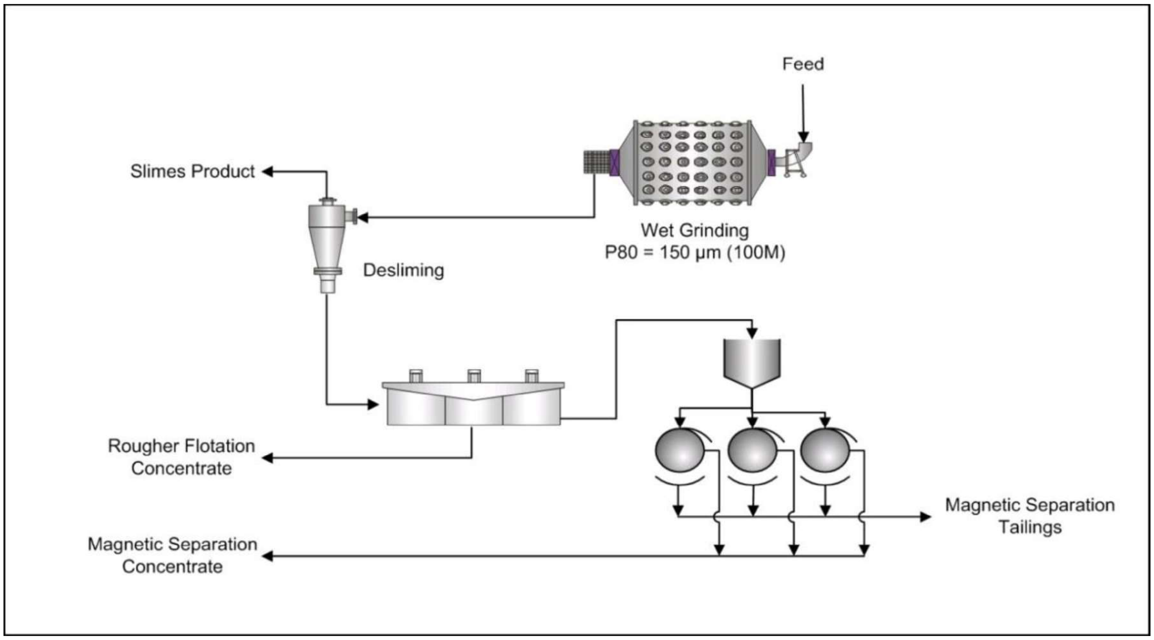
Stage	Flotation Cell	Speed (rpm)
Flotation Cell	Denver D2 60L	1,600

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 was 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 was added into the ball mill prior to wet grinding;
- the sample was treated with a hydrocyclone to deslime the pulp with approximately 5-10% weight going to the overflow;
- flotation separation was conducted on the hydrocyclone underflow;
- magnetic separation was conducted on the flotation rougher tails; and
- weight recorded for 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. Test work 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 sample 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.

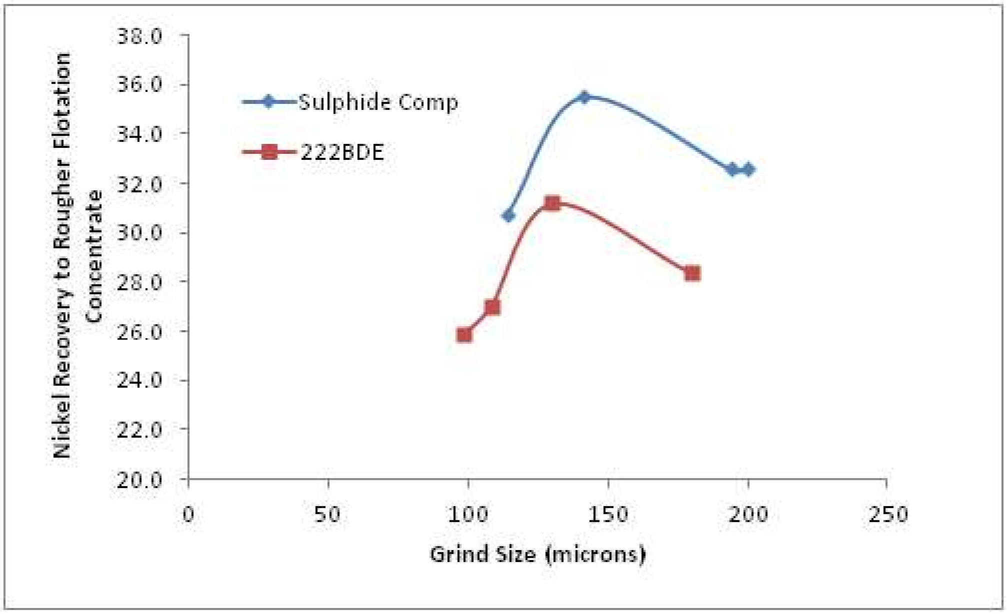
13.7.1.2 Grind Size Selection

The STP grind size was chosen as 150 µm (100 mesh). Test work 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 plots the flotation rougher recovery vs. the P80 of the sample. Both samples show a peak in flotation recovery between 130-160 µm.

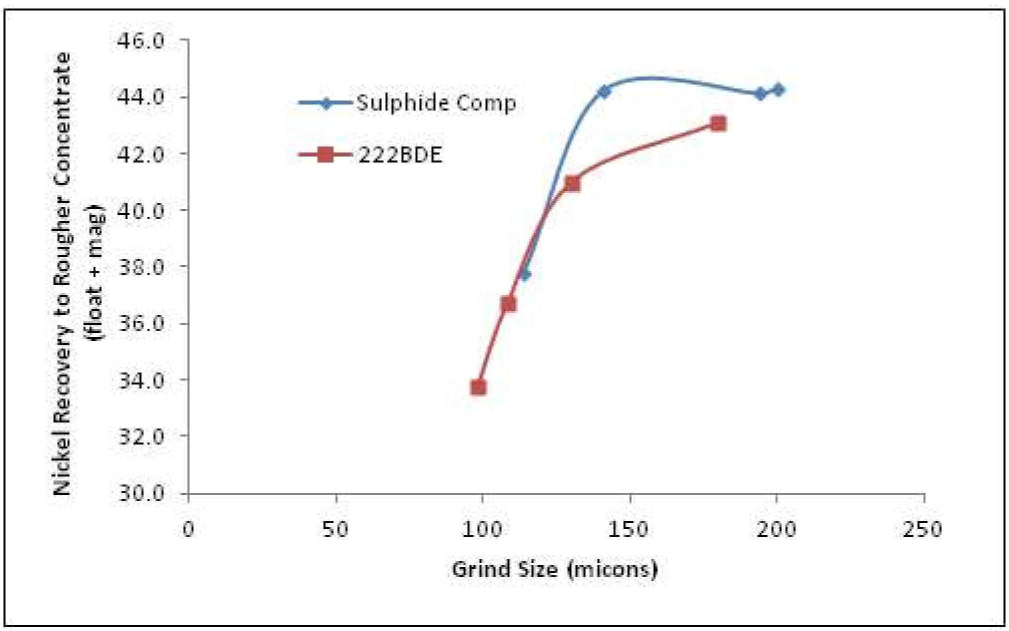
Total rougher recovery is comprised of both the flotation and magnetic recovery. Figure 13-5 shows the total rougher recovery vs. the P80 of the sample. In general, the total nickel recovery is higher as the grind size gets coarser (within the size range tested). This may result from increased kinetics 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).

Figure 13-4: Flotation Recovery as a Function of Grind Size



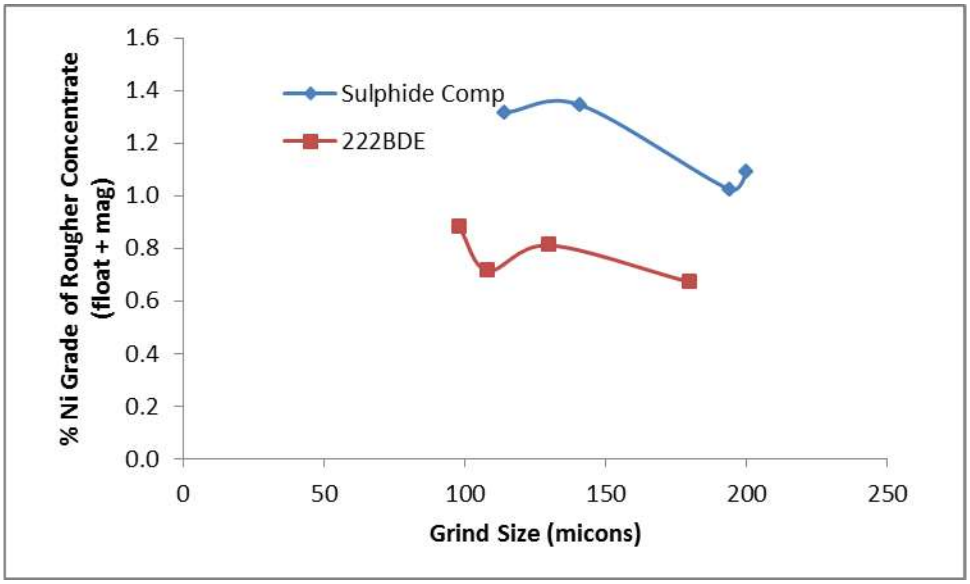
Source: RNC.

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 Table 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	Ni Recy[1] 10 min	% Ni Grade 20 min	Ni 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	Ni Recy[1] 10 min	% Ni Grade 20 min	Ni 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 Table 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 PAS 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 test work, 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	Ni Recy[1] 20 min	% Ni Grade 30 min	Ni 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	Ni Recy[1] 20 min	% Ni Grade 30 min	Ni 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 test work, 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 test work 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	Rougher 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	68.62
STP (original)	68	0	1.3	66.32

[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

Overflow 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	Rougher 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 used relatively high dosages of several reagents; specifically, Calgon (sodium hexametaphosphate) and carboxy-methyl cellulose (CMC). 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 Test Work

	MIBC (g/t)	Cytec 65 (g/t)	KAX (g/t)	Calgon (g/t)	CMC (g/t)	$H2SO_4$ (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 test work 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 test work 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)	$H2SO_4$ (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)	$H2SO_4$ (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 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) Table 13-31 and O/F (slimes) Table 13-32. 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	**78**	**2**	**89**	**225**	**22**	**3767**

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^2h)	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.

13.7.5.1 Separate Coarse and Slimes Tailings Streams Dewatering

For the 2019 Feasibility Study, the optimization of the tailings deposition plan required the plant to generate distinct coarse and fine tailings portions. Test work had been performed on the settling characteristics of the separate streams in 2011. This test work indicated that the slimes were not able to be efficiently thickened on their own, but this needed to be confirmed with updated test work.

A sample of each of the slimes tails and rougher tails 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. Flocculant screening was performed and Magnafloc 333 was chosen for the slimes tailings settling tests while 913 VHM was chosen for the coarse tailings settling tests.

The rougher tails settled as expected from the previous test work. The slimes tails settled poorly as expected.

The slimes settling test was repeated with 2 parts slimes tails to 1-part rougher tails to evaluate if adding a portion of the coarser rougher tails to the slimes tails would positively impact the slimes settling rates.

Table 13-35 is a summary of the results extracted from the Outotec report (Wakefield 2019).

Table 13-35: Summary of Split Stream Tailings Test Results (Wakefield, 2019)

Sample	Solids Loading Rate (t/m^2h)	Rise Rate (m/h)	Floc Dosage	Achievable Underflow Density (%w/w solids)	Achievable Overflow Clarity (ppm TSS)	Max Unsheared U/F Yield Stress (Pa)
Rougher Tails	0.5-1.3	2.4-6.3	30	52-56	139-220	88
Slimes Tails	0.05-0.1	2 -4	50-70	5.6-36.4	294-689	130
2:1 Slimes to Rougher Blend	0.1-0.6	1.6-9.7	50-80	17.7-47.4	283-742	209

The results from this work were used to update the feasibility design basis, a summary of which was shown below as related to the tailings thickening.

Table 13-36: Summary of Split Stream Tailings Thickener Design Criteria

	U/F Density (w/w % Solids)	Flux (t/m^2h)	Flocculant Consumption (g/t)
Coarse Tails (Rougher Tails)	55%	1.0	30
Slimes: Rougher Tails (2:1)	35%	0.5	60

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

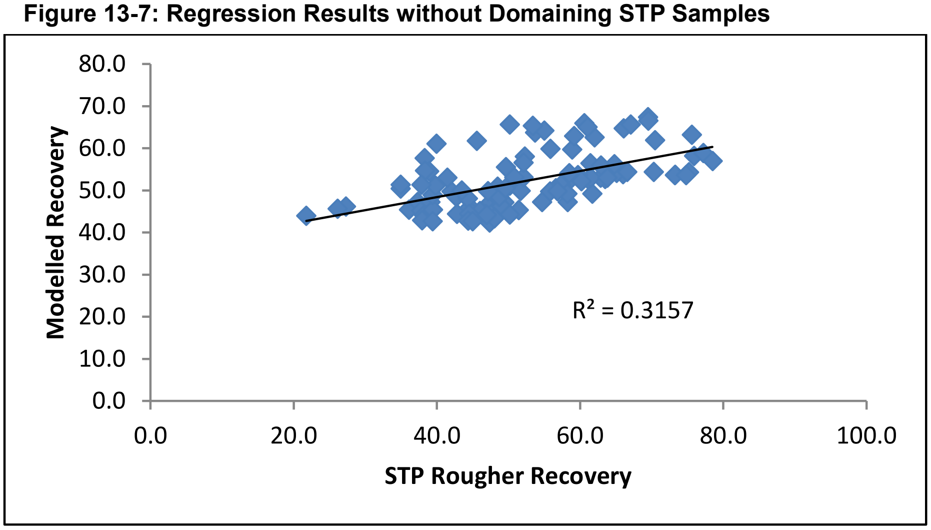
The 105 STP tests formed the basis for the rougher recovery equations. The complete assay and mineralogical data (QEMSCAN) were available for each of the 105 samples.

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 R2 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

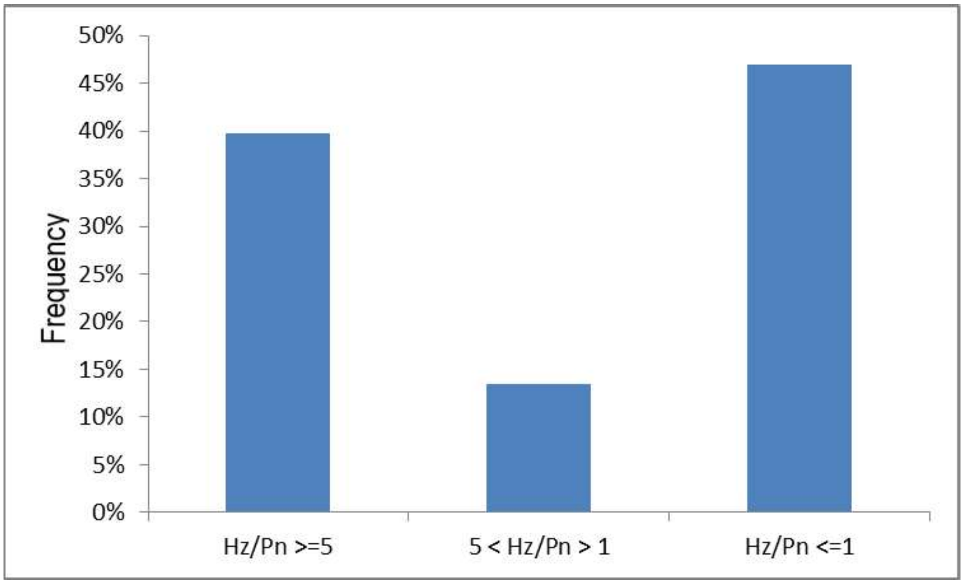


Source: RNC.

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).

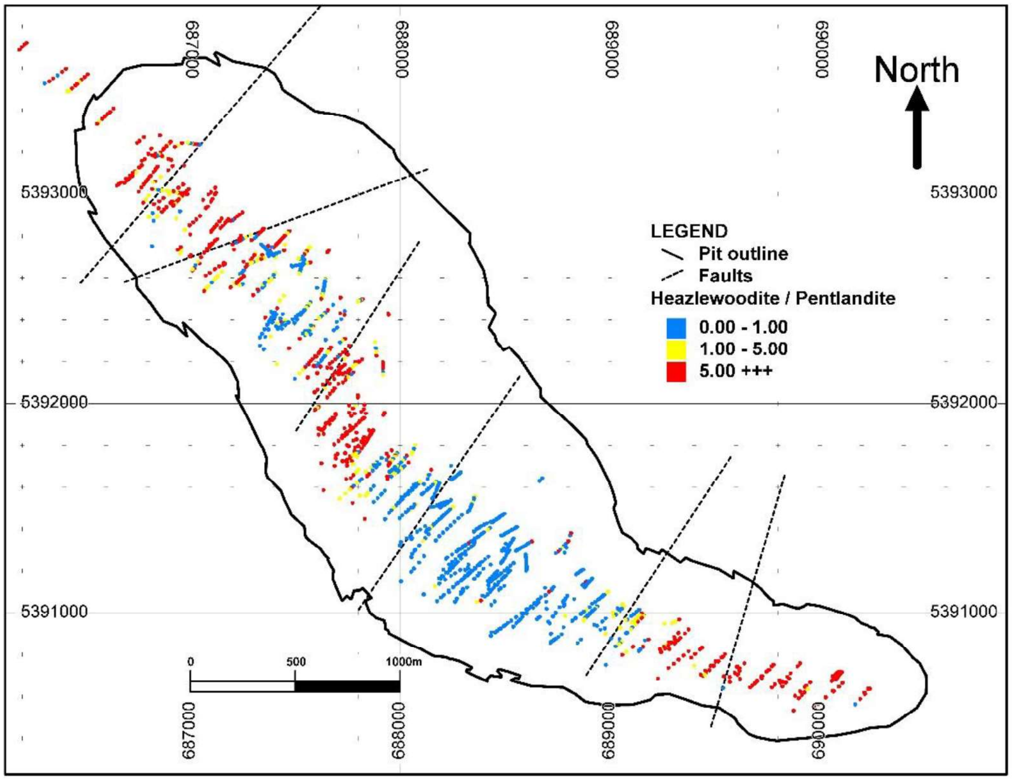
Figure 13-8: Distribution of Hz/Pn Ratio in EXPLOMIN™ Results



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



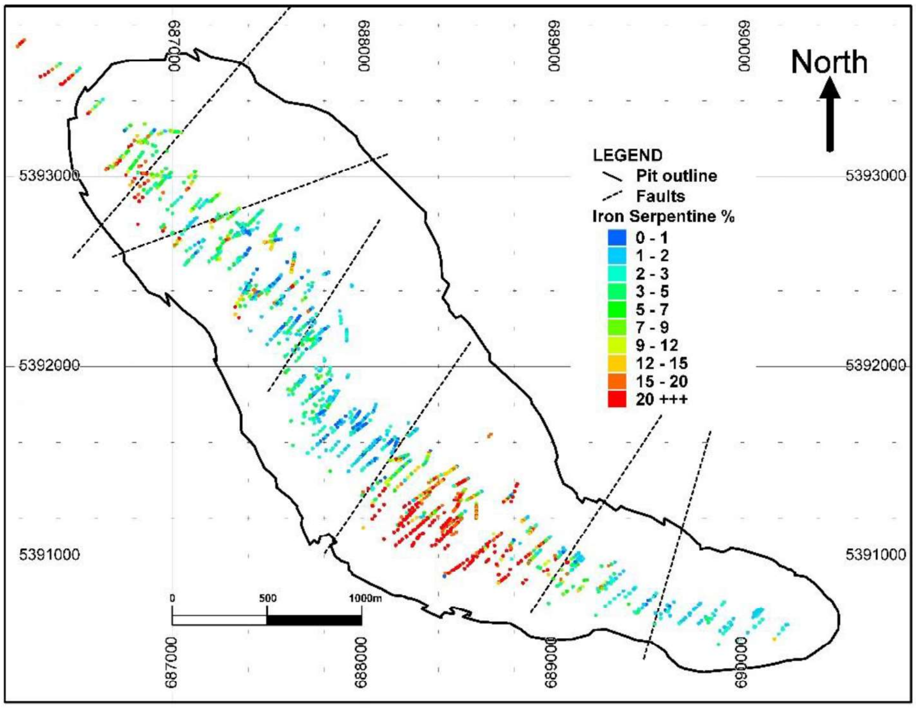
Source: RNC.

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.

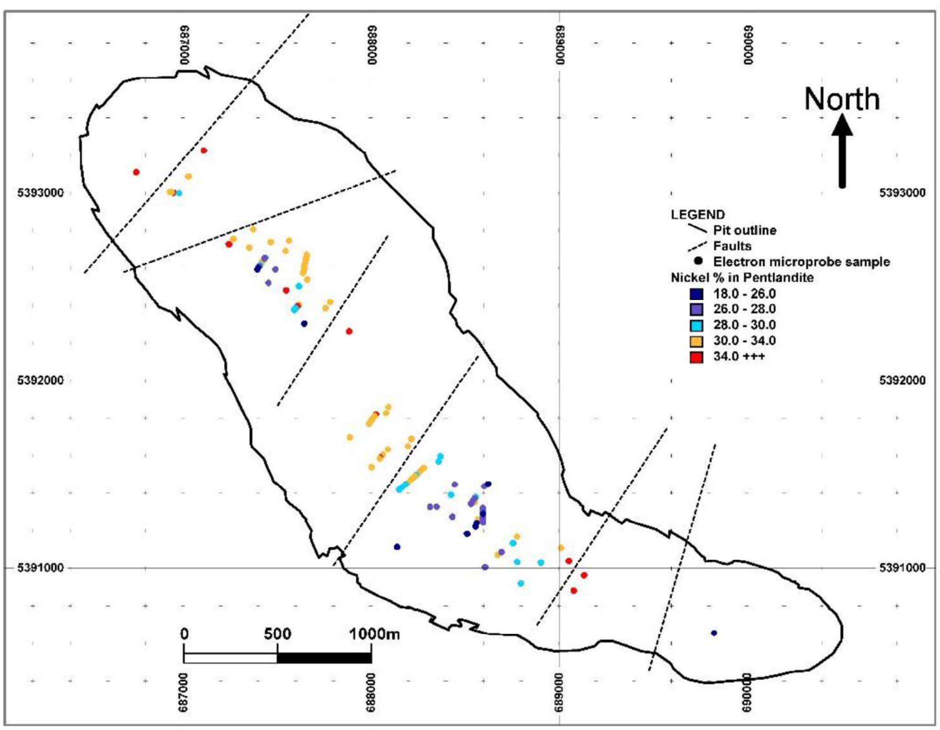
The lower tenor pentlandite (shown in dark blue and black in Figure 13-11) is highly correlated with the areas of Fe Serpentine in both structural Domain 3 and 5. The pentlandite outside these areas has an average Ni tenor of 34%.

Figure 13-10: Distribution of Fe Serpentine within the FS Pit Shell



Source: RNC.

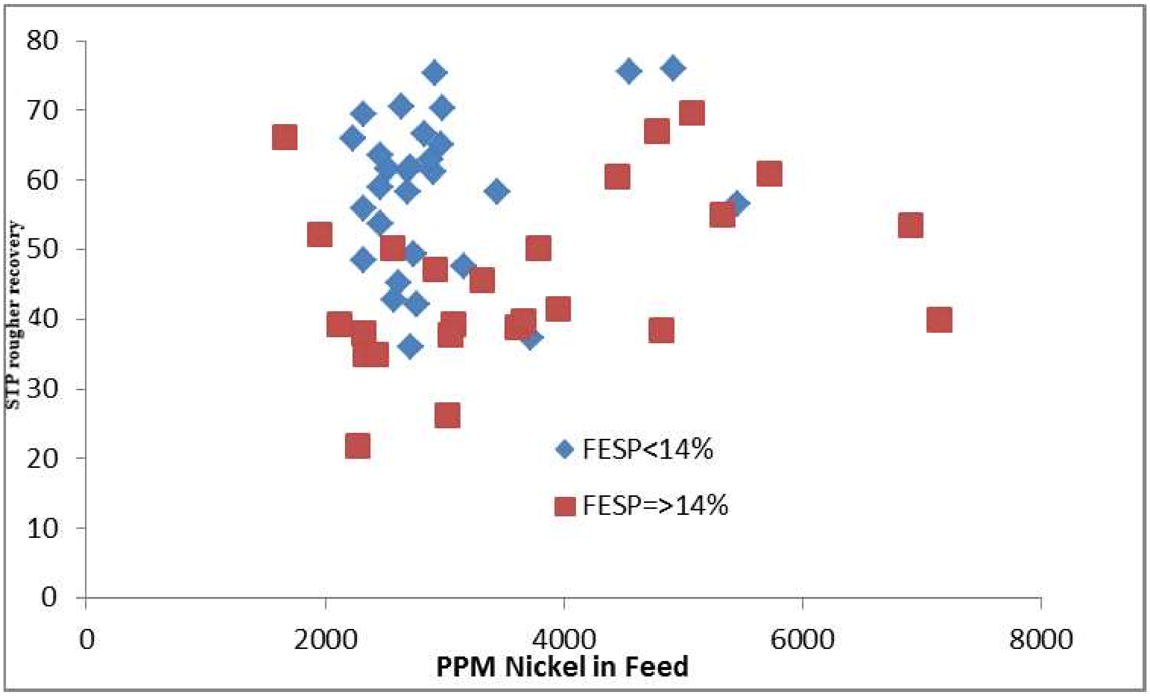
Figure 13-11: Nickel Tenor in Pentlandite



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.

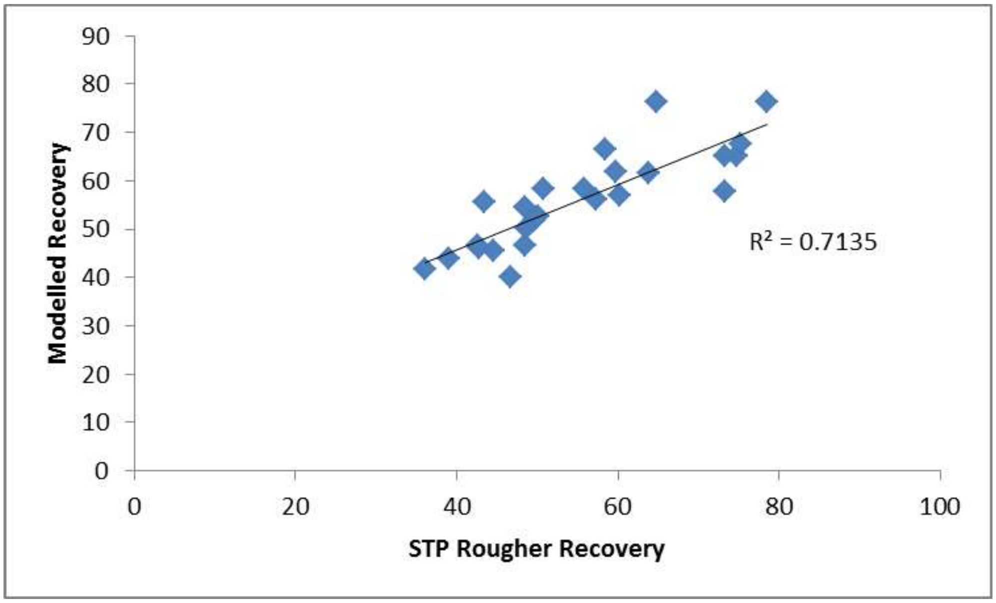
The samples containing over 14% iron serpentine by weight were split out and the regression was run separately.

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

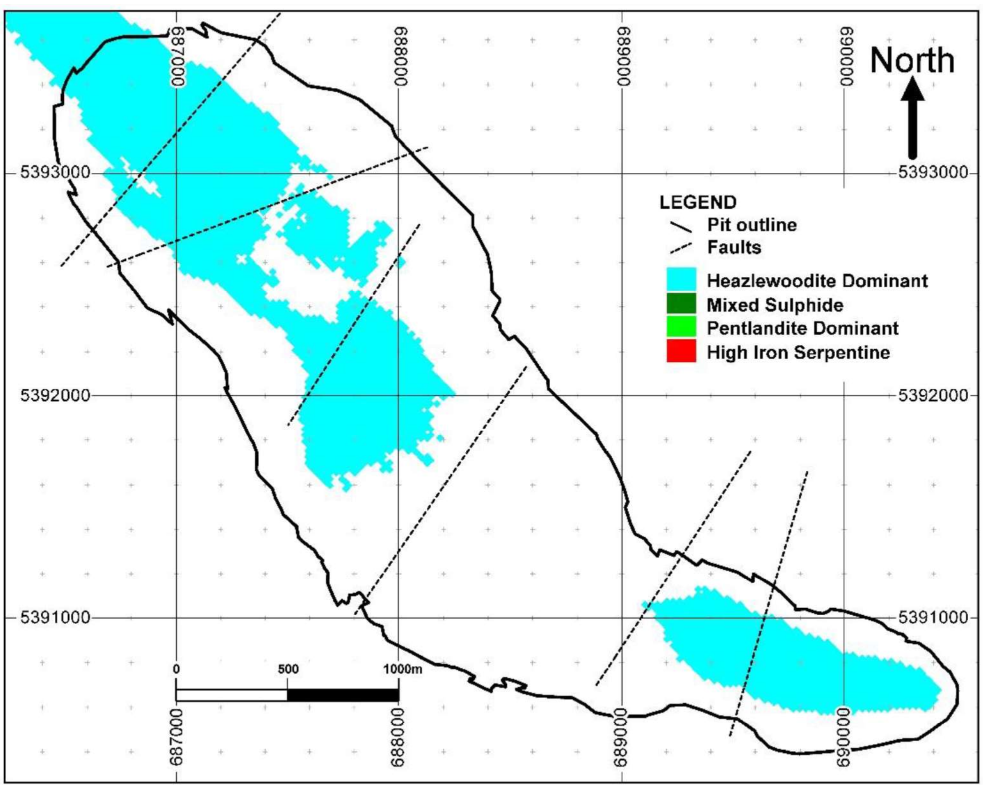
Figure 13-13: Recovery Regression Model for Hz Dominant Samples



Source: RNC.

The distribution of the Hz Dominant zones is shown in Figure 13-14.

Figure 13-14: Distribution of Hz Rich Metallurgical Domain

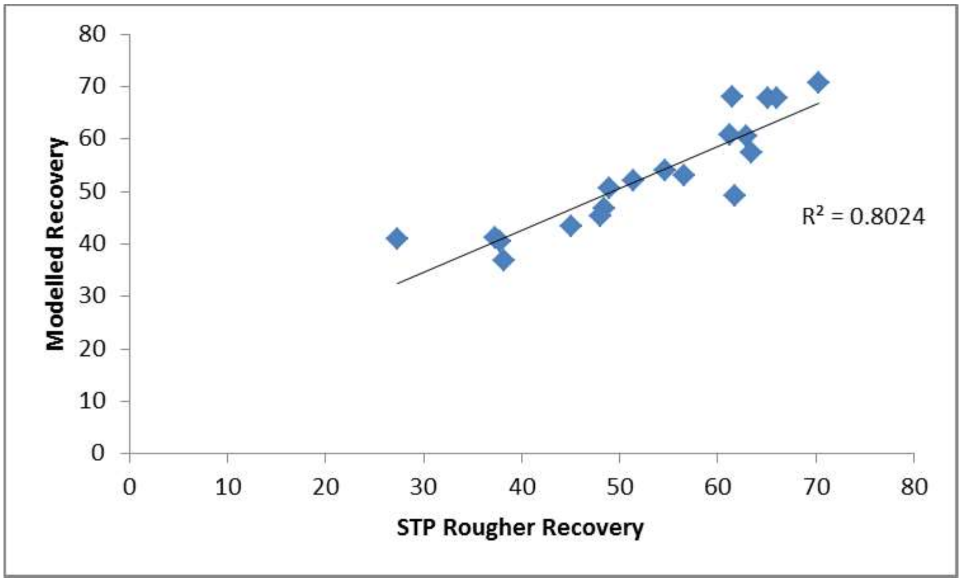


Source: RNC.

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

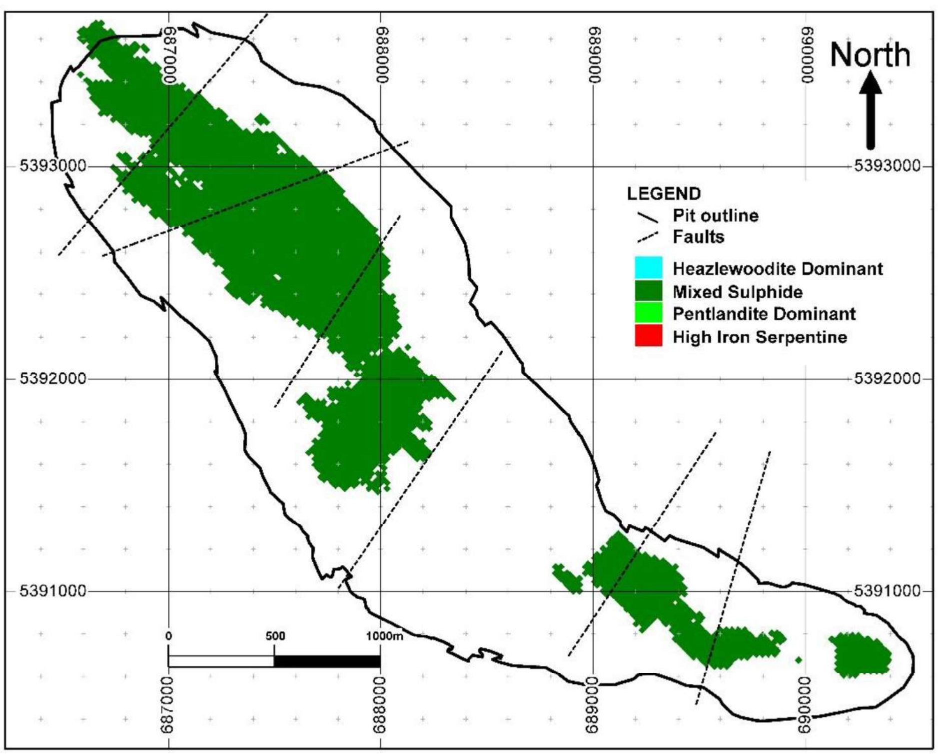
Figure 13-15: Recovery Regression Model for Mixed Sulphide Samples



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

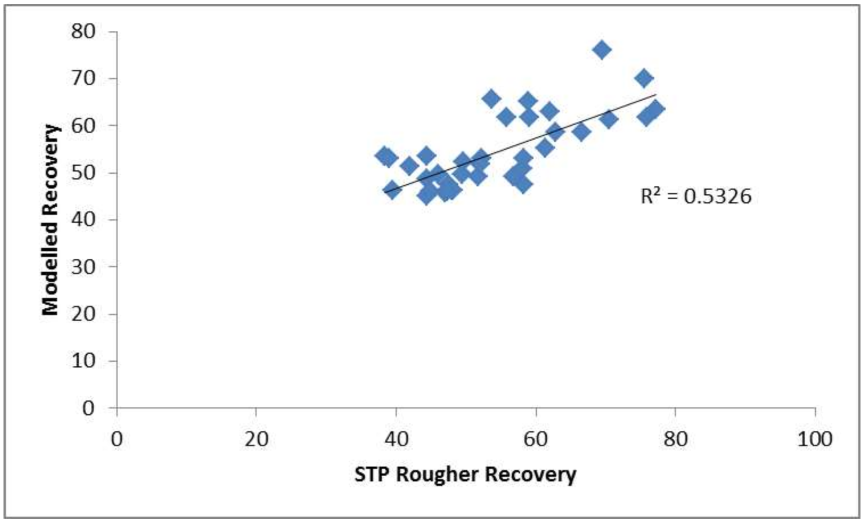


Source: RNC.

13.8.1.3 Results for Pn Dominant: FESP<14%, Pn/Hz <= 1

Rougher Ni Recovery = 43.6 + 00055* S + 0.0111* Ca

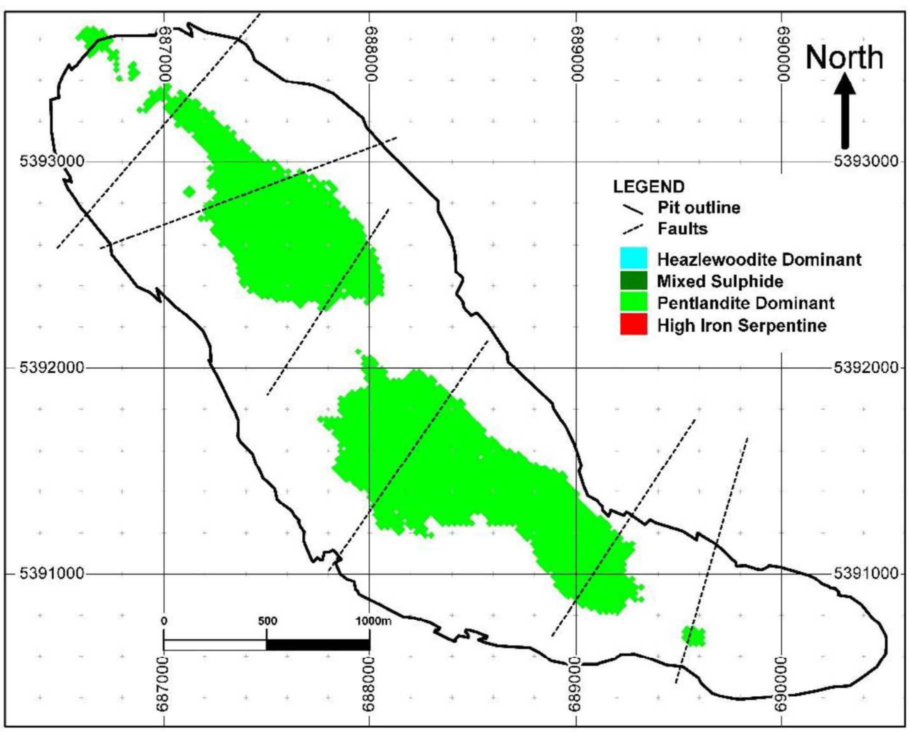
Figure 13-17: Recovery Regression Model for Pn Dominant



Source: RNC.

The distribution of the Pn Dominant domain is shown in Figure 13-18.

Figure 13-18: Distribution of the Pn Dominant Domain

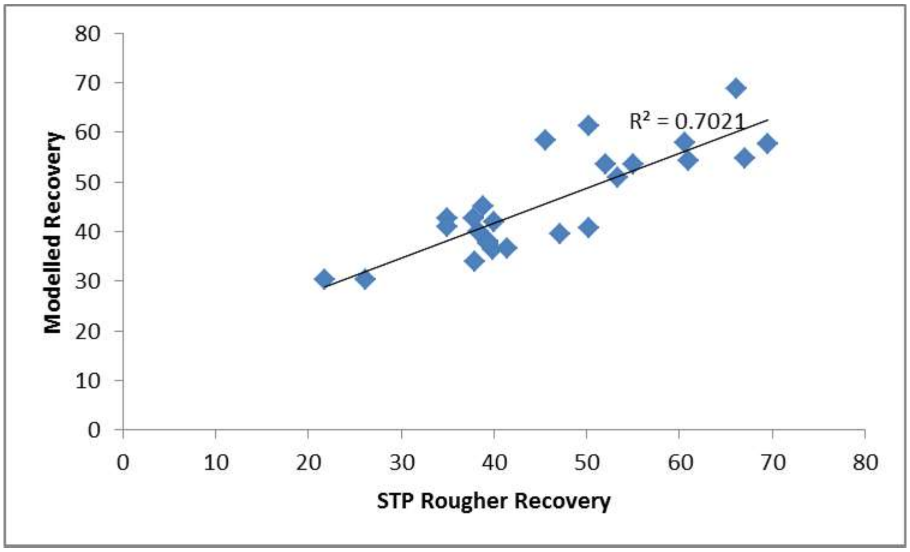


Source: RNC.

13.8.1.4 Results for High Iron Serpentine: FESP >= 14%

Rougher Ni Recovery = 14.83 + 38.9*S/Ni + 0.0143* Cr

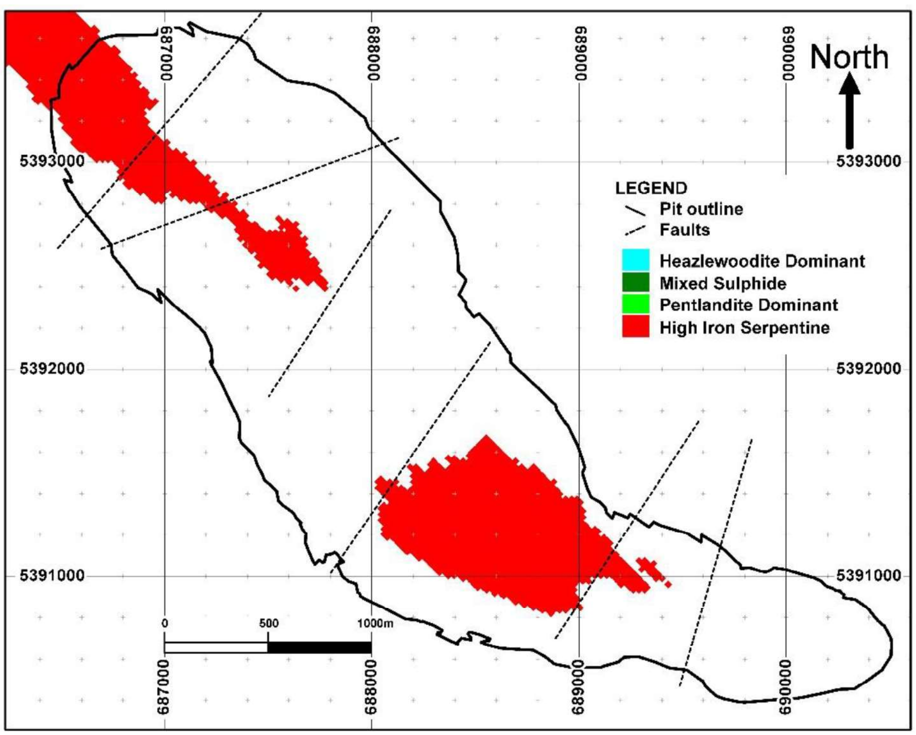
Figure 13-19: Recovery Regression Model for High Iron Serpentine



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

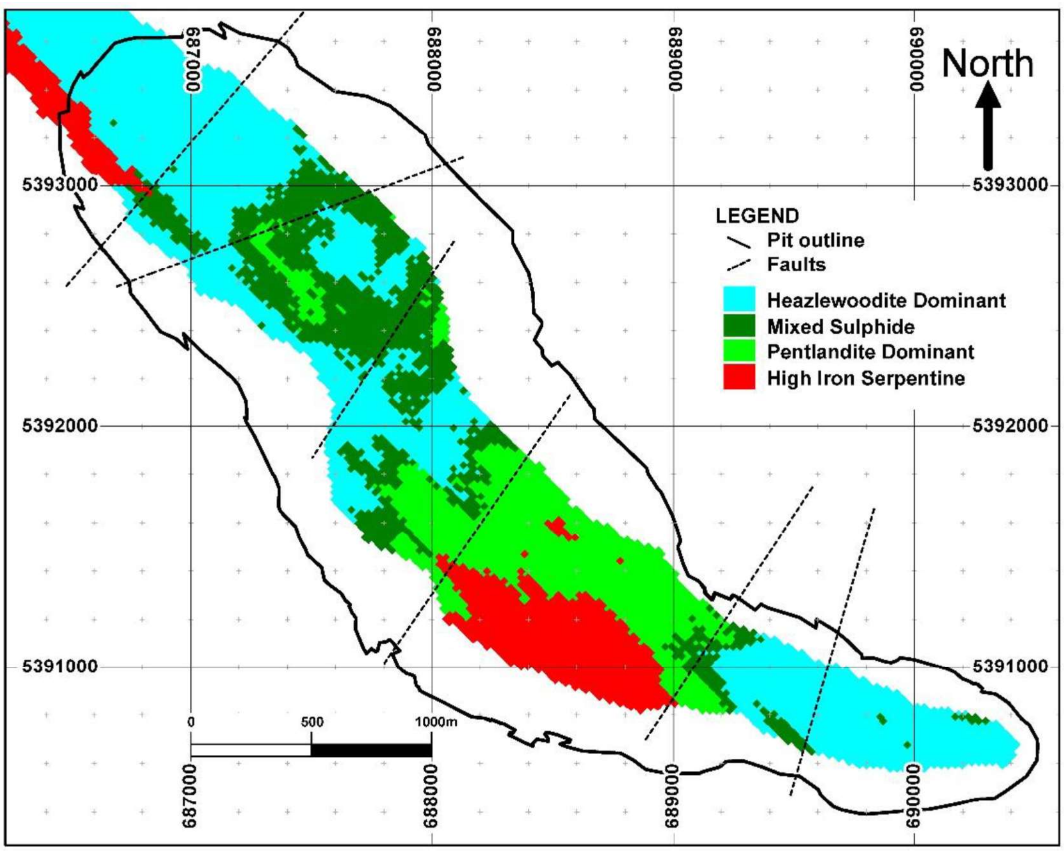
- High Iron Serpentine Domain (FESP >=14%)

  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 13-35.

The cleaner recoveries in Table 13-37 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).

Cleaner Ni Recovery = 0.1215 ln(%S) + 1.0959

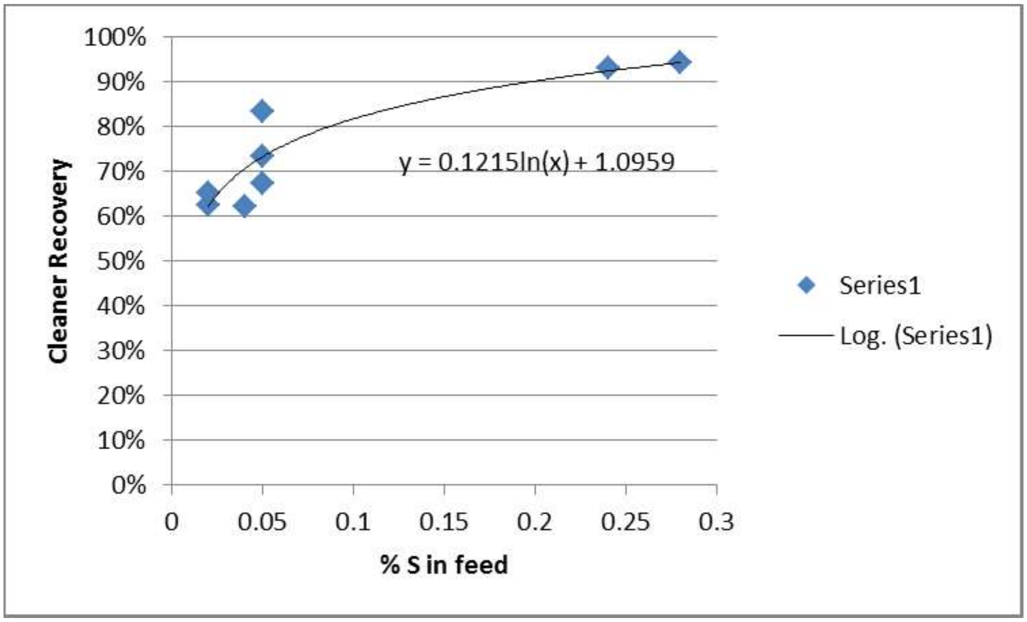
This equation was applied on a block-by-block basis to the Mixed Sulphide, Pn Dominant and Iron Serpentine domains within the resource.

Table 13-37: 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 13-38. Additionally, more samples were tested as part of the locked cycle cleaning tests.

Table 13-38: 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%.

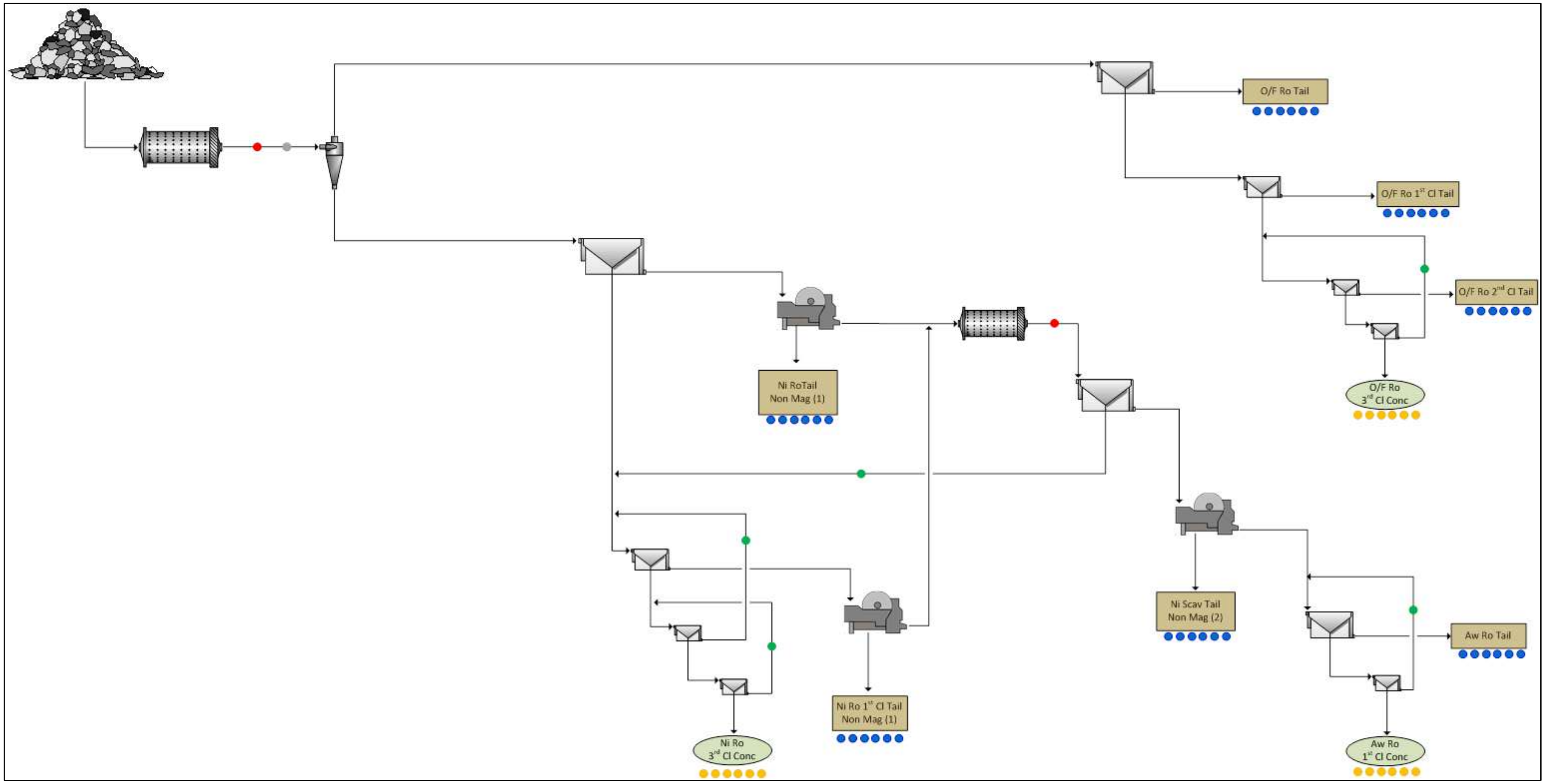
13.8.5 Confirmation of Flowsheet

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 13-39). 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 13-39: Reagent Consumption for the 2013 Locked Cycle Tests

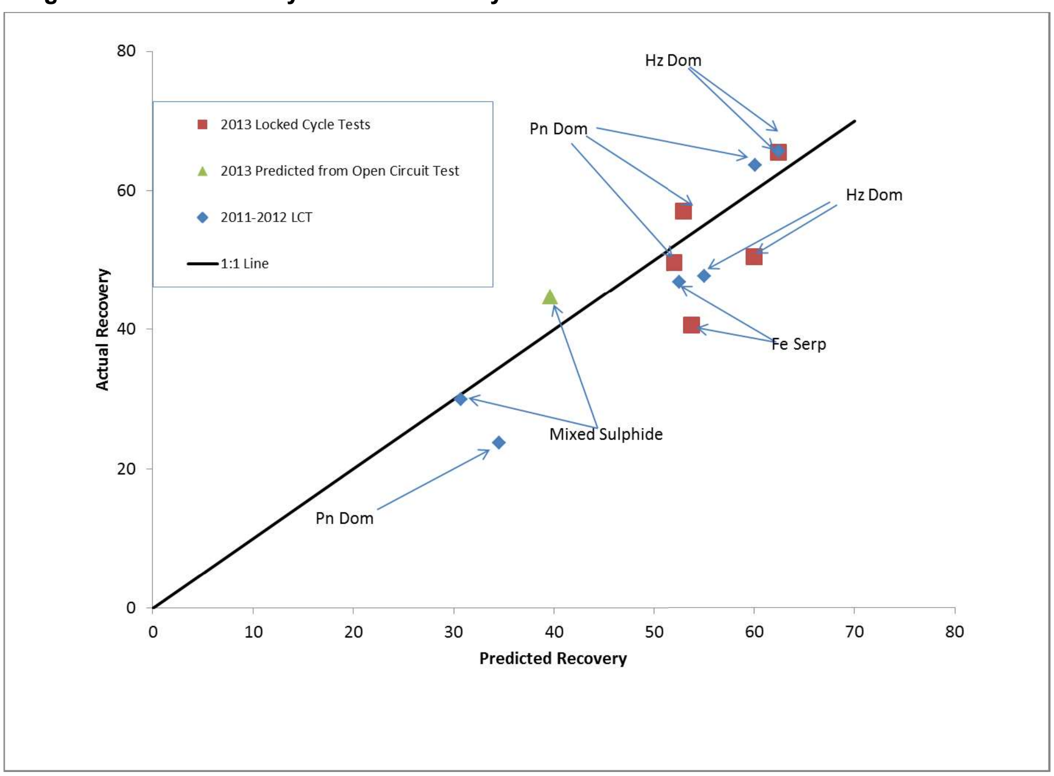
	PAX (g/t)	MIBC (g/t)	Cytec 65 (g/t)	Calgon (g/t)	CMC (g/t)	$H2SO_4$ (g/t)	Cost ($/t)
FS Opex	80	89	2	254	6	3888	1.41
Actual (average of all tests)	89	77	0	135	16	5100	1.25

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 test work. 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

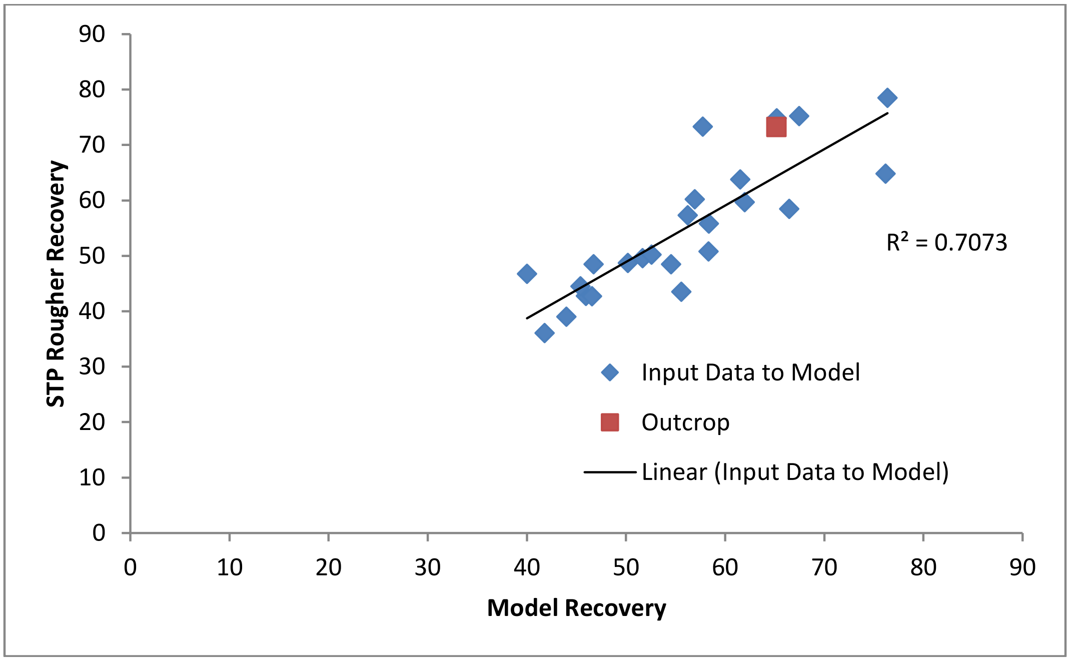
13.8.6 Effect of Stockpile Aging

Over the life of the mine, 511 Mt of ore will be stockpiled and subsequently processed by the mill at a later time, with an average duration on the stockpile of 13 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 By-product 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 13-40 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 33%.

Table 13-40: 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 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 13-41.

The calculated recoveries shown in 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 13-41: 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. Due to the proximity of the feed grade to the detection limit and sampling variability between the feed sample and concentrate sample, some recovery numbers are calculated as greater than 100%. The recoveries for Pt and Pd were downgraded in the model to account for these errors.

Source: RNC.

Table 13-42: 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%

Calculated based on units in concentrate / units in feed. Due to the proximity of the feed grade to the detection limit and sampling variability between the feed sample and concentrate sample, some recovery numbers are calculated as greater than 100%. The recoveries for Pt and Pd were downgraded in the model to account for these errors.

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 13-43 summarizes the results.

Table 13-43: 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 34%.

Other impurities—such 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 Dewatering Test work

In 2015 test work was performed on the concentrate generated from the pilot plant to determine the thickening and filtration characteristics. The sample was sent to Outotec's Burlington laboratory. The following are summaries from the Outotec reports (Ho, 2016).

The first phase of test work evaluated 4 different flocculants to determine which flocculant was most suited to thickening Dumont concentrate. Test work showed BASF MF-342 produced the fastest settling rate and the clearest overflow so it was chosen for the dynamic test work campaign.

The dynamic thickener test work on the Dumont concentrate sample was conducted in a 100mm apparatus. Six dynamic tests were conducted on the sample. A summary of the results is listed in Table 13-44.

Table 13-44: Summary of Dynamic Thickener Tests

pH	Solids Loading Rate (t/m^2h)	Rise Rate (m/h)	Flocculant Dosage (g/t)	Achievable Underflow Density (%w/w solids)	Achievable Overflow Clarity (ppm TSS)	Maximum Unsheared Underflow Yield Stress (Pa)
9.6	0.15-0.25	1.9-4.5	5-10	46-60	30-540	248

Results from the six dynamic thickener tests conducted indicated that final achievable underflow densities ranged from approximately 46 to 60%. Compression simulation tests were conducted on test #3, which resulted in an increase of 3-4% underflow density.

From this test work, 60% w/w solids were chosen for the design basis, with a flocculant dosage of 10g/t and a flux rate of 0.25 t/m^2h.

13.8.9.1 Concentrate Filtration Test work

A filtration test work campaign was conducted on a nickel concentrate from the Dumont project. The sample was thickened in Outotec's 100mm dynamic test unit prior to filtration. Tests were conducted in Outotec's Larox 100 test unit in order to determine the design criteria and suitability of pressure filtration technology for the sample.

Bench scale testing was conducted to evaluate filter sizing and criteria for the Dumont sample. Testing for the nickel concentrate samples yielded the follow results (Table 13-45).

Table 13-45: Summary of Concentrate Filtration test work (Ho, 2016)

pH	Air Drying Time (min)	Filtration Rate ($kgDS/m^2h$)	Filter Cake Moisture (%w/w water)	Filter cake Thickness (mm)	Pumping Pressure (Bar)	Pressing Pressure (Bar)	Air Pressure (Bar)
9.6	0.5-5.0	158-418	12.6-19.6	33-57	6	7-12	7-9

From this test work, and from further discussions with Outotec, the design basis of 10% w/w solids, at a filtration rate of 210 $kgDS/m^2h$, with a cycle time of 12 minutes was chosen.

13.8.10 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.

In 2016, a sample of the bulk concentrate was sent to SGS Minerals to determine the flow moisture and transportable moisture for the concentrate. This testing determines the maximum moisture that can be allowed for bulk shipment to prevent the risk of liquification and potential risk to the shipping vessel.

The test work showed the following results (Table 13-46) which indicate that at the current expected moistures seen from the filtration test work there should be no issues with bulk transportation of the concentrate.

Table 13-46: Results of Flow and Transportation Moisture from SGS Minerals

	Method Reference	Results (%)
Flow Moisture	IMSBC 2013	18.2
Transportable Moisture	IMSBC 2013	16.4

13.8.11 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.

13.9 Generation of Bulk Concentrate for Roasting Tests and Roasting Test Results

In September – December 2015 a pilot plant was conducted at SGS- Lakefield on the Outcrop sample (Hz Domain only) to generate concentrate for downstream testing. The ore for this test work was blasted in 2011 from the only outcrop on the property, located in far southeastern extent of the pit.

The pilot plant treated ~300 tonnes of material in order to generate sufficient concentrate for roasting test work and samples for customer testing. Metallurgical data was gathered on the slimes and rougher circuits. The Aw circuit was not tested due to the absence of Aw in the sample.

13.9.1 Pilot Plant Summary

13.9.1.1 Introduction

In the spring of 2015 a decision was made to complete a pilot test to process ~ 300 tonnes of Dumont material. The goal of the pilot plant was to produce large quantities of concentrate for roasting test work.

As a side benefit, additional information on the metallurgical performance of this sample would be gathered, as historically this outcrop has always posed greater metallurgical challenges that other samples tested to date.

13.9.1.2 Ore Preparation

SGS started receiving the 300 tonnes of the Dumont ore labeled as Outcrop in mid-August 2015. The bulk sample was dumped in a designated area as it arrived throughout the week, during this time the crushing circuit was being setup. A 50 kg sample was taken for Bond Ball Work Index (BWi) and Bond Rod Work Index (RWi) to characterize the ore grindability as well as an additional sample from the pilot plant rougher tails. The results are shown below. The ore is substantially harder and more competent in this limited area of the mine than seen in the other 102 grindability samples. This is expected, given that the area is the only Outcrop of the Dumont deposit, and therefore has been more resistant to weathering than the remainder of the deposit.

Table 13-47: Grindability test work for Outcrop Sample

Composite	A x b	RWI (kWh/t)	BWI (kWh/t)
RNC Outcrop head sample	31.1	19.4	31.1
Rougher Tail (Mag Feed)			38.8*

* Using a modified comparative BWI procedure

In mid-October crushing of the bulk sample to minus ¼” was completed. Air monitoring for fiber content was being conducted during the arrival of the ore and through the crushing. From the crushed product 200 kg was taken as a representative sample of the 300 tonnes for laboratory work before piloting. The 200 kg was crushed to less than 1/12” and split into 15 X 10 kg and 25 X 2 kg charges for the lab work. A head sample was also taken for initial chemical analysis.

The average head assay results for the pilot plant were 0.38% Nickel, 4.44% Iron, and 0.09% Sulphur. These are similar to previous samples from the Outcrop area and are as expected.

13.9.1.3 Laboratory Test Work

Pilot plant set up began in mid-September and lasted up until the end of October. During this time laboratory work designed to calibrate equipment and characterize the sample for piloting had started.

Laboratory Grind Size Determination

In September grind calibration of the primary grind consisting of 4 X 2 kg and 3 X 10 kg grind tests was conducted. For each of the 7 grinds a sample was taken, filtered and dried, to measure the particle size distribution (PDS). Air monitoring was present during the grinding work at the laboratory and during sample preparation once dried.

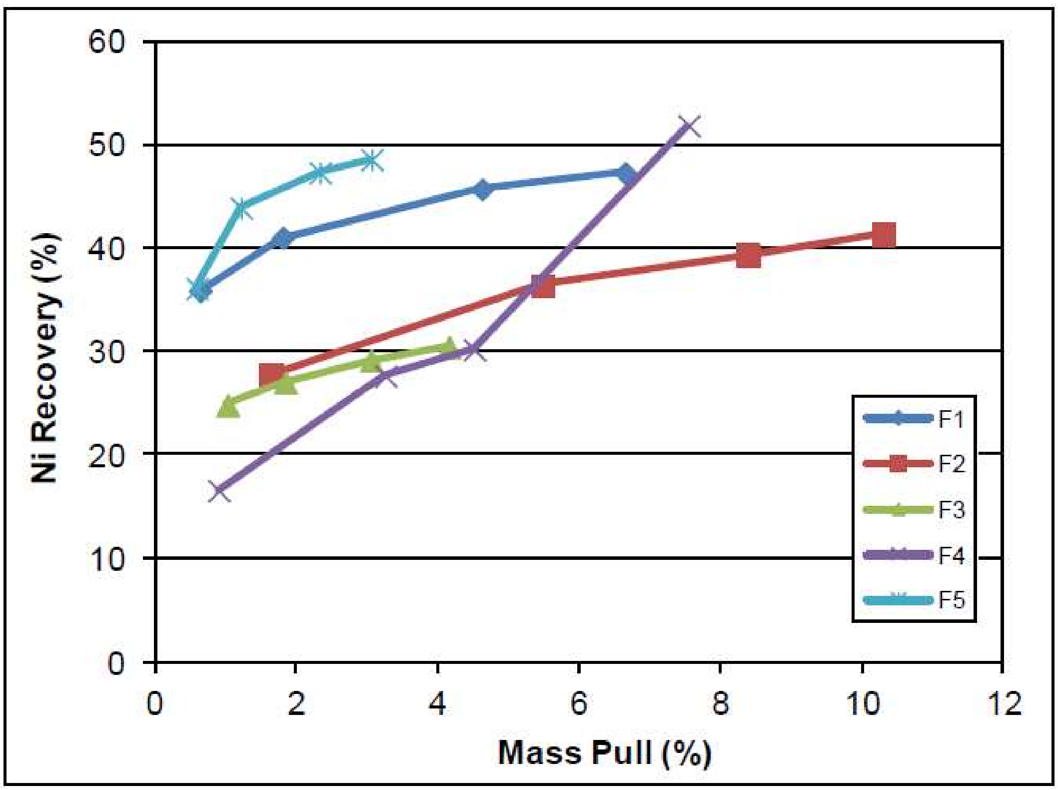
Desliming Test Work

Hydrocyclone calibration tests were conducted using 2 series of tests, (2 X 10 kg), to determine the parameters needed to get the correct mass split of the deslime circuit. The deslime work consisted of grinding the samples to the correct size determined by the grind calibration work and performing a series of samples cuts using different cyclone parameters. Each sample cut was filtered, dried and weighed. A total of 14 samples were produced for the hydrocyclone calibration. For each product a PSD was also measured.

Rougher Test Work

Following this work 3 rougher flotation tests were conducted to look at Ni recovery and flotation kinetics using different PSD and for comparative work from previous studies. Each test required 10 kg of feed material, ground and deslimed using the conditions determined during the equipment calibration work. The deslimed product was then floated using predetermined reagents conditions followed by a magnetic separation of the rougher float tails. For each test a PSD was measured, and 7 samples were produced. The samples were filtered, dried and assayed for Ni to calculate the nickel grade and recovery to the rougher concentrate under different conditions. The results are summarized below in Figure 13-26.

Figure 13-26: Rougher Flotation Test Results



The complete Dumont circuit incorporates a scavenger circuit where a regrind of the magnetic product from the rougher tail is required. Before performing a lab test representing the full circuit, regrind calibration work was needed. Using the magnetic product from the kinetics tests 3, grind tests were conducted to determine the grindability of the magnetic product in the 2 kg grind mill.

Locked Cycle Test

Results from all the lab work conducted throughout the month of September were analyzed and used to prepare for a locked cycle test (LCT). The (LCT) was used to simulate the full Dumont circuit in steady state using the determined conditions from the lab work to preview the expected results from the pilot plant and or to identify any issues that may occur during the pilot plant program. The LCT was conducted in the beginning of October. A total of 6 cycles, 10 kg each, were performed using the full Dumont circuit and conditions (rougher scavenger, and cleaners). Results showed a stable circuit with results similar to the predicted recoveries from the rougher work. The LCT work also added confidence before running the circuit on at the larger pilot plant scale. The Aw circuit was included in the LCT but not included in the pilot plant given the very small portion of Aw in the Outcrop sample.

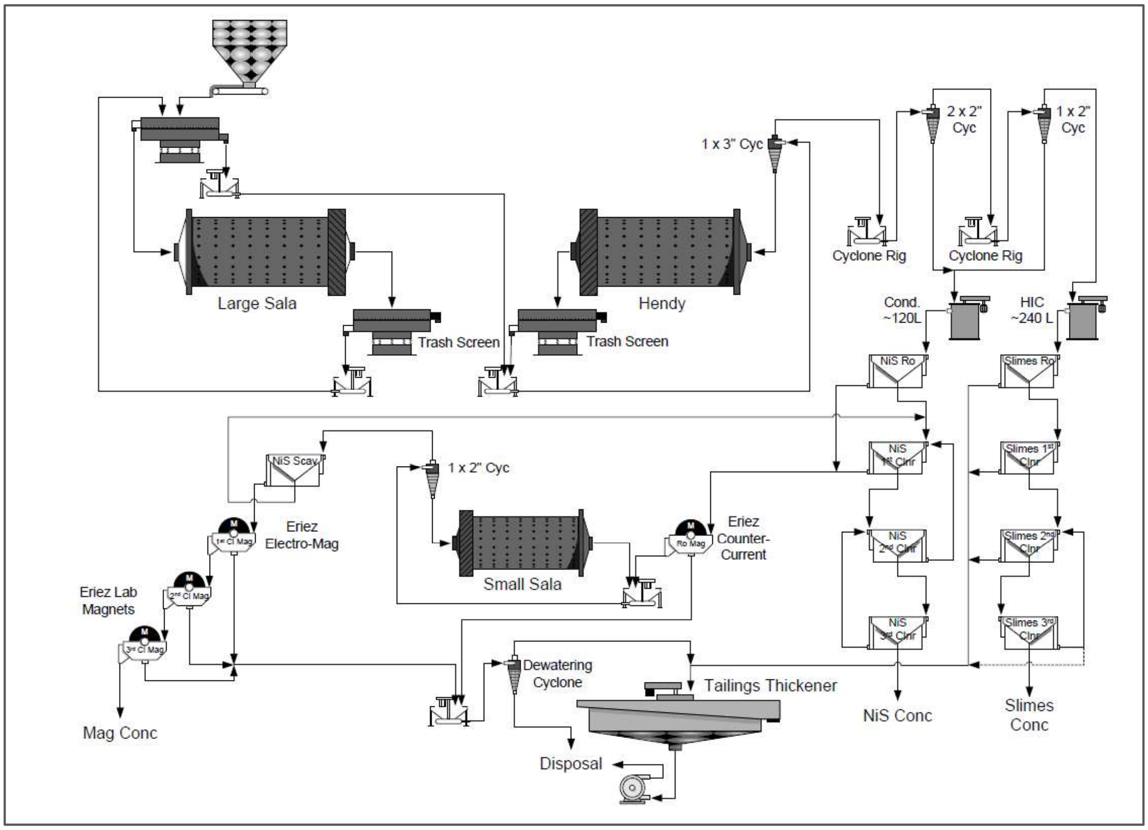
Table 13-48: Metallurgical Balance With O/F and Aw Circuits

Product	Weight		Ni %	
	g	%	Grade	Dist.
U/F 3rd Cl Conc	**174**	**0.60**	**30.0**	**44.7**
O/F 2nd Cl Conc A-C	28.3	0.10	1.61	0.39
Aw Ro Conc	47.2	0.16	1.43	0.58
U/F Ro Tail Non-Mags	16679	57.6	0.18	25.7
U/F 1st Cl Tail Non-Mags	1570	5.42	0.27	3.61
Scav Tail Non-Mags	1690	5.83	0.19	2.75
O/F 2nd Cl Tail A-C	106	0.37	0.33	0.30
O/F 1st Cl Tail A-C	334	1.15	0.53	1.52
O/F Ro Tail A, B, C	6881	23.8	0.28	16.3
Aw Ro Tail	1452	5.01	0.34	4.22
Feed calc.	28961	100	0.40	100
Feed head	NA	NA	0.39	NA

Pilot Plant Summary

Towards the end of October enough information was gathered from the laboratory work and preparation for the pilot plant began. The pilot plant set up was completed and commissioning began in the final week of October and lasted 2 weeks. After commissioning the pilot plant campaign lasted about 4 weeks with a feed rate of 800 kg per hour.

Figure 13-27: 2015 Pilot Plant Flow Diagram



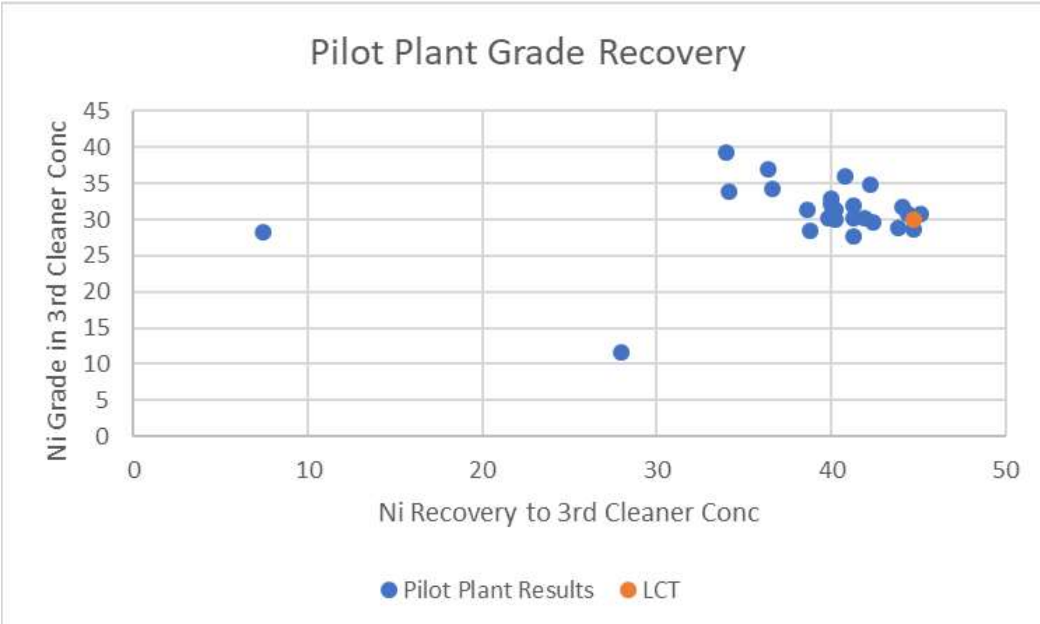
During the pilot plant runs, additional lab work was conducted on the slimes product and final cleaner magnetic product produced from the pilot plant. Scoping work was done on the slimes product to increase nickel recovery from this stream. Changes were made to the slimes circuit in the pilot plant using the conditions investigated in the scoping work. No nickel was expected to be recovered from the final cleaner magnetic concentrate of this bulk sample therefore the awaruite flotation circuit was taken out of the pilot plant circuit.

In operation in the pilot plant, the desliming circuit proved to be the most challenging to stabilize and control. There was no on-line or immediate ability to measure the split between the underflow and the overflow which made understanding the performance of the circuit difficult to control. The split varied even under the same set up due to pumping and inlet water pressure variation. This was a good learning for the full-scale plant design, as we will have the ability to install flow meters and density gauges on both streams which will give the operators real time feedback.

Incorrect splits impact residence times in both circuits, as well as the ability to recovery nickel sulphide from the slimes streams. Although the primary goal was to generate concentrate, a secondary goal was to demonstrate circuit recovery and ability to upgrade to 30% Ni in concentrate. Several optimized runs achieved the laboratory LCT recovery on the pilot scale. Once this was the achieved, focus returned to running the feed through the plant as quickly as possible and generating concentrate for the last several campaigns.

Figure 13-28 illustrates all the results from the 12-hour campaigns on the Dumont Outcrop sample. Although, there is some scatter, the locked cycle test falls on the same grade-recovery curve seen in the pilot plant results.

Figure 13-28: Grade Recovery of Pilot Plant vs. Locked Cycle Test



In general, the visual results from the pilot plant operation showed a relatively stable froth even in the low recovery slimes cleaners. Achieving target concentrate grade was not an issue and most campaigns had an average concentrate grade of >30% nickel.

The bulk Ni concentrate produced from the pilot plant and some of the tailing product were collected to be sent to different labs for additional work such as transport and dewatering characteristics as previously discussed in this chapter.

Processing of the 300 tonnes bulk sample was completed in early December.

13.9.2 Dead Roasting Tests

13.9.2.1 Initial Scoping Roasting Tests

In 2014, two samples of Royal Nickel Dumont Concentrate were dead roasted to eliminate >99.9% of the sulphur to produce a calcine amenable to downstream processing. These samples were ~1kg each and where a composite of the concentrates generated from 2012/2013 locked cycle tests.

A high sulphur removal was achieved when the concentrate had good contact with air, at temperatures greater than 1000°C, and when there was sufficient residence time.

Calcine sample assays were below the detection limit of 100 ppm phosphorus. Metal produced from Dumont concentrate will be low in P and S provided that a carbonaceous reductant low in P and S is sourced.

This initial work did prove that concentrate generated from the Dumont deposit was amenable to dead roasting and that the resulting sulphur content was less than 0.5% as required by the nickel pig iron, FeNi and stainless-steel industry.

13.9.2.2 Bulk Roasting Tests in the 2” and 12” Roaster

The bulk concentrate was shipped to XPS Consulting and TestWork Services in Sudbury, Ontario. The concentrate sample was dried and homogenized.

In 2015, RNC had contracted XPS to build a 12" fluid bed roaster to complete this work. The roaster is shown below in Figure 13-29. The roaster is fed dried concentrate through an educator into the bed of the roaster. The roaster has natural gas to provide energy for heat up, although the roaster is designed to run autogenously once up to temperature. There is also the ability to add oxygen if required to assist with roasting and heat balance.

The fines from the top are captured in a cyclone and re-directed back to the roaster. The sulphur dioxide in the off gas is treated in a scrubber system. There is a bottom dump bed discharge.

Figure 13-29: 12" Roaster

To begin, samples were roasted in the 2" XPS roaster to provide operating parameters for the 12" pilot scale. Cold fluidization tests run in a smaller 2" diameter batch roaster aided in confirming the appropriate space velocity for processing the concentrate. The use of a synthetic olivine starter bed material was also an important factor in creating and maintaining a stable bed within the roaster. The product calcine will demonstrate bed yield, particle growth while demonstrating sulphur content in the product at various temperatures. Over a series of shorter dayshift campaigns in April 2016, the Dumont concentrate was successfully roasted to the target calcine sulphur specification of less than 0.15 wt% S in the 2” roaster with a sulphur elimination of greater than 99%.

Five pilot roasting runs were completed on RNC Dumont nickel concentrate in the newly constructed 12” roaster. The objective of the test work was to confirm the ability to produce a low Sulphur calcine on a larger scale as well as generating larger samples for customer testing.

Initial commissioning and testing of the 12” diameter roaster was problematic due to the difficulty in establishing a bed and maintaining the heat balance within the roaster. Modifications made to the roaster after the initial commissioning runs allowed many of the early challenges to be overcome.

Approximately 480 kg was processed through the 12” roaster in a series of five campaigns. The remaining concentrate was stored for future work.

Some of the roasted calcine was smelted to produce a FeNi product containing >80% nickel. Overall nickel recovery to metal was 99.4% with a nickel partition between metal and slag of 144. This result is within the normal metallurgical performance of the existing nickel laterite operations.

The average feed grade and results from the 12” roaster test work is shown below in Table 13-49. Due to the length of each campaign (6-8 hours), there is some minor dilution from the olivine starter bed, but this isn’t considered a problem for the end user, as they are very insensitive to MgO and SiO_2.

Table 13-49: Feed and Calcine Assays

	Ni (%)	Fe (%)	Co (%)	Cu (%)	Mg (%)	Cr (%)	P (%)	S (%)
RNC Dumont Ni Concentrate	35.2	7.6	0.07	0.14	11.1			13
RNC Dumont Roasted Calcine	26.3	7.1	0.05	0.11	17.4	0.145	0.046	0.3

Samples from the three campaigns were sent to various potential customers in Europe and Asia to determine their suitability for direct usage to produce nickel pig iron, FeNi and stainless steel. Positive feedback was received.

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. As no new drilling information has been collected, this block model and corresponding Mineral Resource Statement was used to support the feasibility study update. 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 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). An update to the parameters of the block model definition was completed by Chelsey Protulipac, P.Geo (APGO #2608). 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 a 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, Ms. Protulipac, 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 with an effective date of May 30th, 2019, is 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. The elements mineral resource estimate for the Dumont Project includes nickel, copper, platinum and palladium, but does not include magnetite.

Table 14-1: Dumont Nickel Project, Quebec, SRK Consulting (Canada) Inc., May 30th, 2019*

Resource Category	Quantity (kt)	Grade		Contained Nickel		Contained Cobalt	
		Ni (%)	Co (ppm)	(kt)	(Mlbs)	(kt)	(Mlbs)
Measured	372,100	0.28	112	1,050	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)	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)	(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 percent nickel inside conceptual pit shells optimized using nickel price of US$7.50 per pound, average metallurgical and process recovery of 43 percent, processing and G&A costs of US$4.33 per tonne milled, exchange rate of C$1.00 equal US$0.77, overall pit slope of 42 degrees to 50 degrees depending on the sector, and a production rate of 105,000 tonnes per day. The qualified person considers that the conceptual pit shells would not be materially different to that if current (2019) conceptual pit optimization assumptions were considered. The technical parameters would be unchanged and with the metal price in Canadian dollars constant due to the decrease in US$ nickel price assumption compensated by corresponding decrease in US$:CAD$ exchange rate, the qualified person considers the reporting cut-off grade of 0.15 percent nickel to be reasonable. Values of cobalt, palladium, platinum and magnetite are not considered in the cut-off grade calculation as they are by-products 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 coextensive models with identical dimensions 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 focused 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.

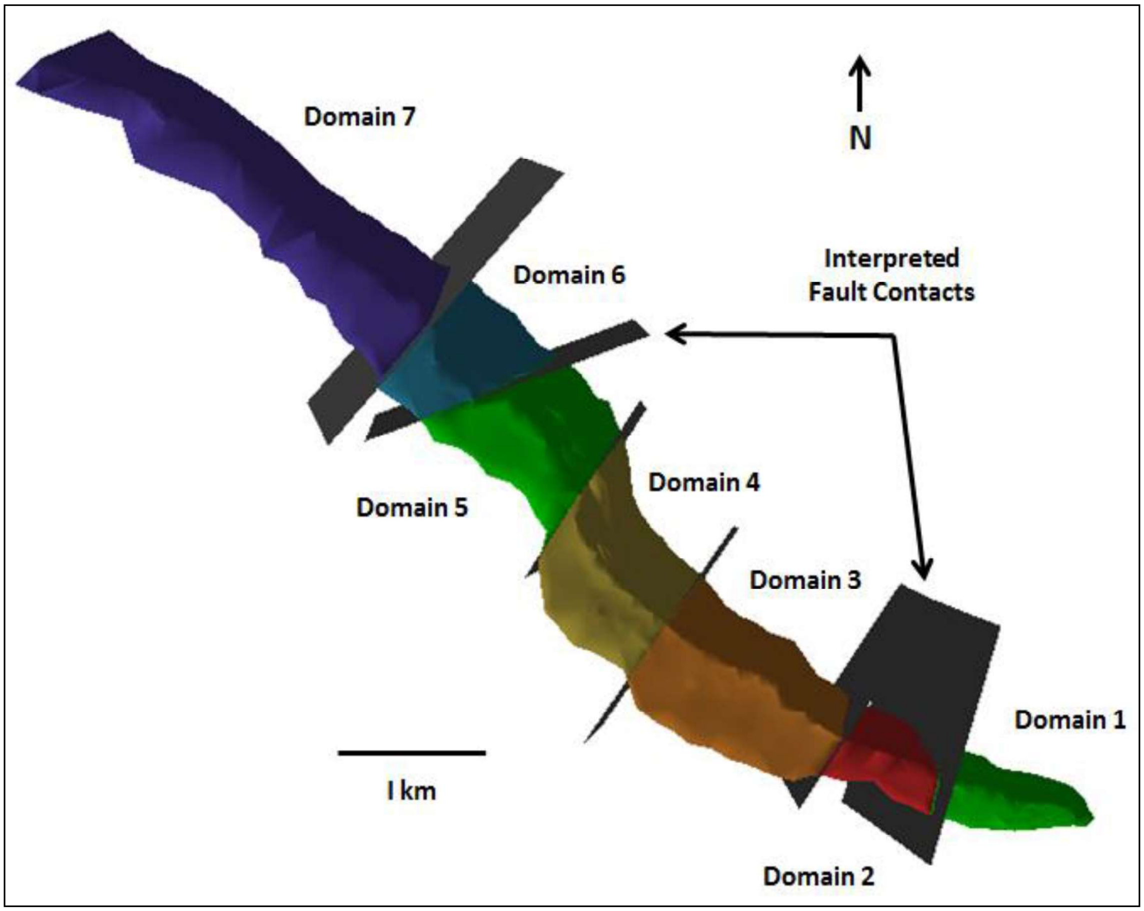
This section describes the resource domains used to constrain the estimation model, the available assays for analysis, compositing methodology, and the treatment of outliers for subsequent modelling. In addition, specific gravity data and its consideration in this resource estimate are also discussed.

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. SRK updated the seven mineralized envelop wireframes in correspondence with the Itasca structural model. Changes were limited to the boundary between the domains and the dunite contact.

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.

Figure 14-1: Distribution of the Seven Mineralized Envelopes Used as Resource Domains to Constrain Resource Estimation



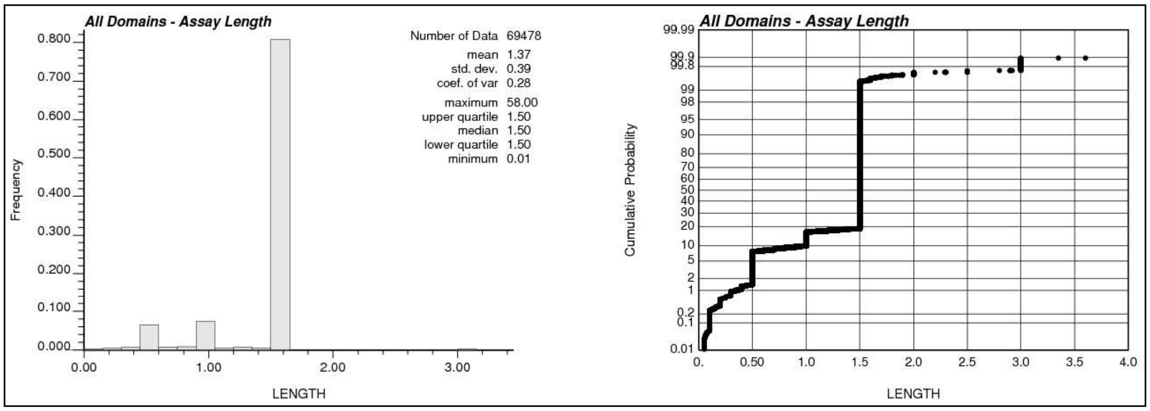
Source: SRK.

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 14-2) 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 14-2: Detection Limit Values

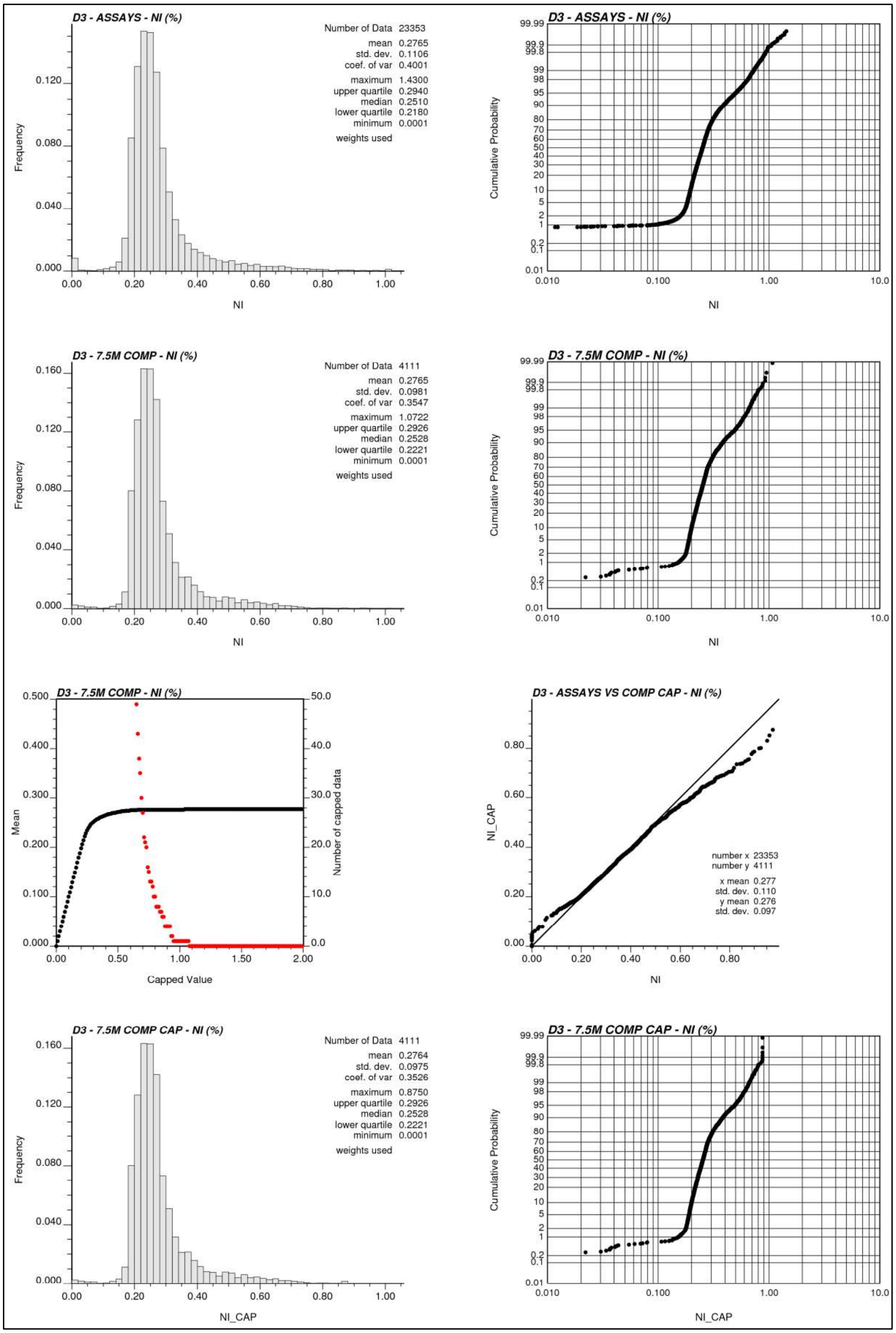
Element	Value	Unit	Element	Value	Unit
Ca	0.01	%	Ni	1	ppm
Co	1	ppm	Pd	0.001	ppm
Cr	1	ppm	Pt	0.005	ppm
Fe	0.01	%	S	0.01	%

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 14-3.

Figure 14-3: Basic Statistics for Nickel in Domain 3



Source: SRK.

Table 14-3: Capping Values for Each Domain

Domain		Element							
		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 comparing the summary statistics for percent nickel is shown in Table 14-4.

Table 14-4: 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 14-5.

Table 14-5: 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).

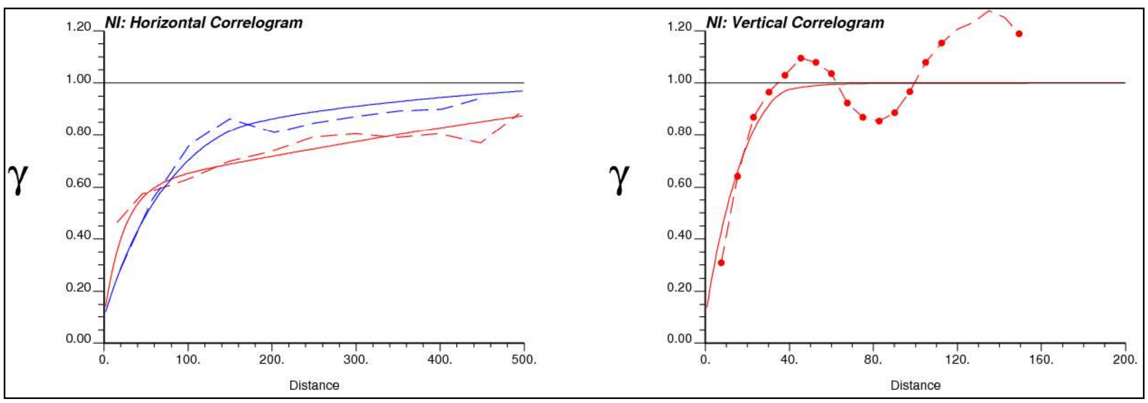
Variogram modelling was based on the combination of the four metrics, and in almost all cases, the correlogram of the main element yielded reasonably clear continuity structures that are amenable to variogram fitting.

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 blocks were rotated 45 degrees on the vertical (Z) axis, aligning the block edges with the strike of the modelled mineralization. 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 14-6.

Table 14-6: Dumont Block Model Characteristics

	Rotation (degrees)	Block Size (m)	Origin* (m)	Extent (m)	Number of Blocks
X	0	20	683,010	692,200	450
Y	0	20	5,394,510	5,390,975	200
Z	45	15	-700	425	75

Note: *UTM coordinates (NAD83 datum, Zone 17)

14.2.3.1 Estimation Strategy for Main Elements

Table 14-7 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 14-7: 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
Minimum Number of Octants	3	N/A	N/A
Minimum Number of Composites per Octant	2	N/A	N/A
Maximum 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. The 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 14-8 provides statistics on the percentage of the block model filled by estimation pass on the basis of the nickel block model.

Table 14-8: 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 metre or 100 metre sections. SRK considers the results of this study still valid and appropriate for resource classification.

Using the results of the 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 imply 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

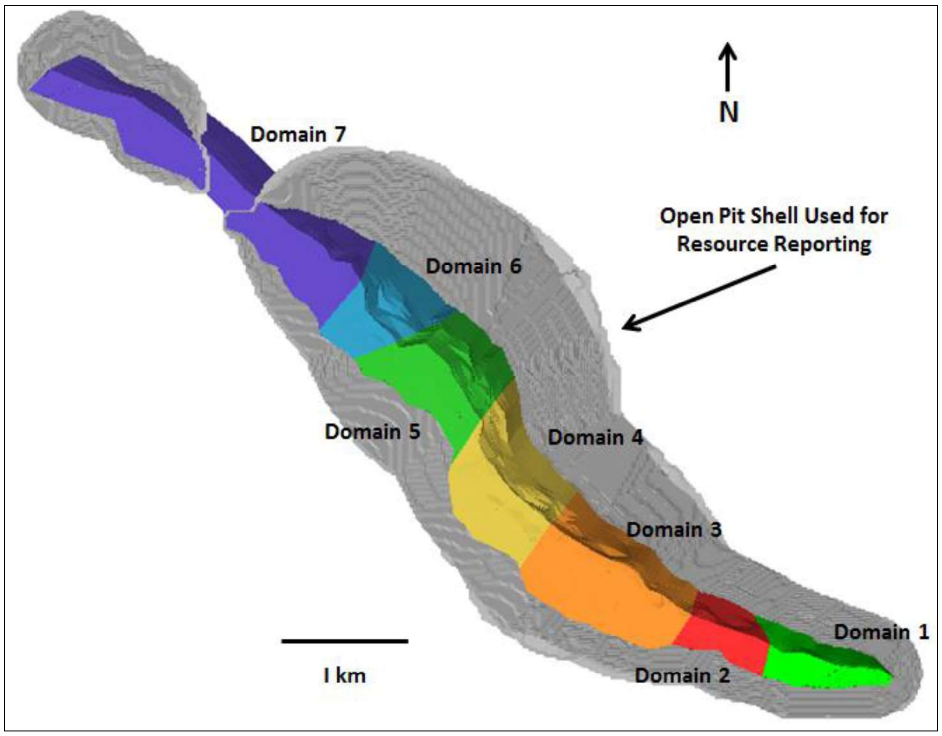
Table 14-9. The qualified person considers that the conceptual pit shell used to constrain reported mineral resources in 2013, would not be materially different to that if current (2019) conceptual pit optimization assumptions were considered. The technical parameters would be unchanged and with the metal price in Canadian dollars constant due to the decrease in US$ nickel price assumption from 2013 to 2019 ($US9.00/lb to US7.75/lb) being compensated by a corresponding decrease in US$:CAD$ exchange rate (0.90 to 0.75). The qualified person also considers the reporting cut-off grade of 0.15% nickel to still be reasonable. 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 14-9: 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 14-10 and as an effective date of May 30th, 2019.

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 14-11 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 14-10: Mineral Resource Statement, Dumont Nickel Project, Quebec, SRK Consulting (Canada) Inc., May 30th, 2019 *

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		

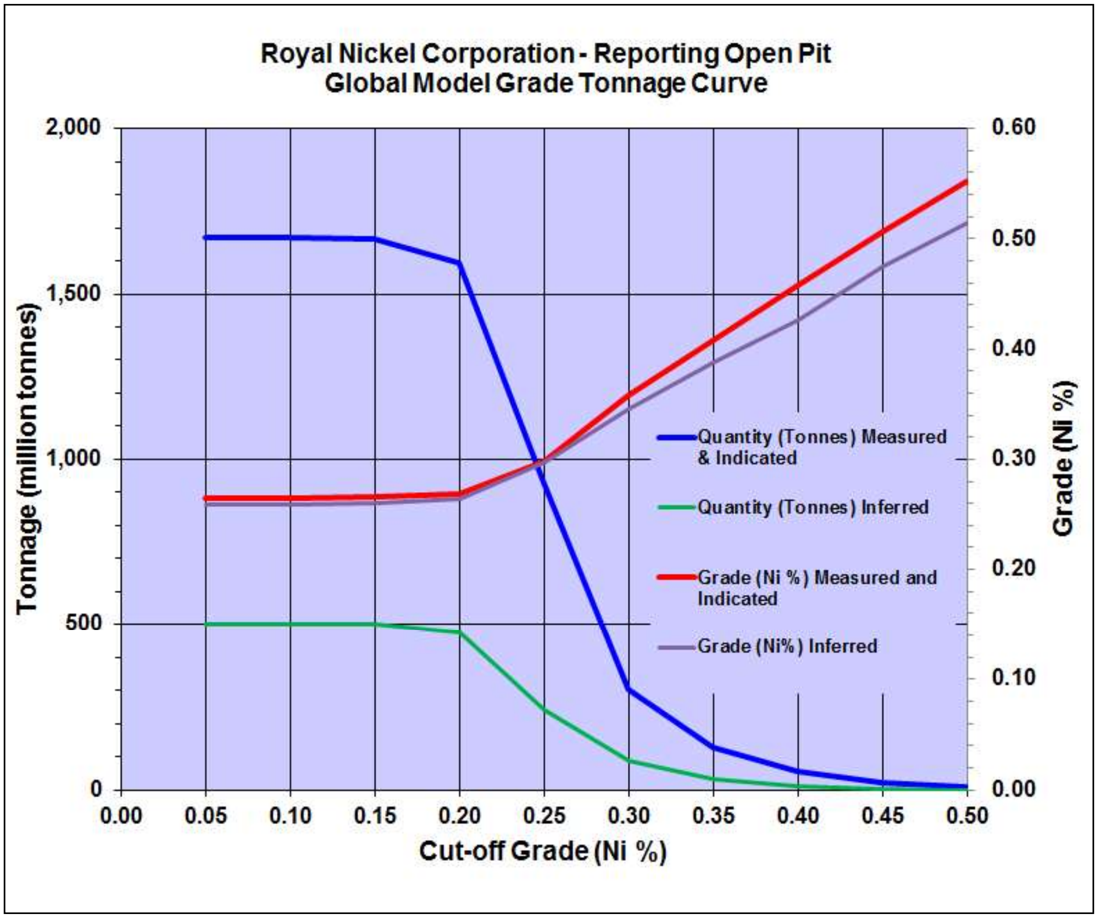
Note: *Reported at a cut-off grade of 0.15% nickel inside conceptual pit shells optimized using nickel price of US$9.00 per pound, average metallurgical and process recovery of 40%, processing and G&A costs of US$6.30 per tonne milled, exchange rate of C$1.00 equal US$0.90, overall pit slope of 42° to 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 by-products 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 14-11: Inpit 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 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* (30 May 2019)[1]

		Grades				Contained Metal			
Category	(kt)	Ni (%)	Co (ppm)	Pt (g/t)	Pd (g/t)	Ni (Mlb)	Co (Mlb)	Pt (koz)	Pd (koz)
Proven	163,140	0.33	114	0.013	0.031	1,174	41	67	162
Probable	864,908	0.26	106	0.008	0.017	4,908	202	220	466
Total	**1,028,048**	**0.27**	**107**	**0.009**	**0.019**	**6,082**	**243**	**287**	**627**

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$4.05 per pound, average metallurgical recovery of 43%, marginal processing and G&A costs of US$4.10 per tonne milled, long-term exchange rate of C$1.00 equal US$0.75, overall pit rock slopes of 40° to 50° depending on the sector, and a production rate of 105 kt/d. Mineral Reserves include mining losses of 0.33% and dilution of 0.43% that will be incurred at the contact between mineralization and waste. The life of mine stripping ratio is 1.02:1. 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 estimated by Dave Penswick, P.Eng. These are based on the mineral resource block model described in the previous chapter. Reserves are contained within an engineered pit design that is based upon a Lerchs-Grossmann (LG) optimized pit shell generated using a nickel price of US$4.05/lb, which is considerably lower than the long-term forecast of US$7.75/lb. Reserves include dilution of 0.43% and mining losses of 0.33%.

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.

The Base Case assumes all concentrate would be roasted, resulting in recovery of only nickel. At higher prices for by-products than currently forecast, it will be more economic to treat all or a portion of the concentrate via conventional smelting and refining, resulting in economic recovery of cobalt, platinum and palladium. Additionally, Dumont reserves contain 44.9 Mt of potentially economic magnetite.

15.2 Reserve Estimation Process Overview

To ensure the scope of design selected for Dumont was optimal, reserves were estimated using an iterative process. This process can be summarized as follows:

- The feasibility study (FS) mine design uses the resource block model described in the previous chapter. This model includes the estimated content of the economic metals nickel, cobalt, platinum, palladium and iron (contained in magnetite). The resource block model also included the estimated recovery of each economic non-ferrous 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 base case design assumes all concentrate will be roasted, resulting in a higher payability and lower realization charges for nickel, at the expense of by-product credits. Alternate scenarios that are described in Chapter 24 included the impact of smelting all or a portion of the concentrate produced, with associated lower payability and higher realization charges partially offset by by-product credits.
- The Lerchs-Grossmann algorithm (LG) was employed in a two-stage process to define the optimal final pit shell to be used as the basis for a subsequent engineered design. The initial stage (the 'Penultimate') used the block model developed for the 2013 FS along with slope recommendations and cost estimates derived from that earlier study. Nested shells generated for the Penultimate case were evaluated using a techno-economic model. Output from this evaluation was an optimal pit shell approximately 20% smaller and containing 10% less ore than the 2013 FS design. This evaluation also identified the optimal development sequence and associated cost structure. These criteria, along with updated slope recommendations that were based on the rock units in which the Penultimate case final walls were located, were applied to the rotated block model described in Chapter 14 to produce the 'Ultimate' case.
- An engineered pit design was produced for the Ultimate case. This design used inter-ramp angles as recommended by the geotechnical consultants and ramps of sufficient width for the 290 t trolley-equipped class trucks planned for use.
- Unplanned dilution and mining losses were applied to the reserve estimate to reflect the potential for additional dilution and losses that would occur when mining at the contact between mineralization and waste.
- 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. It should be noted there is limited material affected by the selection of cut-off, as increasing the cut-off grade to $10/t reduced total ore by 3.2% while reducing NPV by 0.5%.

15.3 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 Table 15-2. It should be noted that key macro-economic assumptions, including Ni price and long term C$ f/x rate, were revised during the course of study. Current assumptions (Ni = US$7.75/lb and C$ = US$0.75) result in a higher NSR value per block than used in the calculation of block values. Re-running the NSR calculation using the current macro-economic assumptions would thus result in some blocks defined as sub-economic being included in the economic reserve. However, and as noted above, the tonnage of marginal value mineralization is small and would not have a material impact on overall economics.

Key assumptions used in the NSR calculations include the following:

- The base case design assumes all concentrate would be roasted at a facility located in China. The potential economic contribution, and associated realization charges, from by-product cobalt and PGEs has been excluded from the calculation. The base case also assumes no production of saleable magnetite. The impact of these by-products and associated assumptions are discussed in Chapter 24.
- Table 15-2 reports the average concentrate grade over the entire life-of-project. The scheduled concentrate grade ranges from a low of 22% to a high of 34%.

Table 15-2: Dumont NSR Calculation for Nickel

Item	Units	Base Case (Roasting)
Long-term Ni Price[1]	US$/lb	US$7.50
Long-term C$ F/X[1]	C$1.00 =	US$0.77
Concentrate Grade	% Ni	29.3%
Concentrate Transport[2]	US$/t	U$93
Concentrate Treatment[3]	US$/t	n/a
Ni Refining[2]	US$/lb	n/a
Payables[3]	% of contained	91.5%

Notes: 1. Macro-economic assumptions used in calculating NSR values were subsequently updated for purposes of economic evaluation. 2. Transportation costs include rail to Quebec (denominated in C$ but for simplicity converted to US$ at assumed long term rate) and shipping to China. 3. The typical commercial terms for treatment of Ni Concentrate no longer express costs of treatment and refining as fixed rates but rather as a deduction in the percentage payables. The implied TC/RC for the assumptions given in Table 15.2 is US$0.45/lb Ni

15.4 LG Pit Shells – Penultimate Case

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 G&A from the NSR of each block (waste blocks with no NSR value have a negative net value).

The Penultimate LG run was performed using the 2013 FS resource model and cost assumptions that were escalated from the 2013 FS. These costs included:

- Mining costs comprised of:
  - a base cost of $1.50/t (for blocks lying at or above the elevation of pit exits); and
  - a cost increment of $0.05/t for every 15 m bench below the exit. Note this cost increment did not take account of the reduction in costs that would be realized from trolley-assisted haulage.
- Marginal processing costs of $5.00/t ore
- G&A costs of $0.50/t ore

Overall slope angles were assigned to the various sectors based on recommendations from the 2013 FS.

The LG algorithm then selected a ‘cone’ of ore and associated waste stripping that maximises NPV. By varying the Revenue Factor (RF, or percentage of the long term metal price), it was possible to generate higher value nested cones that can be used to identify the optimal development sequence. The smallest cone was generated with RF21 (21% of the US$7.50 Ni price = US$1.58). Shells were generated for each subsequent 1% increment in the Ni price then aggregated into 13 potential stages of mine development as summarized in Table 15-3. Note that the aggregation process took account of the practical limitations of mining with large equipment and was not simply based on the RF.

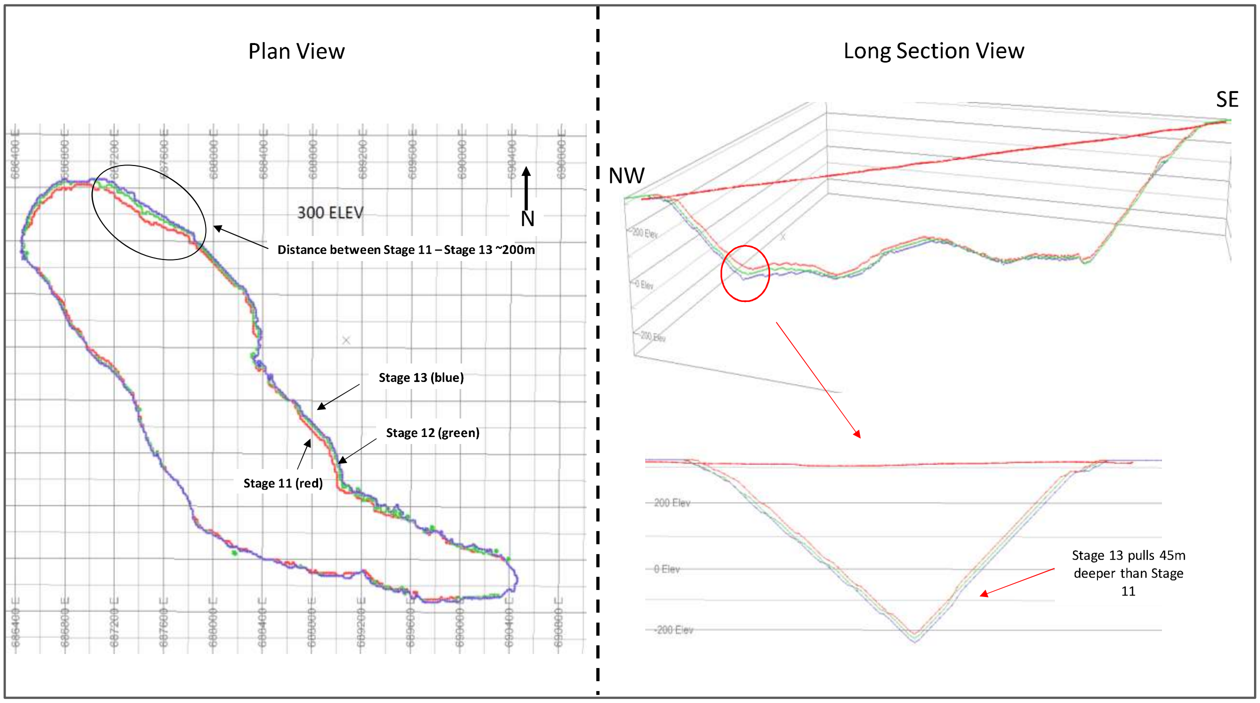
Table 15-3: Penultimate LG – Stages of Pit Development

Stage	RF	Mill Feed (Mt)	Waste Rock (Mt)	TOTAL (Mt)	Stripping Ratio	Grade % Ni	Avg. Recovery	NSR $/t
1	45%	8	9	17	1.24	0.246	49.3	$22.50
2	33%	131	100	231	0.77	0.288	46.1	$24.80
3	42%	106	251	356	2.37	0.274	44.8	$22.97
4	43%	69	55	124	0.80	0.270	37.3	$18.78
5	43%	183	83	267	0.45	0.256	39.5	$18.85
6	47%	72	152	223	2.11	0.253	38.3	$18.25
7	45%	87	30	116	0.34	0.248	42.6	$19.60
8	50%	55	103	158	1.87	0.251	37.1	$17.56
9	48%	47	43	90	0.92	0.294	45.3	$24.58
10	49%	159	6	165	0.04	0.284	47.1	$24.86
11	55%	137	154	291	1.13	0.275	44.7	$22.74
12	57%	81	124	206	1.53	0.258	40.8	$19.24
13	59%	78	122	200	1.57	0.259	39.5	$18.53
Total to Stage 11		**1,053**	**987**	**2,040**	**0.94**	**0.270**	**43.0**	**$21.65**
Total to Stage 13		**1,212**	**1,234**	**2,446**	**1.02**	**0.268**	**42.7**	**$21.29**
Selected as Optimal Final Pit Shell								

The thirteen stages were evaluated using a spreadsheet techno-economic model. The evaluation revealed that NPV increased fairly rapidly until Stage 11, then moderated but continued to increase through Stage 13. It was expected that when engineering constraints were applied to the design, the economic performance of Stages 12 and 13 would, at best, likely approximate that of Stage 11. The smaller stage was thus selected as the basis for the Ultimate LG run. Note that the evaluation did not continue past Stage 13 as the total tonnage and associated operating footprint approximated the limits of that for which permits have been awarded.

Figure 15-1 compares the footprint of Stages 11 – 13. The only area of material deviation between the stages is approximately 1 km of strike length in the north west, where the distance between Stage 11 and Stage 13 opens to 200 m. To provide the operating mine with the ability to later pushback to Stage 13 (should future economic conditions warrant), a buffer of 400 m has been used when siting infrastructure in this area.

Figure 15-1: Penultimate LG – Comparison of Stages 11 - 13



Source: RNC.

15.5 LG Pit Shells – Ultimate Case

The Ultimate case LG evaluation was performed using the updated resource model developed for the current FS. This model has been rotated. Compared to the un-rotated model, planned dilution with the new model is approximately 2% lower for the scope of design selected.

Cost assumptions used in the Ultimate case, were also updated, using output from the Penultimate case, which was based on the selected scope along with current quotes from suppliers of goods and services. The updated costs summarised below were not markedly different than those used in the Penultimate Case:

- Mining costs comprised of:
  - a base cost of $1.56/t (for blocks lying at or above the elevation of pit exits); and
  - a cost increment of $0.028/t for every 15 m bench below the exit. Note this cost increment did take account of the reduction in costs that would be realized from trolley-assisted haulage.
- Marginal processing costs of $5.07/t ore
- G&A costs of $0.56/t ore

Overall slope angles were updated with particular focus placed on:

- The upper 70m of the pit, where benches will be 10 m (discussed in Chapter 16)
- Zones of coalingite, where the slopes designs are more conservative than previously used.

Output from the Ultimate LG was evaluated in the same manner as the Penultimate, with RF 54 being selected as the basis for the engineered design. A comparison of RF 54 and Penultimate Stage 11 is given in Table 15-4.

Table 15-4: Comparison of Ultimate LG RF54 with Penultimate Stage 11

Item	units	Penultimate	Ultimate
Ore	Mt	1,053	1,075
Total Material	Mt	2,040	2,108
Strip Ratio	waste : ore	0.94	0.96
Grade	% Ni	0.268	0.270
Con'd Ni	Mlbs	6,227	6,391
Recovery	% of con'd	43.0	43.3
Rec'd Ni	Mlbs	2,681	2,770
NSR	C$ / t	$21.65	$21.86

15.6 Engineered Pit Design

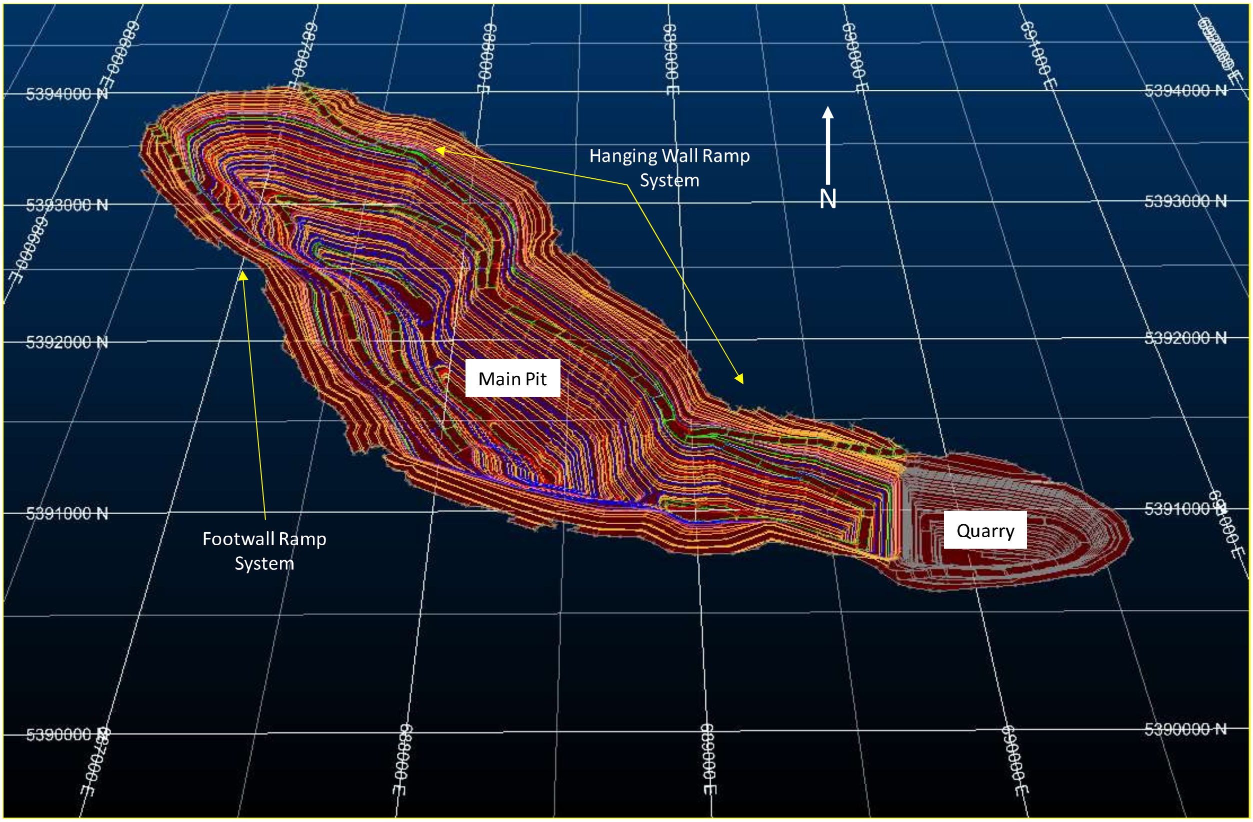
Pit shells generated using the LG algorithm represent a theoretical design and, while the final walls honour the imposed overall slope constraints, it cannot be considered a practical design as no provision is made for ramps. The engineered design includes ramps of the following widths:

- 37 m for 2-way traffic with 290 t trucks where trolley assist will not be utilized (the ramp width approximates 4.0x the running width of trucks and allows for 3 lanes of traffic with a ditch and berm)
- 42 m for 2-way traffic with 290 t trucks where trolley assist will be utilized
- 20 m for areas where low density traffic and/or 90 t trucks will be employed, including the initial phase of development that will subsequently be used for contingent water storage during the remainder of normal pit operations.

Figure 15-2 illustrates the final engineered pit. The following aspects of the design are highlighted:

- The ‘Quarry’ to the extreme south east is where mining initiates (as it provides the only outcrop). During the pre-strip period, approximately 14 Mt are excavated to create a reservoir of approximately 5M m^3 capacity. This reservoir is used to provide start-up water to the mill (before steady-state conditions are achieved in the TSF) as well as surge catchment during the annual freshet and other periods of heavy precipitation.
- The Main Pit is an excavation of 2,003 Mt that operates until the end of year 19. The design provides separate ramp systems for both the hanging wall and footwall. This reduces geotechnical risk as instability in one sector has a lesser impact when vehicles can choose from multiple exits.
- Following completion of mining in the Main Pit, tailings will be impounded in the resulting void and there will no longer be requirement for water storage in the Quarry. At this point, mining of the Quarry resumes, with a further 63 Mt excavated over a period of 4 years.

Figure 15-2: Engineered Final Pit Design



Source: RNC.

A comparison of the engineered design and Ultimate LG RF54 shell on which it was based is given in Table 15-5.

Table 15-5: Comparison of Engineered Design & Ultimate LG RF54 Shell

	Mill Feed (Mt)	Grade (% Ni)	Con'd Ni (Mlbs)	Recovery (% of con'd)	Rec'd Ni (Mlbs)	Waste (Mt)	Strip Ratio
Ultimate LG	1,075	0.270	6,391	43.3%	2,770	1,033	0.96
Engineered	1,028	0.268	6,082	43.1%	2,624	1,052	1.02
Variance	**-4.4%**	**-0.6%**	**-4.8%**	**-0.4%**	**-5.3%**	**1.9%**	**6.6%**

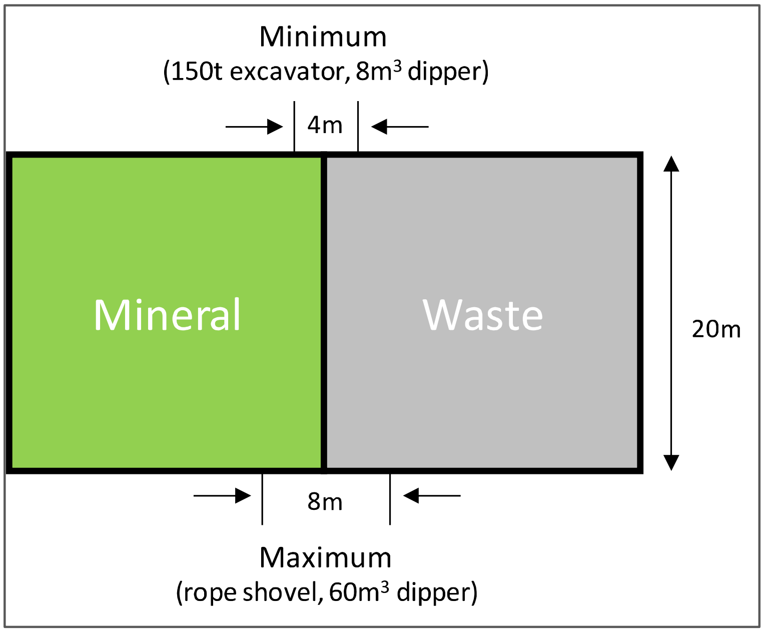
15.7 Dilution and Mining Losses

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.

Unplanned dilution and mining losses (defined as dilution factors not already inherent within the block model) will potentially occur at the contacts between ore and waste. Allowance has been made for the mixing of 2 – 4 m barren waste with an equivalent tonnage of mineralization on the other side of each contact (see Figure 15-3 below). The extent of mixing corresponds to the width of dipper on the various loading units that will be employed as well as the spacing of blast holes (approximately 2x the dipper widths and therefore the combined extent of unplanned dilution +

mining losses). As Dumont mineralization is disseminated with large continuous ore zones between hanging wall and footwall waste rock contacts, there is only a limited tonnage of mineralization located along contacts. As a result, overall dilution will be low at 0.43% over the life of mine. The trend in mineralization is gradational, with highest grades toward the interior of the mineralized zone. Consequently, the average grade at the contact is lower than the overall average and mining losses are low at 0.33% for an estimated mining recovery of 99.67%.

Figure 15-3: Dilution and Mining Losses at Ore-Waste Contact (Plan View)



15.8 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 (Ni only for the Base Case, but potentially including Co, Pt and Pd if the operating mine smelts all or a portion of concentrate; as well as Magnetite if that mineral is also recovered). 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., annual maintenance of the mill); and
- General & Administration.

Note that the marginal cost calculation excludes costs associated with the terrestrial TSF as this facility will have been decommissioned and tailings will be pumped into the mined out pit by the time the lowest value reserves are reclaimed from the stockpile.

As shown in Table 15-6, these costs total approximately $7/tonne. This theoretically calculated cut-off was also 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. It should be noted that there is only a limited tonnage of mineralization ranging from $6 - $10/t cut-off

(34 Mt, or ~ 3% of total reserves) and the impact on NPV of changing the cut-off to either $6 or $10 is less than 0.5%.

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. Table 15-6 summarizes this calculation, showing that the theoretical cut-off is 0.08% Ni.

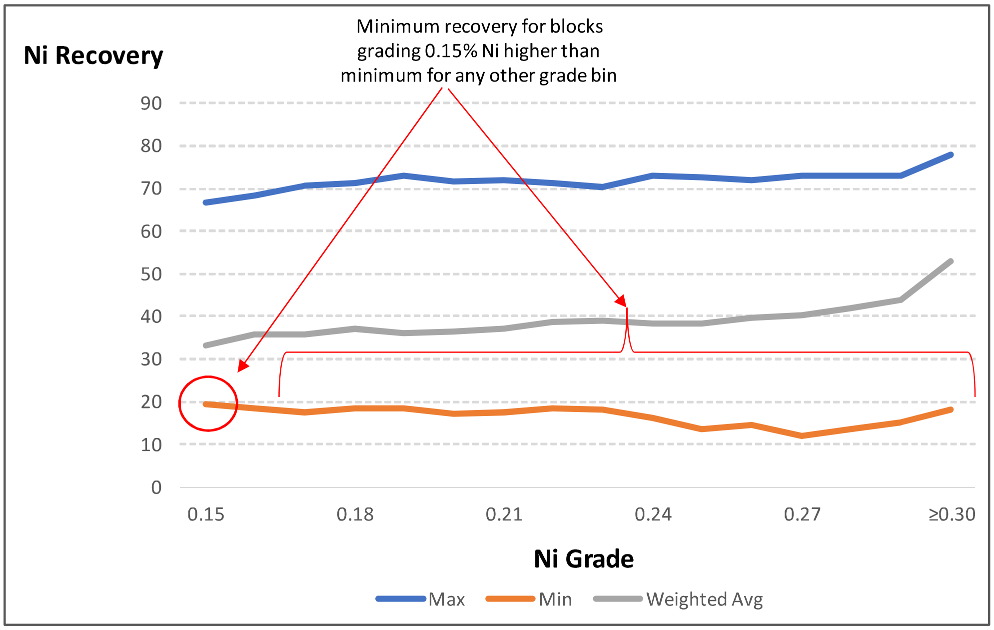
A key variable in the calculation shown in Table 15-6 is metallurgical recovery. Recovery is generally a function of mineralization and independent of grade – as shown in Figure 15-4; the minimum recovery for all blocks grading 0.15% Ni is actually higher than the minimum value for any other grade bin. However, to account for the possibility that lower grade mineralization may exhibit lower recovery, the theoretical cut-off of 0.08% Ni has been increased by approximately 80% 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.

Table 15-6: Cut-Off Grade Calculation

Parameter	Units	Value
Ni	US$/lb	$7.75
Realization	US$/lb	$0.45
NSR	US$/lb	$7.30
F/X	C$=	$0.75
Net Smelter Return	**$/lb**	**$9.73**
Mine Re-Handle[1]	$/t	$1.27
Mill Cost[1]	$/t	$4.95
G&A Cost[1]	$/t	$0.56
Sustaining Capital[1,2]	$/t	$0.23
Subtotal	**$/t**	**$7.01**
Payable Ni	lb/t ore	0.72
Payables	lb/t ore	91.5%
Recovered Ni	lb/t ore	0.79
Average Recovery	% of con'd Ni	43%
Contained Ni	lb/t ore	1.83
Head Grade	**%Ni**	**0.08**

Notes: 1. Final output from financial model for engineered design, 2. Sustaining Capital for period following pit closure (when lowest value mineralization reclaimed from stockpile)

Figure 15-4: Ni Recovery by Grade Bin



15.9 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 $12/t in the final two years of the mine life (Years 23 and 24), after the main pit has been depleted and the Quarry is expanded to its final limits. The highest value used as the ROM cut-off of $32/t, which will be used in Yr6 immediately preceding the expansion of the mill to 105 ktpd.

This variable cut-off is accounted in Table 15-7, which illustrates the conversion of resources to reserves. The open pit mine reserves declared in Table 15-7 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 15-7: Conversion of Resources to Reserves (figures do not add due to rounding)

Material Within	Measured Resources			Indicated Resources			Total		
Engineered Pit Shell	**Kt**	**% Ni**	**Ni M lbs**	**Kt**	**% Ni**	**Ni M lbs**	**kt**	**% Ni**	**Ni M lbs**
Resource	336,759	0.29	2,121	691,289	0.26	3,982	1,028,048	0.27	6,104
ROM	163,140	0.33	1,176						
Stockpile	173,620	0.25	946						
Dilution	1,370	0.00	0	3,039	0.00	0	4,410	0.00	0
ROM	293	0.00	0						
Stockpile	1,077	0.00	0						
Mining Losses	1,370	0.20	6	3,039	0.21	15	4,410	0.21	21
ROM	293	0.23	1						
Stockpile	1,077	0.19	5						
Proven Reserves	163,140	0.33	1,174				163,140	0.33	1,174
Probable Reserves	173,620	0.25	941	691,289	0.26	3,968	864,908	0.26	4,908
Total Reserves	**336,759**	**0.29**	**2,114**	**691,289**	**0.26**	**3,968**	**1,028,048**	**0.27**	**6,082**

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 based on the work that was undertaken for the 2013 feasibility study.

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 in 2013 (SRK (2012), referred to as the 2013 GW Model). Hydraulic parameters, taken from the field testing results, were assigned to the model domains. This model has since been updated to include the revised TSF designs (Golder (2019), referred to as the 2019 GW Model). However, at the time of this present study the 2019 Model had not been updated to incorporate the revised pit designs. The output from the 2019 model produced the following:

- Groundwater inflow to the pit (site water balance input) was estimated to range between 3,700 and 4,900 m^3/d during mine operations. As the 2019 pit design is approximately 50 m shallower, and of a similar footprint to that used in 2013, the estimate of groundwater inflow to the pit remains reasonable for the latest pit design; and
- Boundary conditions were used in pit slope pore pressure modelling for the geotechnical program.

Pore pressure modelling was carried out in 2D sections inherited from the slope stability models. Because the latest pit designs were not included in the 2019 GW Model, the boundary conditions extracted from the 2019 GW Model were specified distal to the pit crest. Specific scenarios were modelled including 'undrained' and 'drained' conditions to test the sensitivity of the slopes to various pore pressure conditions. 'Undrained' conditions assumed no increase in the permeability near the pit face due to blasting damage. Whereas 'drained' conditions assumed that the zone of highest damage close to the pit face was free draining. The delineation of the damaged zone was inherited from the slope stability model geometry, designated as D1 to D4, extending deeper into the simulated pit face as damage and dilation effects lessen with depth. These damage zones were of similar depth/extent in all sections and are discussed in more detail in Section 16.2.1.

Where the slope stability was shown to be sensitive to pore pressure, the position of the free draining surface was varied to the extent of the next deepest damage zone to provide an estimate of pore pressure conditions that achieved the required slope stability.

The pore pressure modelling did not explicitly model the estimated depressurisation of the pit face, but rather was used to inform the stability modelling of the expected pore pressure distribution at specific set backs from the face based on the damage zone limits. Therefore, the modelling should be looked at as not an assessment of what depressurisation may occur, but what will be required to obtain slope stability within the accepted FOS for the project.

Pore pressure modelling of the pit slopes indicated that only one section, HW-06, will require depressurization. In the vicinity of this section, the pit wall will need to be dewatered to approximately 30 m back from the slope. Pore pressure monitoring instrumentation, such as VWPs, will be required to monitor the performance of any dewatering system.

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.

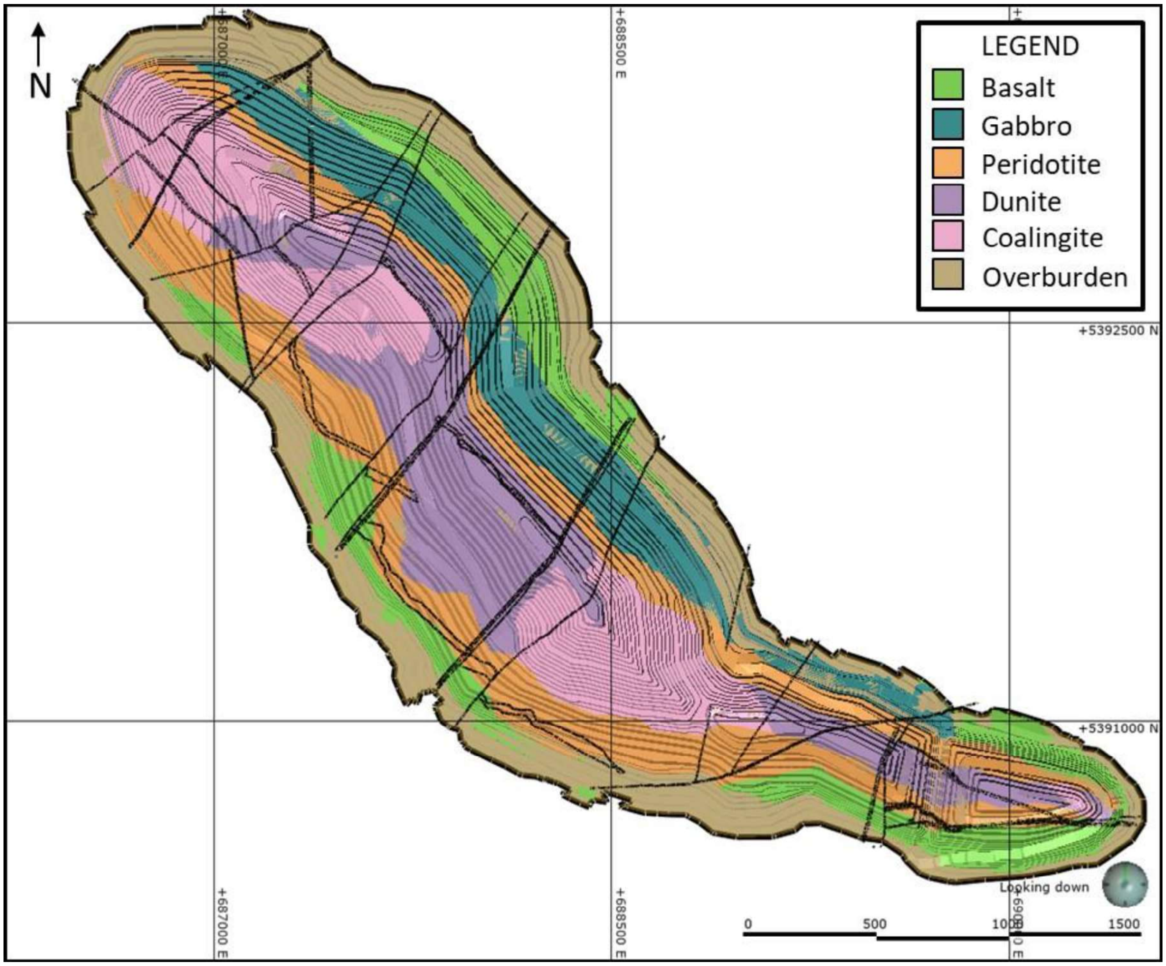
Three detailed structural geology studies have been undertaken on the Dumont deposit area: the first in 2010, the second in 2011, and the third, a feasibility level interpretation and consolidation of knowledge, in 2012. The deposit-scale structural geometries were modelled using regional geophysical 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, 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; a representative cross-section is given in Figure 16-2.

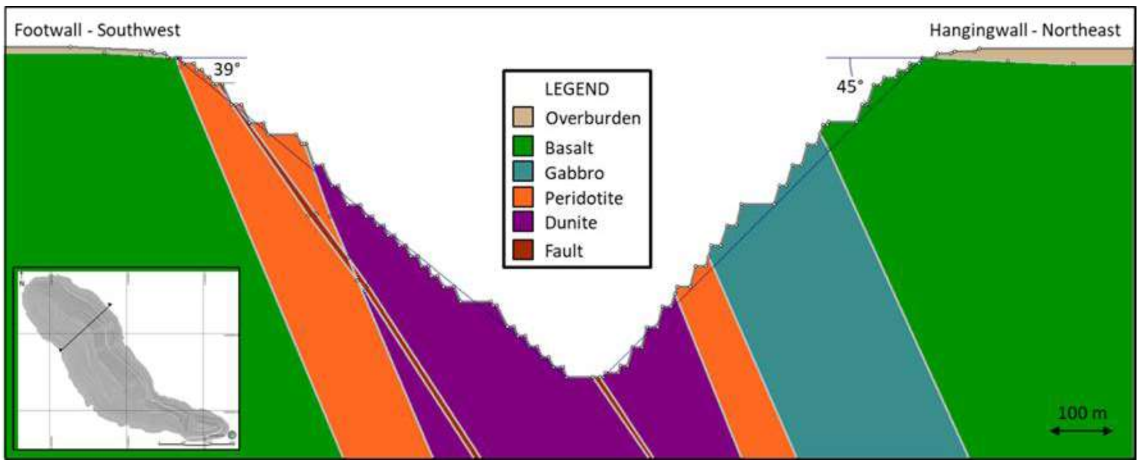
The structural investigation found three distinct structural domains for the Dumont open pit area. These domains are bounded by major structures and exhibit minor to moderate differences in joint and foliation orientation and properties. The northeast-to-southwest-trending faults are steeply dipping 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 representative cross-section is given in Figure 16-2.

Table 16-1: Representative Geotechnical Characteristics of Dumont Rock Types

Material	Specific Gravity (t/m^3)	Uniaxial Compressive Strength (MPa)	Fracture Frequency (ff/m)	Rock Mass Rating [RMR_{89}]
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 expected to be kinematic. The failure modes are anticipated to 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. Within the fault damage-zones and exposed (and activated) dun-CG domain rocks, unravelling on a bench-scale may occur.

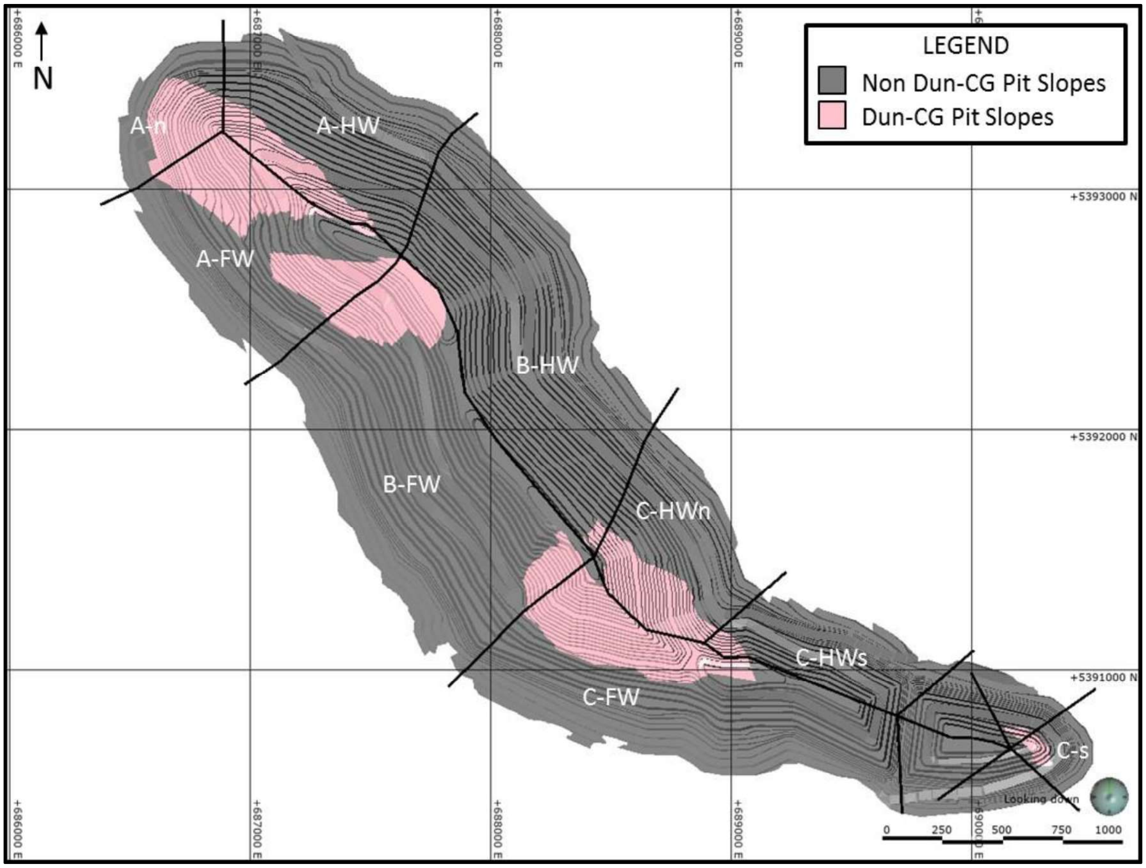
Considering the geotechnical domains and the likely slope directions, slope design sectors were generated, as seen in Figure 16-3, and design parameters developed for each design sector. The design parameters were based on a maximum stack height of 120 m, separated by a geotechnical safety berm of 20 m.

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 parameters listed above, 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 managed operationally.

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 Design Domain / Sector (Face Dip-Dir°)		Bench Height (m)	Bench Width (m)	Bench Face Angle (°)	Inter-Ramp Angle (°)
A-n (130), A-HW (200), C-HWn (250), C-s (270), and C-s' (090)	Non-CG	30	10.5	75	58
	Dun-CG	15	11.0		45
	Non-CG above RL 260	10	6.0		49
	Dun-CG above RL 260	10	7.5		44
B-HW (240) and C-HWs (200)	Non-CG	30	14.5	75	53
	Dun-CG	15	11.0		45
	Non-CG above RL 260	10	6.0		49
	Dun-CG above RL 260	10	7.5		44
A-FW (050) and C-FW (010)	Non-CG	30	10.5	70	54
	Dun-CG	15	10.5		43
	Non-CG above RL 260	10	6.0		46
	Dun-CG above RL 260	10	7.5		42
B-FW (060)	Non-CG	30	10.5	65	51
	Dun-CG	15	10.0		41
	Non-CG above RL 260	10	6.0		43
	Dun-CG above RL 260	10	7.0		41

Note that a) the maximum stack height allowed is 120 m, b) geotechnical berm-width is 20 m, c) single-benching is 15 m high, d) use single-benching one bench below and three benches above faults oriented within ±015° to the bench-crest azimuth, e) use single-benching for dun-CG domain, and f) double-bench only if pre-split blasting is 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 damage 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 occur. Allowance has been made for the possibility of this toppling in the design and it may be an area for improved slope angles once initial slopes have been established in the various domains and the rock mass performance assessed on a bench and inter-ramp scale.

16.2.3 Recommendations

As the pit is being excavated, the extent of horizontal fracturing related to unloading needs to be assessed. If the extent of fracturing extends deeper than currently anticipated, the use of 10 m bench heights may have to be extended deeper.

There is some potential for more faults within the proposed pit area than have already been interpreted. Further work (drilling and geological/geotechnical mapping) is required to satisfactorily understand the structural geology of some portions of the deposit area. The possible face parallel fault structures along the upper west wall need to be further investigated soon in the mine life, to assess the potential threat to that upper wall in and around the main access ramp.

During construction, the slopes established in the southeast pit should be used as an opportunity to investigate the behaviour of the rock mass (in terms of failure mechanisms) 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.

The extent and influence of the Dun-CG unit and the time dependent effect of its degradation on bench and ramp stability needs to be investigated and evaluated during the early exposures in that unit.

The south-east quarry pit should be kept pumped empty to avoid increase slope water pressures in the adjacent pit slope face to the north.

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 was observed to extend 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, typically ranges in thickness between 1 and 16 m but is usually around 5 m thick. The silt consistency varies 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.

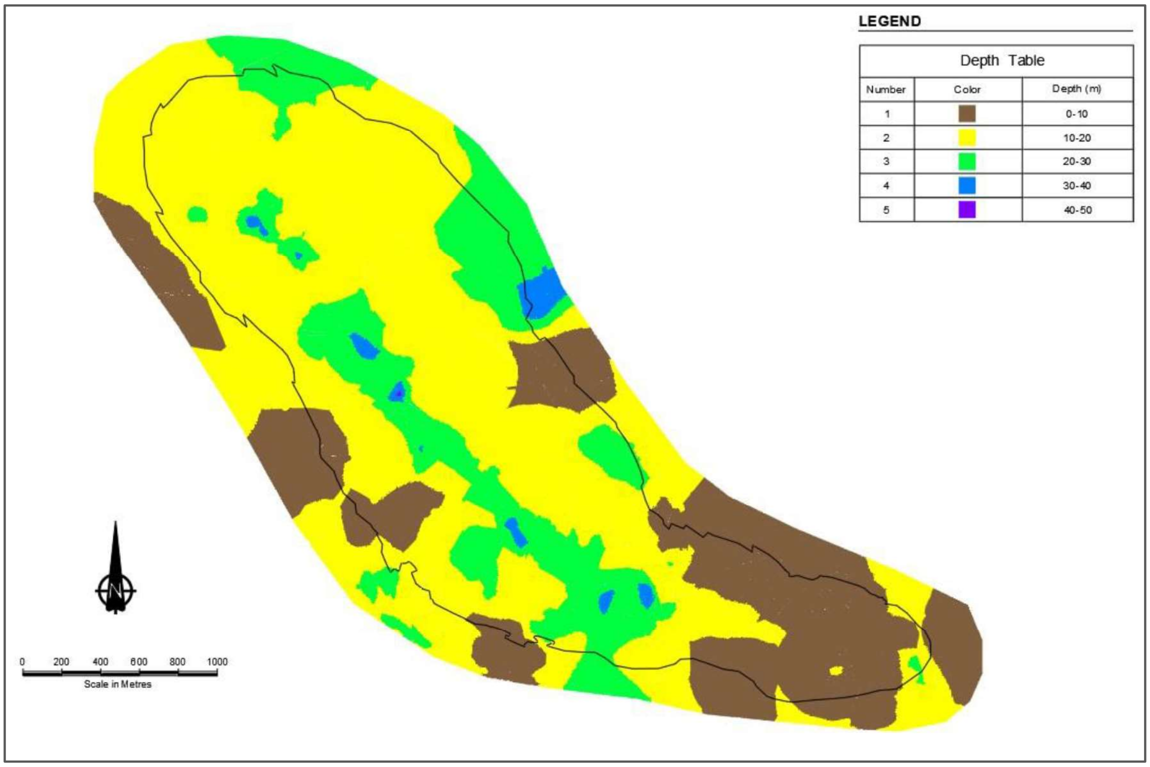
16.2.4.1 Database

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

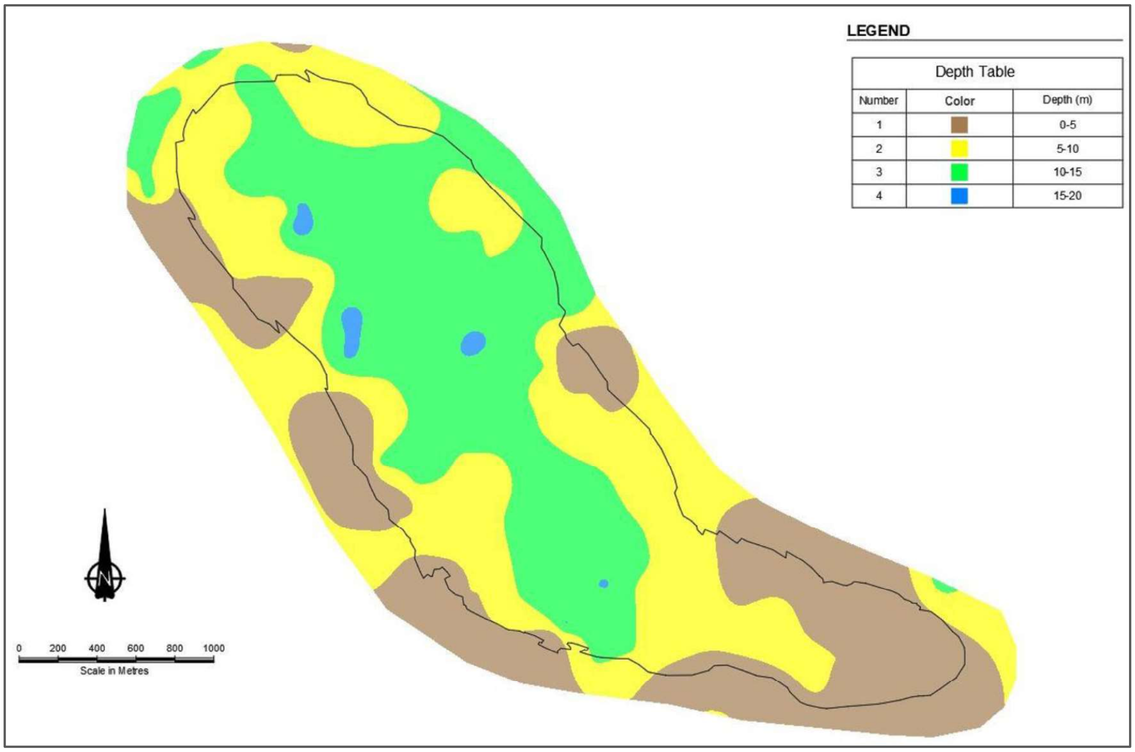


Source: SRK.

In general, where the overburden is less than about 7 m in thickness, the soil profile typically 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 about 7 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.

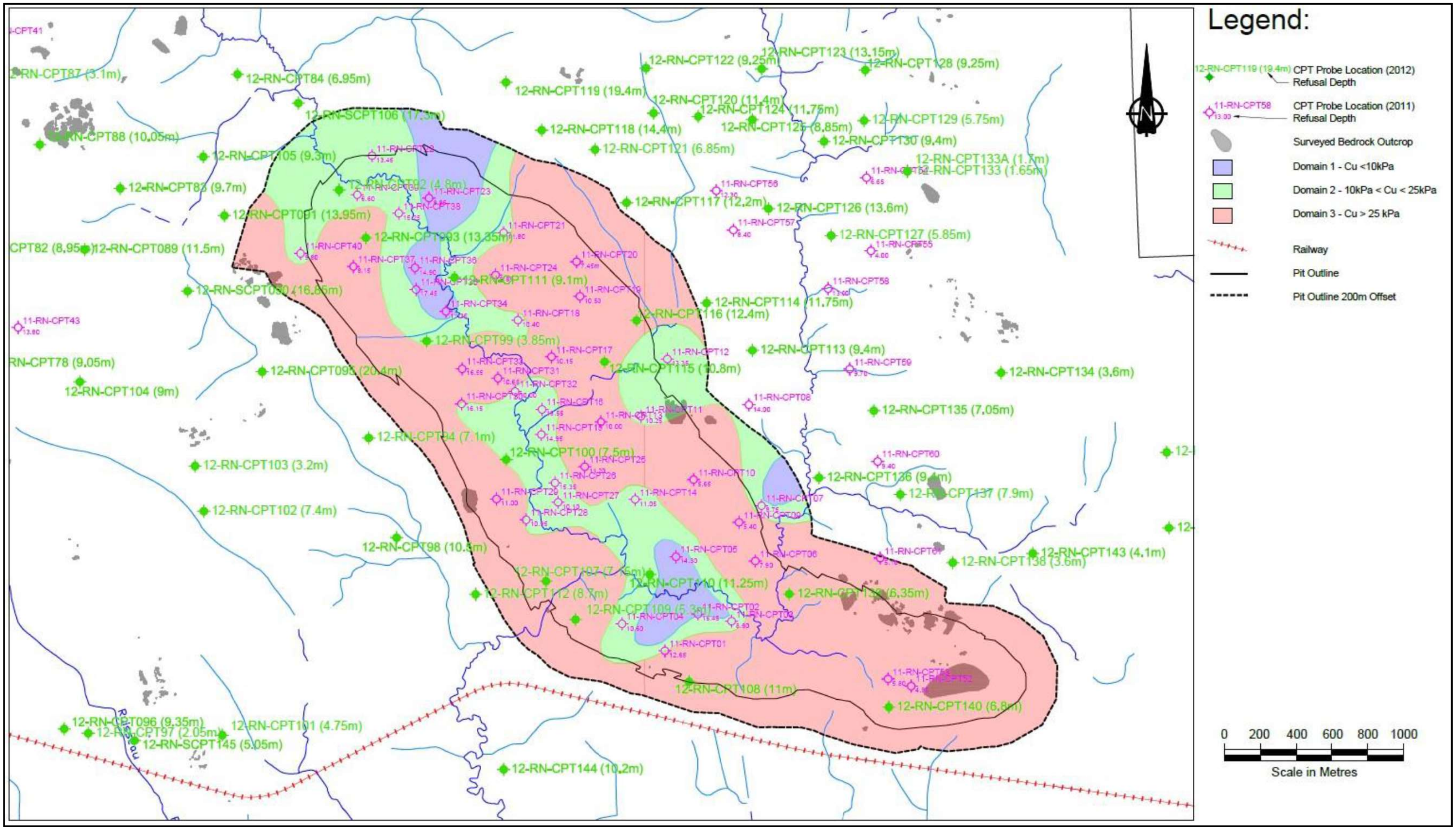
The grey clay, due to its low undrained shear strength, is the weakest unit within the overburden materials. Table 16-3 summarizes the average geotechnical properties of the grey clay based on laboratory testing. The CPT data compares well with the laboratory test results, but also confirms the undrained 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.

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 (cu) 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 will ultimately measure approximately 4.9 km along strike, 1.4 km at the widest point and reach a maximum depth of 520 m below surface. A total of 2,080 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.

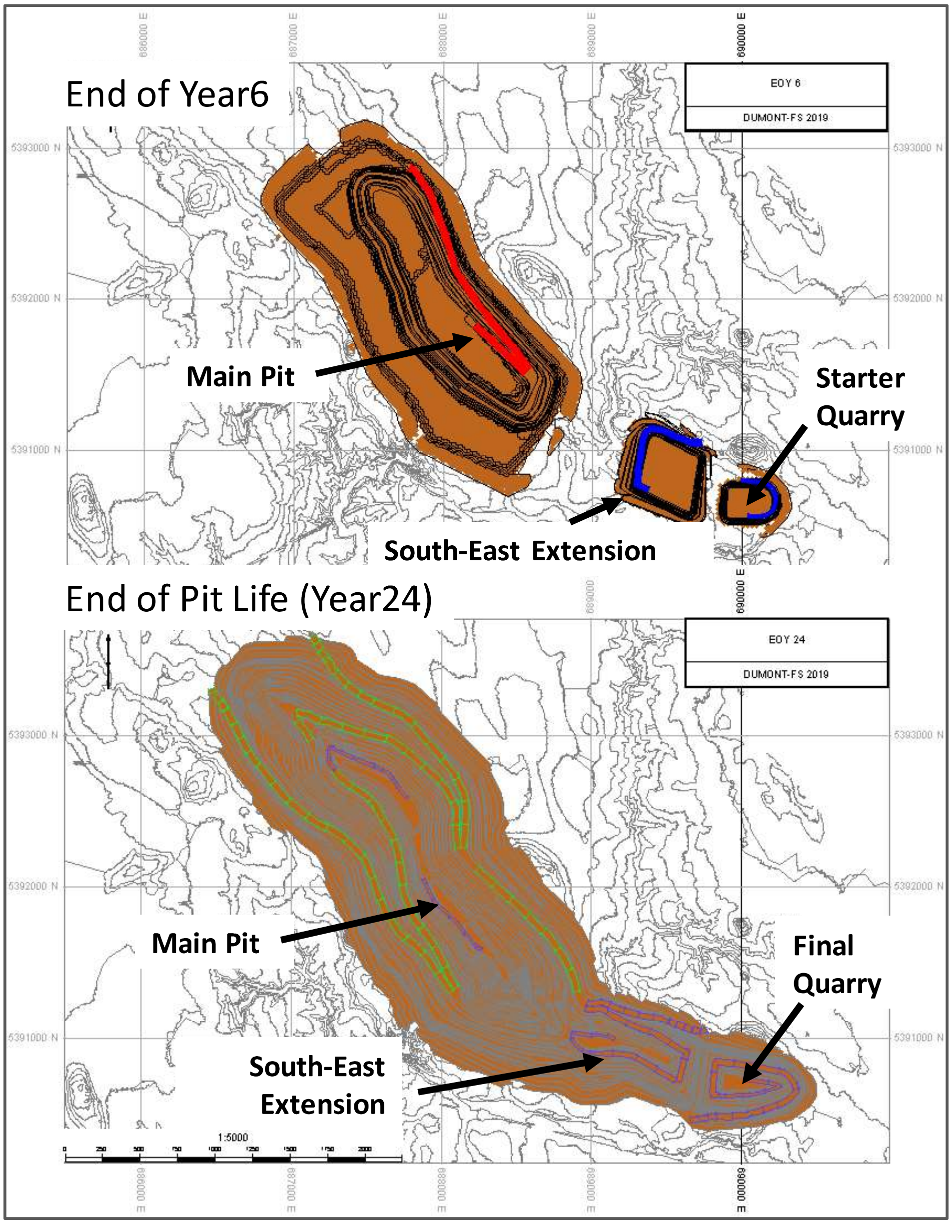
The pit is comprised or three distinct areas (Figure 16.7 on the following page):

- The “Quarry”, at the south east limit of the deposit hosts the only outcrop and is thus where mining initiates. The Quarry is initially excavated to a volume of approximately 5 Mm3 and serves as contingent storage for water during the period the Main Pit is operational. Following completion of operations in the Main Pit, the Quarry is expanded to its full limits of approximately 30 Mm3.
- The “South-East Extension” (SEE), measures approximately 92 Mm3. Development of the SEE commences immediately following that of the Quarry, from which it is separated by a saddle of rock. After the SEE reaches its final limits, it is backfilled to just below surface with waste rock from the Main Pit.
- The “Main Pit” represents approximately 85% of the total excavation.

As described in section 16.2.4 previously, the ore body is covered with overburden of varying depth. Overburden, which comprises 8.3% of the total material that will be excavated, is comprised of differing material 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 material type. Waste rock, which represents 42.3% of the total material excavated, will mainly be stored in a single large dump. The remainder of waste rock will be used for construction of various infrastructure including roads and the tailings storage facility (TSF). Ore, which makes up the remaining 49.4% 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 one of three low-grade stockpiles. Tailings from the treatment of ore will be impounded in the TSF while the Main Pit is operational. In later years, when mill feed is sourced from the Quarry and/or stockpiles, tailings will be impounded within the depleted Main Pit shell.

The mine will have de-watering systems and an electrical supply system for the electrically powered mining equipment, including large excavators, shovels and trolley-assist haul trucks. Unit operations will consist of drilling, blasting, loading and hauling.

Figure 16-7: Dumont Open Pit



Key criteria used in the design of the open pit reflect the size of equipment that will be used and include:

- The final wall bench height will be 10 m to the Overburden – Rock interface (where material will be primarily mined with excavators of various sizes). Below this horizon, benches will be 15 m in height and material will primarily be mined with large rope shovels.
- The ramp width will measure 42 m for the bulk of the mine but reduce to 37 m for sectors where trolley-assist will not be employed. In the initial phase, where smaller equipment will be used, and at depth, where the limited tonnage allows traffic to be restricted to 1-way, ramps will be reduced to 20 m width. All ramps have been designed to a gradient of 10%

- Pushbacks were designed using a target minimum mining width of 150 m in the main body of the pit, to ensure productivity of the large rope shovels. At depth and in the Quarry, where smaller loading equipment and/or 1-way traffic would be employed, this minimum was reduced to 30m on final benches.
- 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 sequence;
- 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 Sections 15.4 - 5 previously. This work was completed in two stages, as summarized in the following paragraphs.

The Penultimate LG optimization entailed calculating the net value of each block in the model by subtracting estimated costs for mining, processing and administration from the NSR. The estimated costs were taken from the 2013 FS and escalated. These were based on the full mill production rate of 105 kt/d.

Inter-ramp slope angles were assigned to the various sectors, also based on recommendations from the 2013 FS. Inter-ramp angles were then adjusted to overall slope angles by taking account of the anticipated ramp geometry along with the 42m ramps that will be used for 290 t class trolley-assist haul trucks.

The LG algorithm then selected a 'cone' of ore and associated waste stripping that maximises NPV. By varying the metal price, higher value nested cones were generated. These allowed the optimal development sequence to be identified. The smallest shell was generated with a revenue factor of 21% of the full long term evaluation price (RF 21, or US$1.58). Nested shells were generated for each subsequent 1% increment in the RF. These nested shells were then aggregated into 13 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 11, which contained 987 Mt ore and 2,040 Mt total material (respectively, 16% and 19% less than the 2013 FS). Beyond Stage 11, the increase in NPV moderated but continued through Stage 13, which approximated the tonnages of both ore and total material of the 2013 FS. An expansion beyond Stage 13 was not contemplated as the total material and associated operating footprint would exceed the limits for which permits have already been awarded.

The Ultimate LG Optimization used a rotated block model along with updated cost estimates and slope angle recommendation which were now based on the Penultimate Case rather than the 2013 FS. The updated slope angles are flatter than those from the 2013 FS in the upper 70m of the deposit, which is now planned to be mined using 10m benches rather than 15m, and in sectors where coalingite is present. The amount of flattening ranges from 3 - 7.

The Ultimate LG Optimization selected a shell that approximated the limits of Stage 11 from the Penultimate Run, as this had been shown to be optimal (note that infrastructure was located to permit a subsequent pushback to the limits of Stage 13 without any material being sterilized). This was achieved with RF 54, as has been reported in Section 15.4

16.3.3 Phase Design & Sequence

Based on the nested shells contained within the RF 54 Ultimate LG Shell, the following eight phases of mining were selected (See Figure 16.8 on the following page):

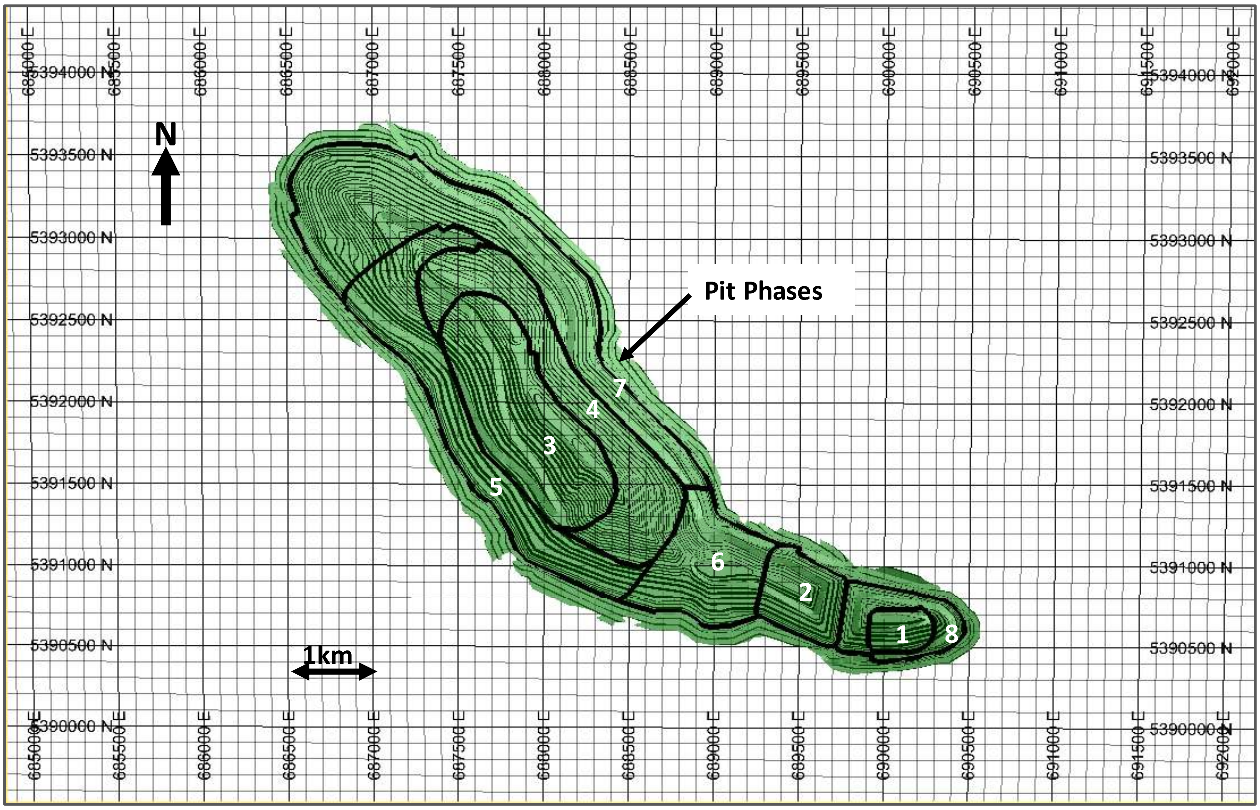
- Phase 1: The Starter Quarry, which targets the only outcrop. The void created by mining of Phase 1 will serve as a reservoir to hold the start-up water requirements for the mill. A more detailed calculation of the start-up water requirements for the operation showed the 10 Mm^3 provided for in the 2013 FS was excessive. Accordingly, the Quarry has been reduced in size to approximately 5 Mm3, or approximately 25% larger than the start-up storage requirements. Longer term, while the Main Pit (Phases 2 – 7) is in operation, the Quarry will also provide contingent surge storage capacity for the freshet and other periods of higher precipitation.
- Phase 2: The volume of rock required for construction exceeds that contained within Phase 1 by approximately 3 Mm3. This will be provided by Phase 2, which is a sub-set of the SEE and targets material with limited overburden cover. Phase 2 is located immediately west of the 'Saddle' separating it from the Quarry.
- Phase 3: This is the highest value portion of the entire pit and is targeted as soon as sufficient construction rock has been liberated from Phases 1 & 2.
- Phase 4: A Main Pit pushback to the hanging wall of Phase 3
- Phase 5: A Main Pit pushback to the footwall of Phase 3
- Phase 6: An extension to the final limits of the SEE
- Phase 7: The final phase of the Main Pit, extending to the west, hanging wall and at depth.
- Phase 8: Following completion of the Main Pit, tailings will be impounded inpit and there will no longer be a requirement for the contingent water storage within the Quarry (as the much larger mined out Main Pit will provide this requirement). Phase 8 represents the extension of the Quarry to the limits of the RF 54 pit shell. Note a rock pillar will remain between this satellite pit and the SEE immediately adjacent.

The phases numbered above represents the optimal sequence for mining, as determined through testing the various alternatives of hanging wall, footwall and strike extension pushbacks from Phase 3.

Key differences between the current phases designs and those selected for the 2013 FS include:

- The minimum width of individual phases has been increased from 100m in 2013 to 150 m currently, to ensure that the large rope shovels can be productively utilized.
- The 2013 phases were 'split' along the east-west axis in order to minimize instantaneous stripping requirements. However, this necessitated additional ramp systems to provide access to all four quadrants of the deposit. In turn, this reduced the density of traffic on individual ramp systems, making the project less amenable to use of trolley-assist. The current phase design continues to utilize a dual redundant ramp system but has reduced the number of total inpit ramps from 4 to 2.

Figure 16-8: Phases of Open Pit Development



Source: RNC.

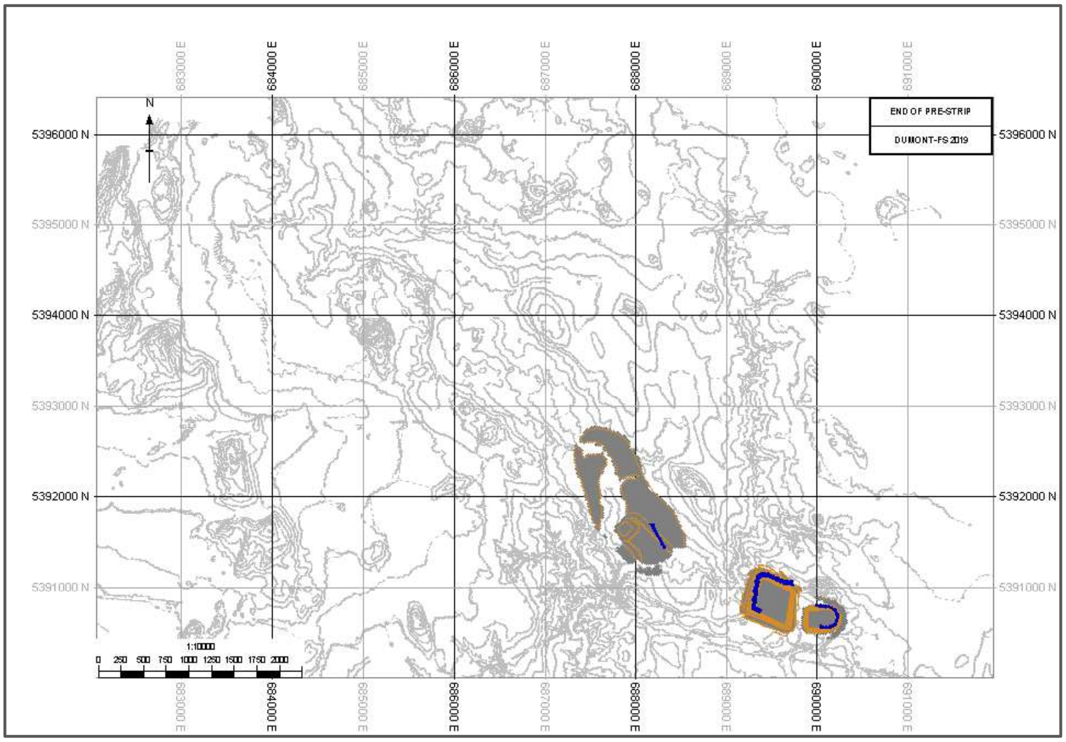
16.3.4 Annual Plans & Mine Schedule

The phase designs were then used as the basis for annual plans that are illustrated in Figures 16.9 through 16.17 on the following pages. Note that to clearly illustrate the SEE design, the inpit waste dump (WRD2) has been omitted from Figures 16.15 – 16.17

Mining schedules were first developed from the engineered phases design in a spreadsheet, that allowed the optimal to be identified. Degrees of freedom tested included:

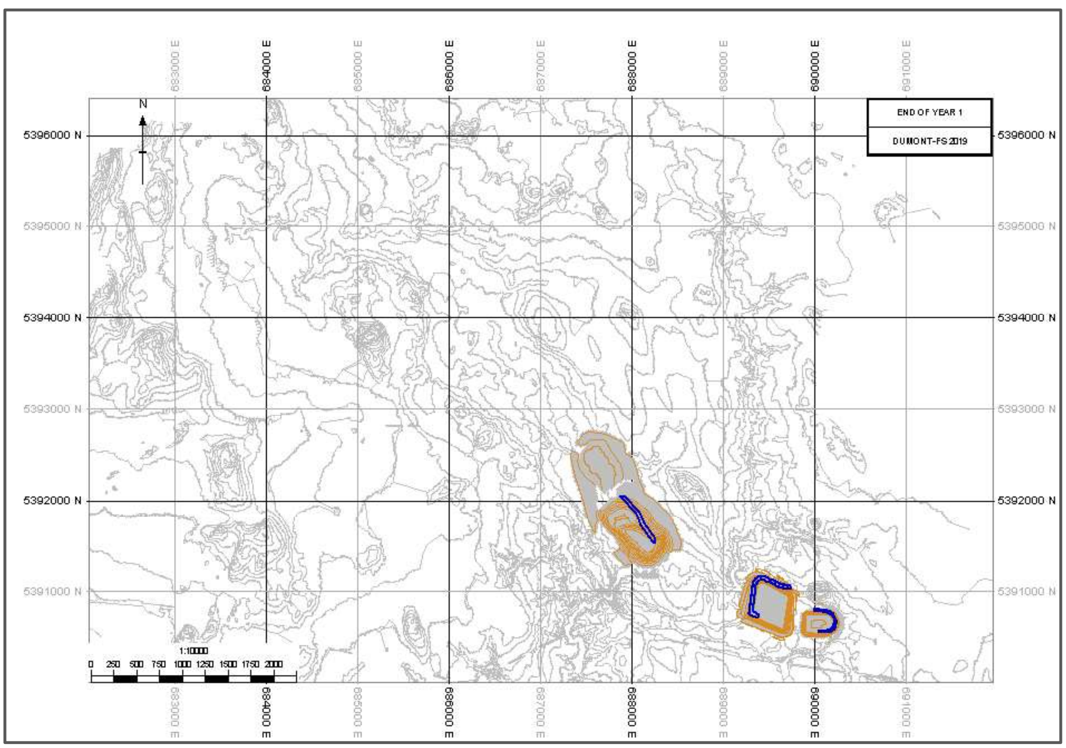
- Early start date for a phase, with the scheduling increment for a phase being quarters of 3-months
- Specific numbers of fleet utilized in each phase. Fleet included
    - A 'team' of 1 x 90 t and 1 x 150 t class backhoes loading 45 t articulated trucks (steady-state capacity of 7.2 Mtpa for 1 team)
    - 300 t class face shovel excavators loading 90 t trucks (capacity of 8 Mtpa per excavator)
    - 600 t class face shovel excavator loading 290 t trucks (capacity of 20 Mtpa per excavator)
    - Rope shovels loading 290 t trucks (capacity of 36 Mtpa per shovel)
- The total number of each class loading unit and early start date for each machine

Figure 16-9: Mine Development – End of Pre-Strip



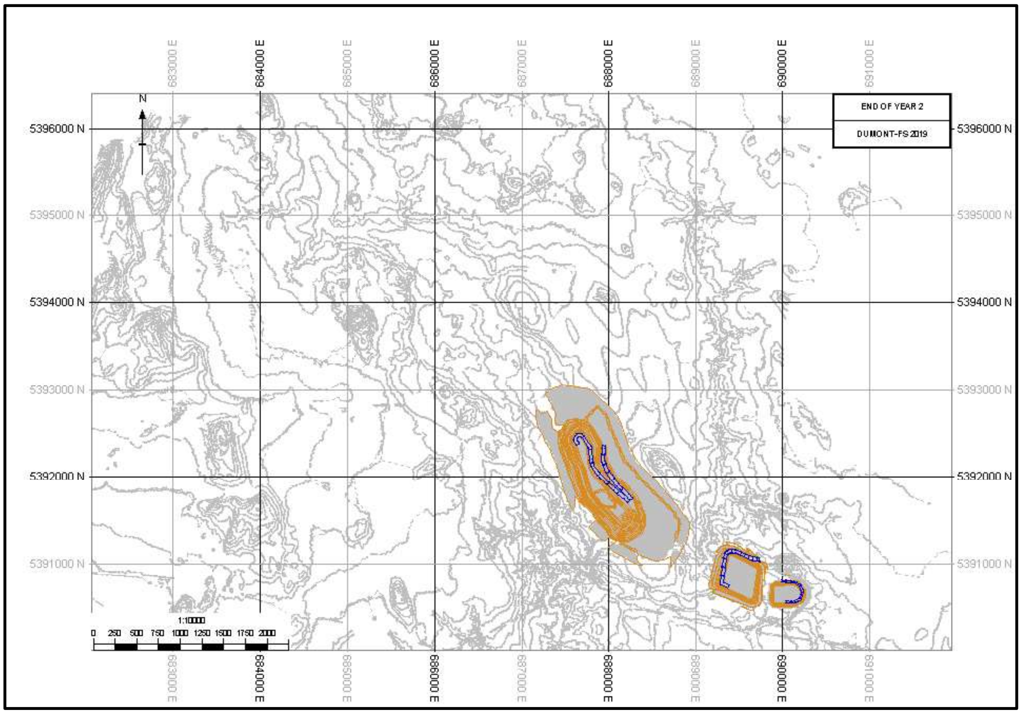
Source: RNC.

Figure 16-10: Mine Development – End of Year 1



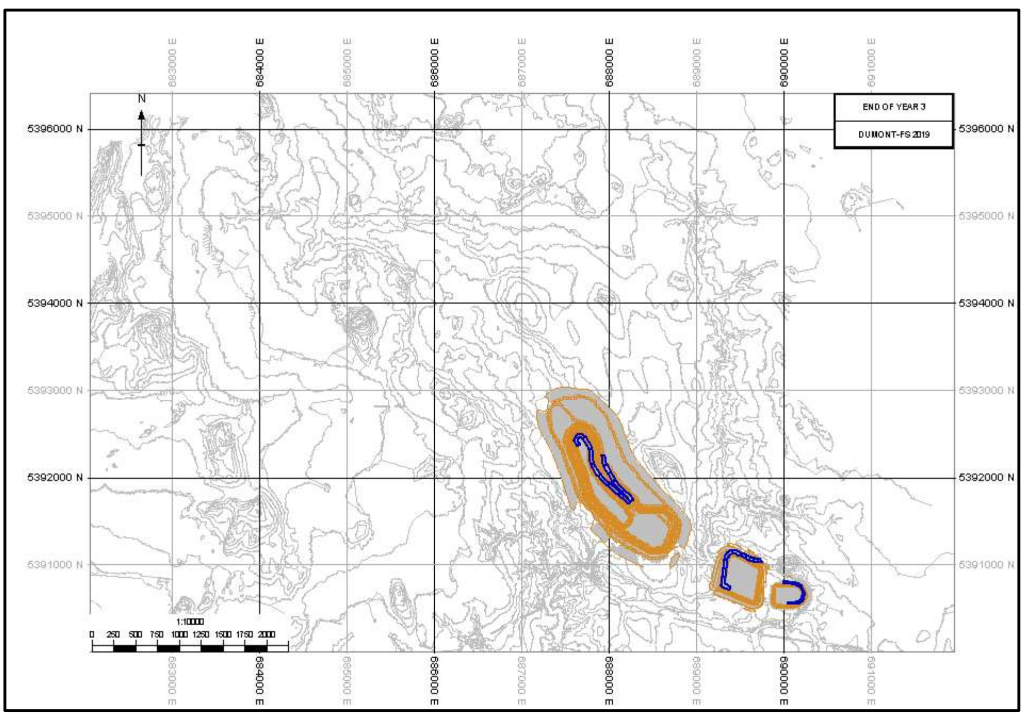
Source: RNC.

Figure 16-11: Mine Development – End of Year 2



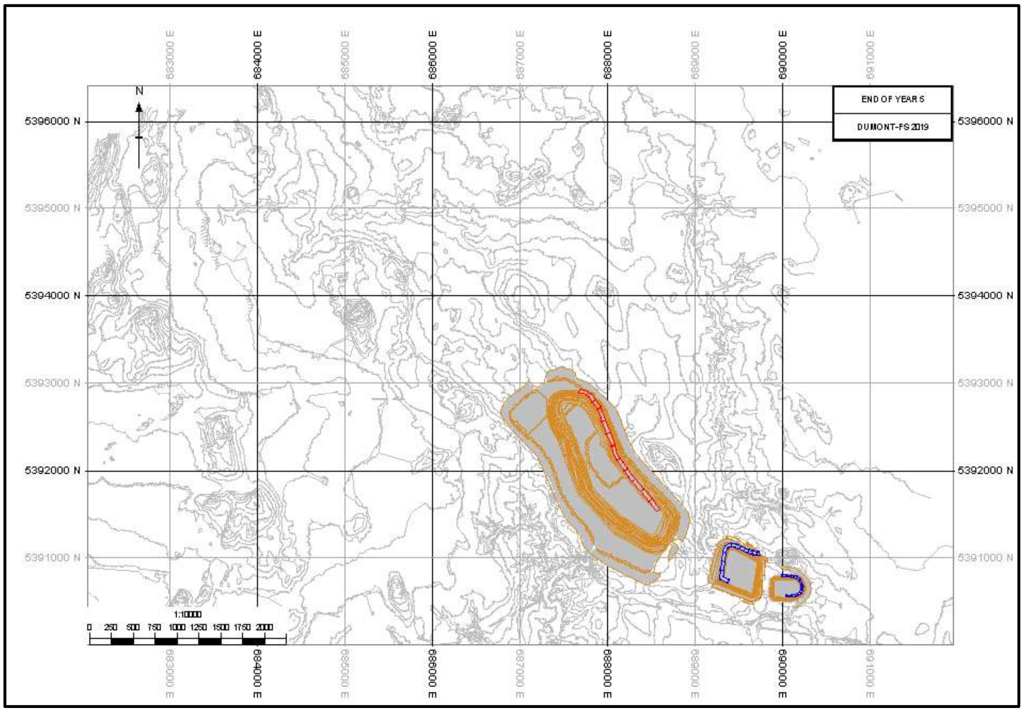
Source: RNC.

Figure 16-12: Mine Development – End of Year 3



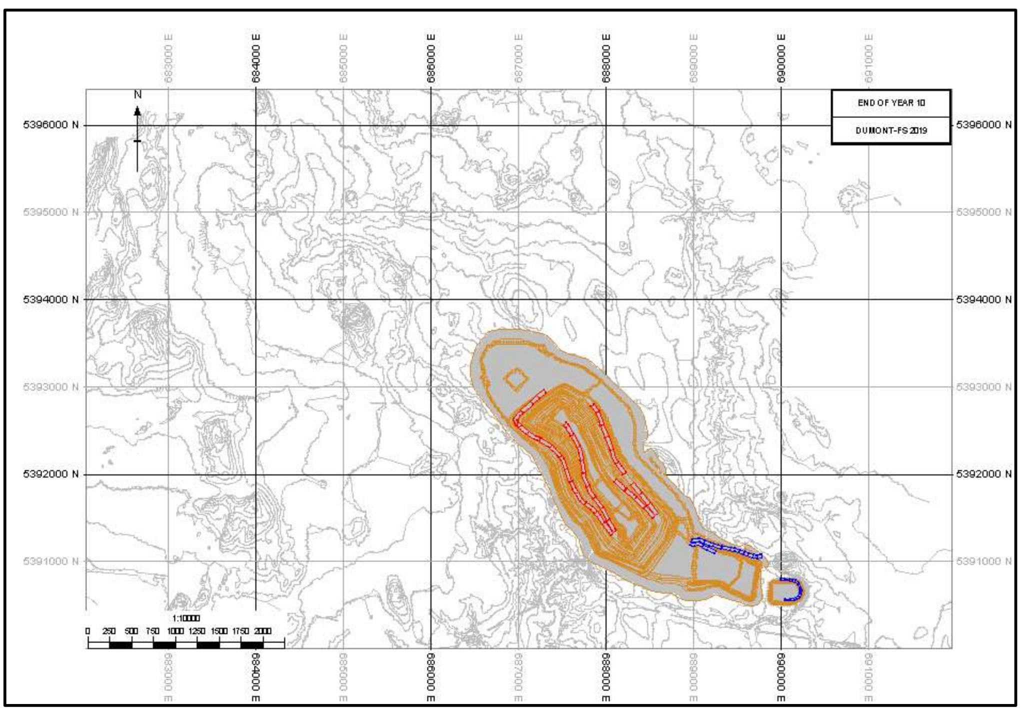
Source: RNC.

Figure 16-13: Mine Development – End of Year 5



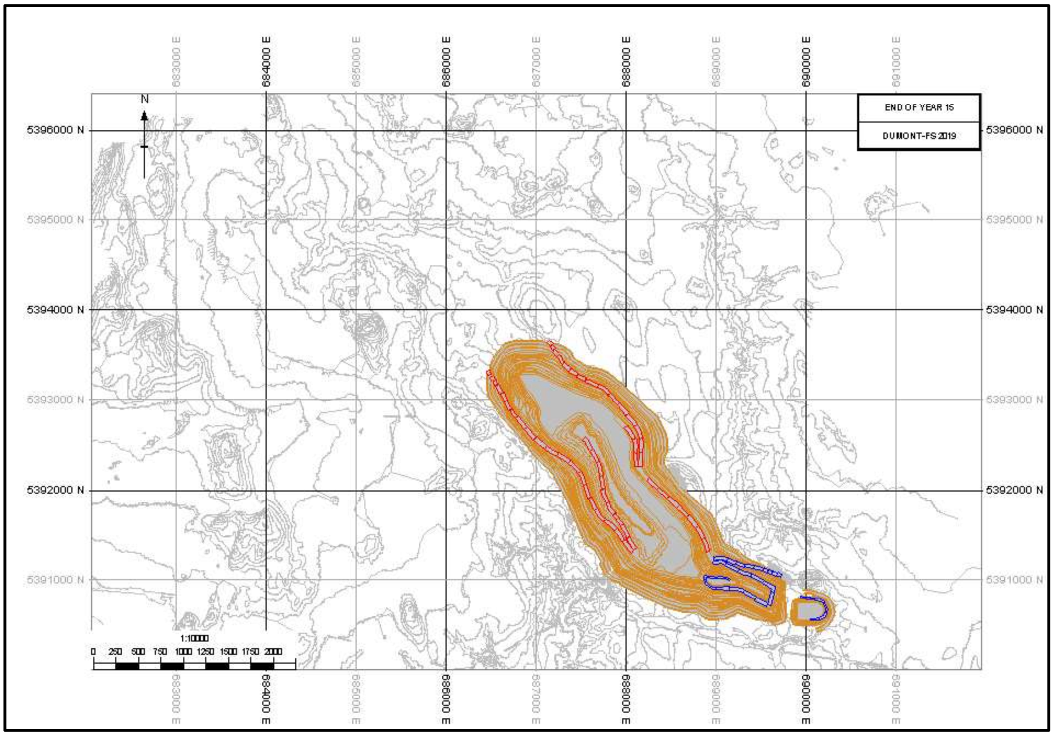
Source: RNC.

Figure 16-14: Mine Development – End of Year 10



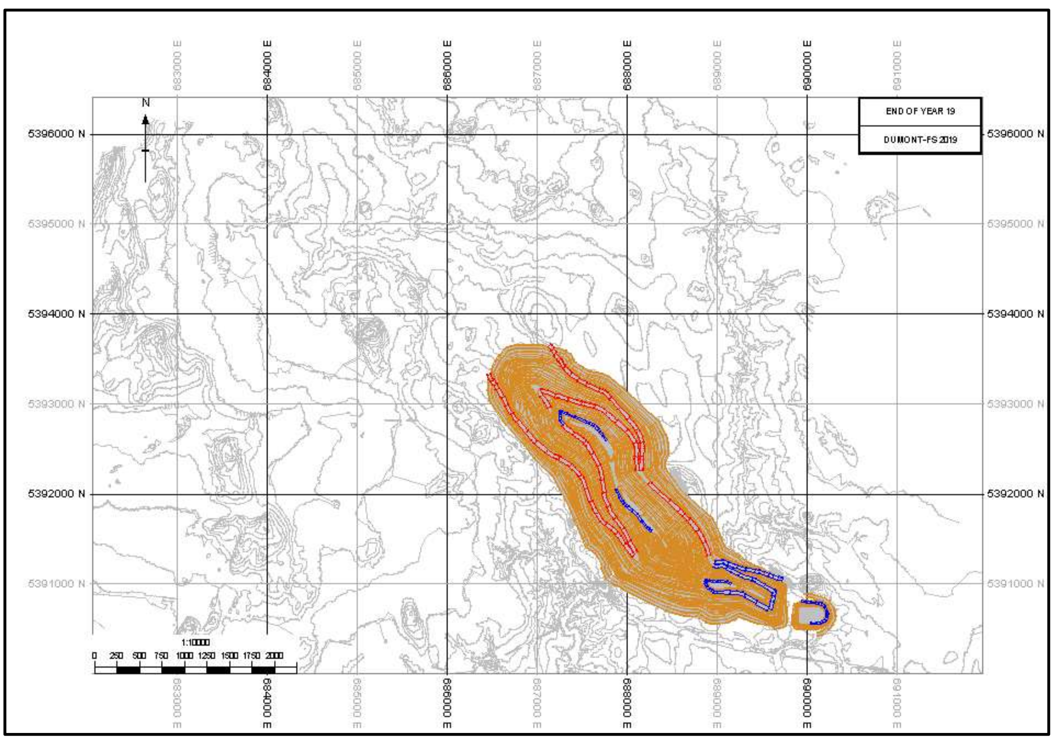
Source: RNC.

Figure 16-15: Mine Development – End of Year 15



Source: RNC.

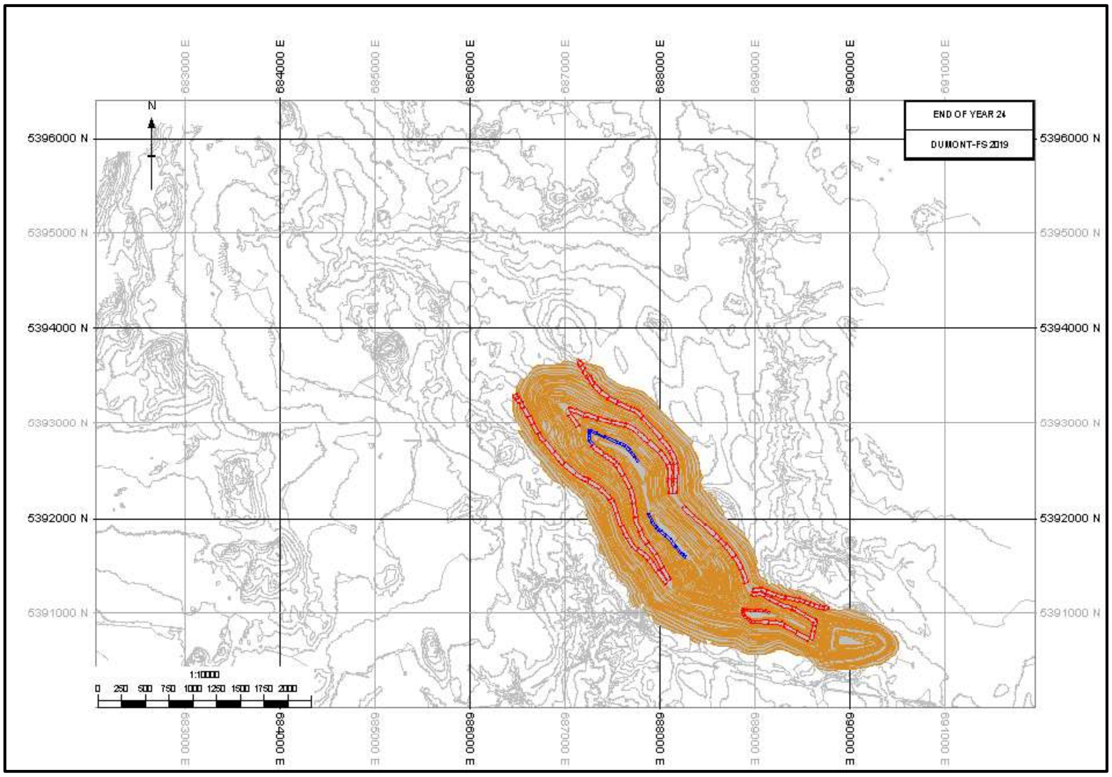
Figure 16-16: Mine Development – End of Year 19 (End of Main Pit Life)



Source: RNC.

Ausenco

Figure 16-17: Mine Development – End of Year 24 (End of Mining)



Source: RNC.

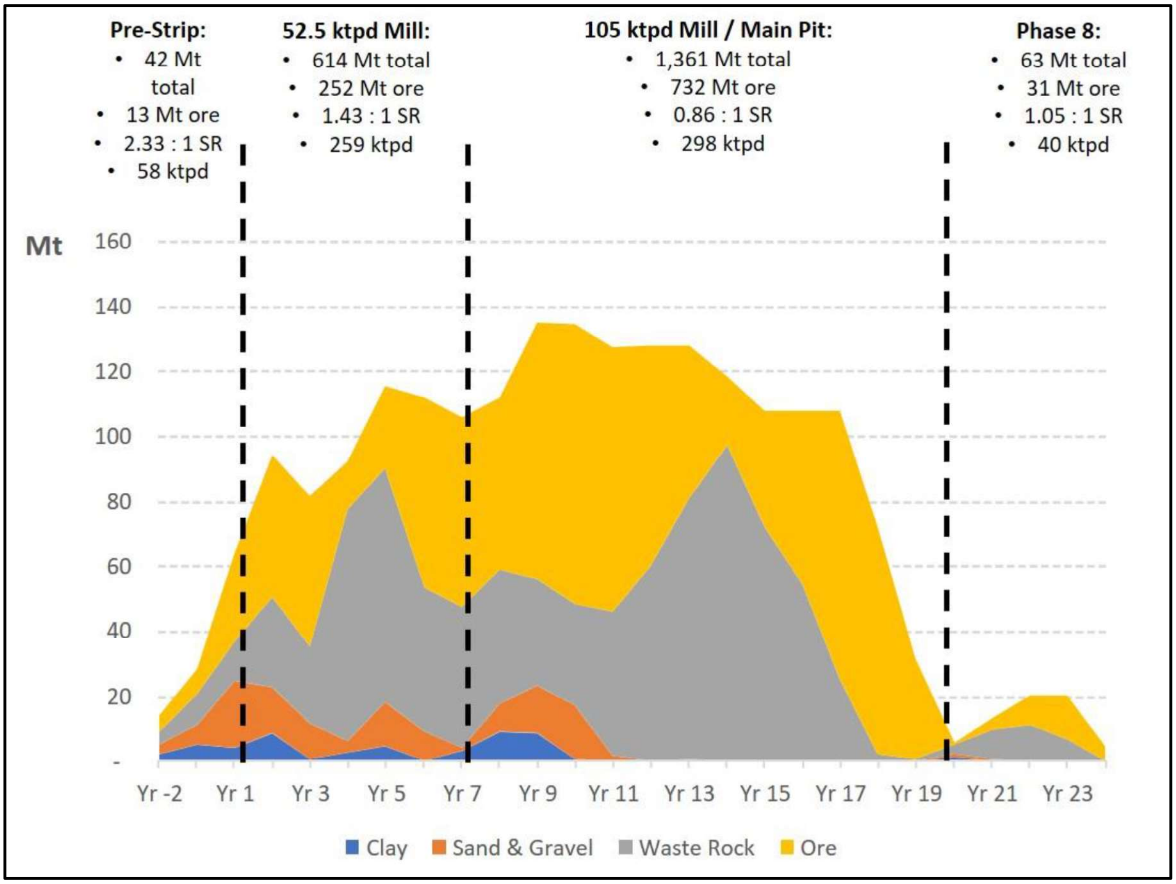
The mine schedule can be summarized as follows:

- Pioneering commences in Phase 1 with 1 team of backhoes in Q-8. The team will complete the required scope early in Q-7, at which time the first 300 t excavator is commissioned. This excavator remains until Phase 1 complete in Q-1.
- The backhoe team moves from Phase 1 to Phase 2 and is active until the end of Q-4. Sufficient area is opened up to commission the second 300 t excavator in Q-6. This is then replaced with the first 600 t excavator during Q-2. The 600 t excavator remains in the phase until the end of Q3.
- The backhoe team move from Phase 2 to Phase 3 in Q-3 and remain active until the end of Q1. At this point, the backhoes are not required in the pit until stripping of Phase 4 commences in Q5 and these units can be redeployed for construction of the TSF. The two 300 t excavators are redeployed into Phase 3 as they complete their allotted scopes in Phases 1 and 2 and remain until the end of Q5. The second 600t excavator is commissioned on the stockpiles in time to feed the mill in Q1, then is relocated to Phase 3 in Q3 and is joined the following quarter by the other unit once it is finished in Phase 2. The first shovel is commissioned in Q1 and remains until Q10.
- The backhoe team are redeployed to the pit to open up Phase 4 during Q5 – 8. They are joined by the two 300 t excavators once they are complete Phase 3 and remain until the end of Q9. The 300 t excavators are then not required until Phase 5 is opened up and therefore represent contingent capacity. The two 600 t excavators are redeployed as they complete Phase 3 before one excavator is redeployed to the surface stockpiles. The second remains until the end of Q26, at which time it is also dispatched to the stockpiles. The first shovel is moved from Phase 3 upon completion in Q10 and joined by the second, which begins commissioning in Q11.

- Backhoe stripping of Phase 5 extends from Q15 – Q18, while the two 300 t excavators are active from Q16 – Q19. The first 600 t excavator that was active on the stockpiles is redeployed to this Phase in Q19 and remains until Q41. The excavator is then sent to the stockpiles, where it remains for the remaining life of mine. The two shovels are moved from Phase 4 in Q22 and Q26, respectively.
- Phase 6 requires only limited backhoe stripping and thus is completed in Q27. The following quarter, the two 300 t excavators begin operating and remain active until Q31. Next quarter, the 600 t excavator that was on the stockpiles is moved in and remains active until the end of Q37. At this point it is replaced by one of the shovels from Phase 5. This shovel remains active until the phase is completed in Q55.
- Phase 7 is the largest phase and requires 9 quarters of backhoes stripping, that commence as soon as stripping of Phase 6 is complete in Q28. They are joined by the two 300t excavators in Q33 and these units are active for 8 quarters. The 600 t excavator in Phase 6 is moved in Q38 and remains until Q55. The third shovel is commissioned in Q39 and joined by the second (from Phase 5) in Q45. The third shovel is moved upon completion of Phase 6 in Q55, at which time the 600 t excavator is put on stand-by.
- The Main Pit and SEE (Phase 2 – 7) are completed in Q76. At this point the shovels move to the stockpiles. After a 3-month delay to drain the Quarry and relocate piping, stripping of Phase 8 commences in Q78. As material will have had opportunity to drain, stripping will be performed using a 300 t face shovel and continue for 4 quarters. In Q82, this unit is replaced by a 600 t excavator, which will be active for 14 quarters.

Figure 16-18 and Table 16-5 and Table 16-6 provide a summary of the annual production.

Figure 16-18: Summary Mine Production Schedule



Source: RNC.

Table 16-5: Life-of-Mine Plan (Mining)

Mining																												
Ore Mining	**units**	**TOTAL**	**Yr -2**	**Yr -1**	**Yr 1**	**Yr 2**	**Yr 3**	**Yr 4**	**Yr 5**	**Yr 6**	**Yr 7**	**Yr 8**	**Yr 9**	**Yr 10**	**Yr 11**	**Yr 12**	**Yr 13**	**Yr 14**	**Yr 15**	**Yr 16**	**Yr 17**	**Yr 18**	**Yr 19**	**Yr 20**	**Yr 21**	**Yr 22**	**Yr 23**	**Yr 24+**
Diluted Ore	Mt	1,028	5	8	27	44	46	15	25	59	59	53	79	86	82	68	47	21	36	54	83	70	31	0	4	9	14	4
Grade Ni	% Ni	0.268	0.254	0.244	0.303	0.289	0.297	0.268	0.277	0.285	0.260	0.239	0.247	0.265	0.263	0.251	0.255	0.257	0.255	0.257	0.277	0.295	0.317	0.239	0.238	0.248	0.250	0.250
Contained Ni	Mlbs	6,083	28	42	182	279	304	86	152	369	336	278	432	503	475	377	265	117	203	305	507	455	218	2	19	49	75	23
Recovery Ni	% of con'd	43.2	51.3	50.2	47.2	48.1	44.7	39.2	42.5	47.0	42.6	37.4	39.2	42.7	42.0	36.5	37.5	39.7	38.8	40.5	44.5	49.6	53.8	47.3	47.1	47.7	45.7	43.5
Recoverable Ni [1]	Mlbs	2,627	14	21	86	134	136	34	65	174	143	104	169	215	199	138	99	46	79	124	226	226	117	1	9	24	34	10
Total Expit Mining	**units**	**TOTAL**	**Yr -2**	**Yr -1**	**Yr 1**	**Yr 2**	**Yr 3**	**Yr 4**	**Yr 5**	**Yr 6**	**Yr 7**	**Yr 8**	**Yr 9**	**Yr 10**	**Yr 11**	**Yr 12**	**Yr 13**	**Yr 14**	**Yr 15**	**Yr 16**	**Yr 17**	**Yr 18**	**Yr 19**	**Yr 20**	**Yr 21**	**Yr 22**	**Yr 23**	**Yr 24+**
Ore	Mt	1,028	5	8	27	44	46	15	25	59	59	53	79	86	82	68	47	21	36	54	83	70	31	0	4	9	14	4
Waste Rock	Mt	879	4	9	12	28	23	72	72	44	43	42	33	31	44	60	80	97	72	54	25	2	1	3	9	11	6	0
Clay	Mt	49	2	5	4	8	1	3	4	0	3	9	8	1	-	-	-	-	-	-	-	-	-	1	0	-	-	-
Sand & Gravel	Mt	124	3	6	20	14	11	3	14	9	1	8	14	16	1	-	0	-	-	-	-	-	-	1	0	-	-	-
Total Expit	**Mt**	**2,080**	**14**	**28**	**64**	**94**	**81**	**92**	**115**	**112**	**106**	**112**	**135**	**134**	**128**	**128**	**128**	**118**	**108**	**108**	**108**	**72**	**32**	**5**	**13**	**20**	**20**	**4**
Rehandled Low Grade Ore	Mt	273	-	-	4	-	2	13	6	-	12	12	-	-	-	0	10	24	16	6	-	-	8	38	35	29	25	34
Hauling Distances	**units**		**Yr -2**	**Yr -1**	**Yr 1**	**Yr 2**	**Yr 3**	**Yr 4**	**Yr 5**	**Yr 6**	**Yr 7**	**Yr 8**	**Yr 9**	**Yr 10**	**Yr 11**	**Yr 12**	**Yr 13**	**Yr 14**	**Yr 15**	**Yr 16**	**Yr 17**	**Yr 18**	**Yr 19**	**Yr 20**	**Yr 21**	**Yr 22**	**Yr 23**	**Yr 24+**
45t Articulated Trucks	Mt	39	6	6	0	6	-	3	1	-	2	7	7	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	1-way metres	4,741	3,502	4,345	3,014	3,991	-	8,489	5,541	-	6,858	4,862	4,359	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
90t Trucks	Mt	105	8	16	15	13	2	2	4	-	2	10	16	12	-	-	-	-	-	-	-	-	-	5	1	-	-	-
	1-way metres	4,582	4,228	4,892	3,326	4,214	4,526	8,470	5,089	-	8,800	6,935	4,108	4,875	-	-	-	-	-	-	-	-	-	1,776	3,060	-	-	-
290t Trucks	Mt	1,883	-	7	49	75	80	84	61	112	102	95	112	122	128	128	128	118	108	108	108	72	32	-	12	20	20	4
	1-way metres	5,848	-	4,240	3,670	4,123	4,553	5,469	4,698	4,725	4,337	4,776	5,014	5,807	6,402	7,130	7,405	4,568	6,937	7,763	8,051	7,711	6,764	-	2,589	3,855	5,611	7,506

Note:
1. Mined recoverable Ni higher than actual recovered Ni due to assumed losses during process ramp-up

Source: RNC

Table 16-6: Life-of-Mine Plan (Processing)

Processing

Yrs 1 - 15	**units**	**TOTAL**	**Yr -2**	**Yr -1**	**Yr 1**	**Yr 2**	**Yr 3**	**Yr 4**	**Yr 5**	**Yr 6**	**Yr 7**	**Yr 8**	**Yr 9**	**Yr 10**	**Yr 11**	**Yr 12**	**Yr 13**	**Yr 14**	**Yr 15**
Ore Processed	Mt	1,028	-	-	17	19	19	19	19	19	26	38	38	38	38	38	38	38	38
Grade Ni	% Ni	0.268	-	-	0.33	0.34	0.36	0.29	0.30	0.37	0.30	0.25	0.27	0.30	0.29	0.26	0.26	0.25	0.26
Contained Ni	Mlbs	6,083	-	-	123	145	154	122	128	155	176	213	229	255	245	221	220	213	220
Recovery Ni	% of con'd	43.2	-	-	52.3	55.0	52.7	47.8	46.9	56.9	51.0	42.2	44.6	50.2	49.7	41.8	42.4	42.0	42.4
Recovered Ni	Mlbs	2,625	-	-	64	80	81	58	60	88	90	90	102	128	122	92	93	89	93
Concentrate Produced	ktonnes	4,061	-	-	95	111	134	82	102	134	135	148	160	195	181	136	144	136	139
Payable Ni	Mlbs	2,402	-	-	59	73	74	53	55	81	82	82	93	117	112	84	85	82	85

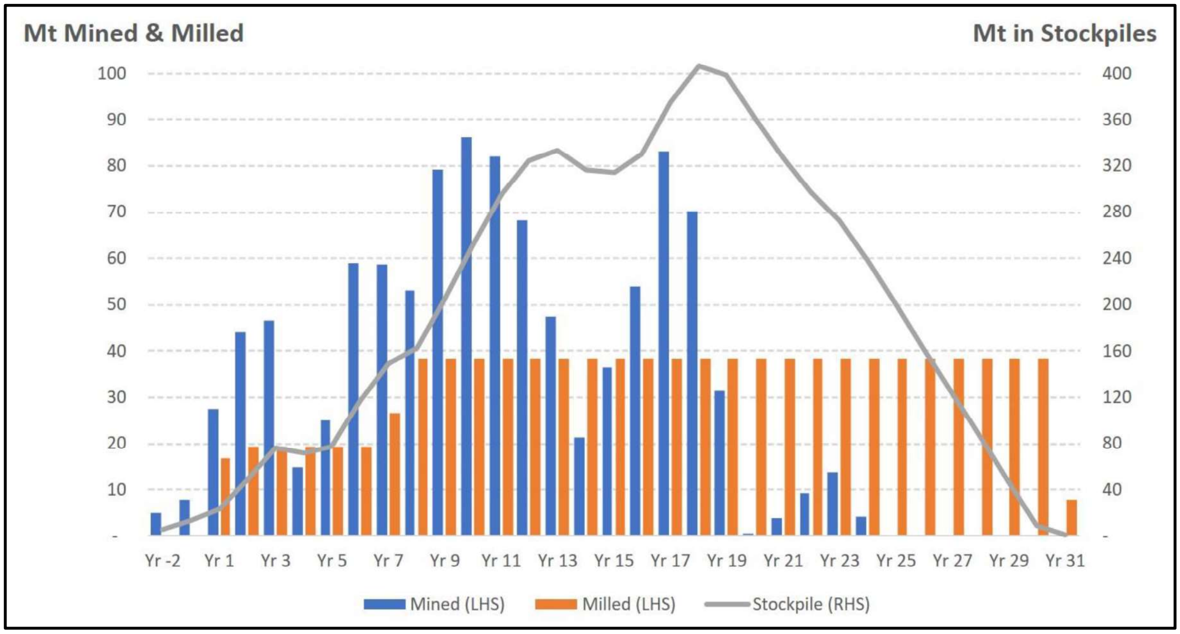
Yrs 16 - 31		**Yr 16**	**Yr 17**	**Yr 18**	**Yr 19**	**Yr 20**	**Yr 21**	**Yr 22**	**Yr 23**	**Yr 24**	**Yr 25**	**Yr 26**	**Yr 27**	**Yr 28**	**Yr 29**	**Yr 30**	**Yr 31**
Ore Processed	Mt	38	38	38	38	38	38	38	38	38	38	38	38	38	38	38	8
Grade Ni	% Ni	0.27	0.32	0.33	0.30	0.25	0.25	0.24	0.24	0.24	0.23	0.23	0.23	0.23	0.22	0.22	0.22
Contained Ni	Mlbs	232	272	279	256	211	210	207	202	199	197	197	197	196	185	185	37
Recovery Ni	% of con'd	44.7	52.1	55.6	52.2	41.0	41.5	40.4	37.6	34.1	32.9	32.9	32.9	31.7	24.9	24.9	24.9
Recovered Ni	Mlbs	104	142	155	134	86	87	83	76	68	65	65	65	62	46	46	9
Concentrate Produced	ktonnes	160	229	239	205	130	130	123	112	106	105	105	105	101	81	81	16
Payable Ni	Mlbs	95	130	142	123	79	80	76	70	62	59	59	59	57	42	42	8

16.3.5 Low-grade Ore Stockpiles

A key component of the mine plan is accelerated mining of ore from the pit, with higher value ore fed directly to the mill and lower value material temporarily stockpiled. A total of 511 Mt will be loaded to the low-grade stockpiles, representing 49.7% of total ore. The philosophy of ensuring that the mill is fed with the highest value ore available results in 112 Mt of low-grade material being reclaimed while the Main Pit is active, using one or both 600t hydraulic excavators. Phase 8 is unable to satisfy the milling rate and a further 142 Mt will be reclaimed while this phase is active. This material will be loaded using one or more rope shovels that have been removed from the Main Pit. Note the grade distribution of Phase 8 is such that 99.1% of total ore mined is planned for ROM delivery. The remaining 257 Mt of stockpiled material will be reclaimed after pit closure, also using rope shovel(s). Reclaiming the low-grade material extends the life of project into Year 31 of mill production (see Figure 16-19).

The strategy of accelerated mining has the additional advantage of creating a void (i.e., the mined-out open pit), which would accommodate approximately 428 Mt or 42% of the total tailings produced, thus reducing the surface footprint of operations.

Figure 16-19: Mill Production & Low-grade Stockpile

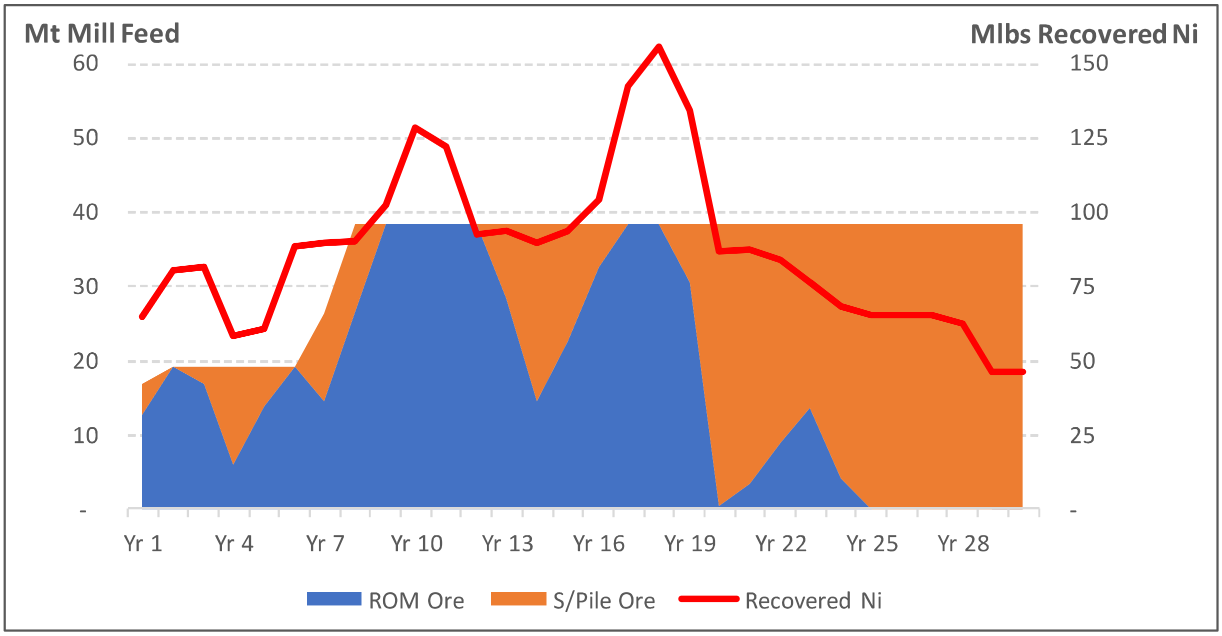


Source: RNC.

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 73 Mlbs recovered Ni during the first 6.5 years of production when the concentrator throughput is 52.5 kt/d. Maximum output during this time is 88 Mlbs in Yr6.

After throughput is increased to 105 kt/d, output increases to an average of 111 Mlbs recoverable Ni (maximum is 155 Mlbs) for Yr8 - 19 while the Main Pit is active. This drops to 83 Mlbs during the four years that Stage 8 is active. After the pit is depleted and the only source of ore is low grade stockpiles, output drops to an average of 59 Mlbs, as shown in Figure 16-20.

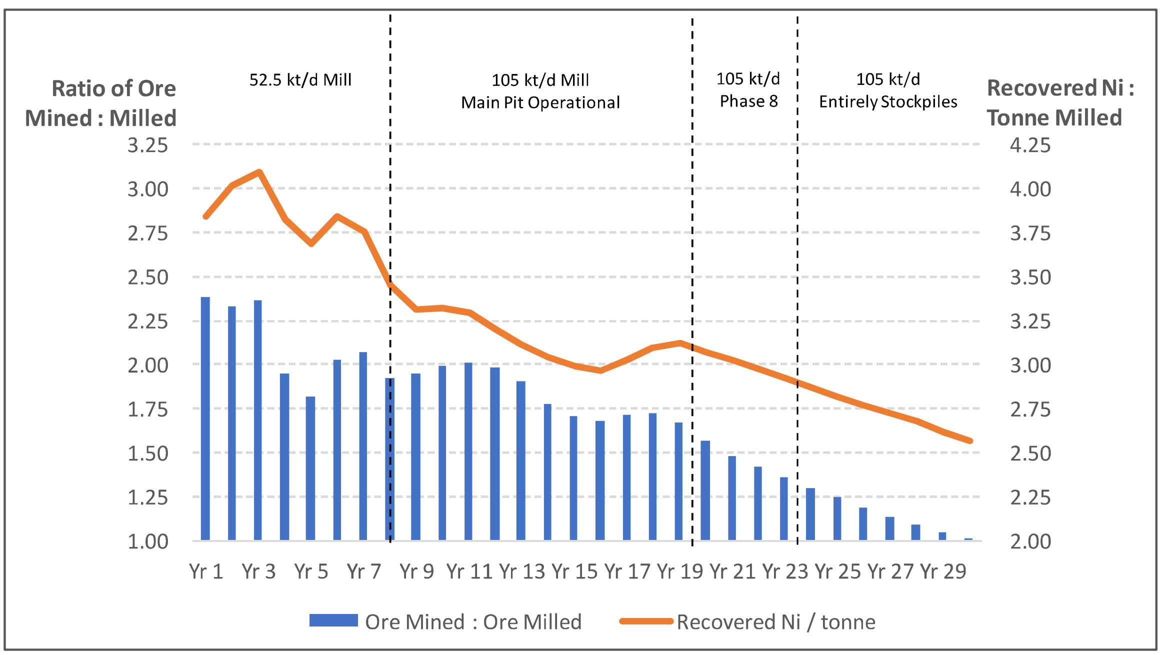
Figure 16-20: Mill Feed by Source & Ni Output



Source: RNC.

Figure 16-21 illustrates the cumulative recovered grade of ore treated through the mill as a function of the accelerated release of ore from the pit.

Figure 16-21: Cumulative Ni per Ore Milled vs. Expit Ore Release



Source: RNC

As a result of the low strip-ratio material mined prior to mill expansion, the mine plan is able to release a peak of 2.4 tonnes of ore for every tonne milled and the recovered grade of ore peaks at 4.0 lbs per tonne ore. The average value for the 7 years of operation at the minimal milling rate of 52.5 ktpd are 2.1 tonnes mined per tonne release and 3.8 lbs recovered Ni per tonne milled. Following expansion and the move into higher stripping ratio areas of the pit, the ratio of ore mined to ore milled drops, averaging 1.5 over the remaining 12 years that the Main Pit is operational. When production moves to Phase 8, the Quarry is only able to provide 17% of mill requirements and, with the remainder sourced from lower grade stockpiles, recovered Ni drops to 2.2 lbs per tonne. For the remaining 8 years that the sole source of feed is stockpiles, recovered Ni declines steadily to average 1.6 lbs over the period.

There is significant variation in the value of material stockpiled and three distinct stockpiles will be employed in order to ensure the highest value material can be presented to the mill soonest. These include:

- LGO1, which is comprised of the highest value material and located within the final pit shell, closest to the primary crusher. The cut-off for material directed to this stockpile will be an NSR value of $23/t. A total of 31 Mt ore will pass through this stockpile, with the maximum impounded at any one time being 15 Mt. This stockpile will be depleted prior to the end of Yr 7, when the pit expands into the area it will be located. The stockpile is divided into two lobes by a stream (LG01w and LGO1e, as shown in Figure 16.24)
- LGO2, which is a larger dump, will also be located close to the crusher but outside the final pit shell. Prior to depletion of LGO1, this stockpile will be fed with material valued $16 - $23/t. Following depletion of LGO1, material grading above $23/t will also be stored in this stockpile. A total of 173 Mt will pass through the stockpile, with the maximum impounded at any one time being 108 Mt. Approximately 47% of ore tipped on this dump will be reclaimed while the Main Pit is still active, with the remainder being reclaimed before the completion of Phase 8 operations.
- LGO3 is the largest low-grade stockpile and located furthest from the crusher. It will receive material between the mine cut-off of $7/t and the lower cut-off to LGO2 of $16. Over the life of mine, a total of 308 Mt will be impounded within this stockpile. None of this material will be reclaimed while the Main Pit is still operational. Reclamation from this stockpile commences when LGO2 is depleted, in Q87. This dump will be sub-divided into two areas based on value of ore. Higher value ore (217 Mt of average value $14/t) will be reclaimed first, followed by the lower value (90 Mt averaging $10/t)

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 LGO3 in two sub-areas and the exact sequence for reclamation, will also be governed by operating parameters that will be established once production commences.

16.3.6 Waste Dumps

The pit excavation includes 49 Mt clay, 124 Mt sand and gravel (S&G) and 879 Mt waste rock.

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 utilized in construction of the TSF (including lining exposed bedrock surfaces) and for reclamation of dumps at the end of pit life. This will thus not be impounded in waste dumps. The remaining grey clay 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 lifts of 5 m each. The bulk of this material will be contained within the mixed material dump (OVB1) that is located centrally on the hanging wall side of the pit. If necessary, clay could also be contained within the smaller overburden (OVB2) at the southeast extremity of the property.

Approximately 11% of the S&G (which also includes organics and till) will be used in TSF construction or reclamation of dumps. A further 64% will be impounded in OVB2 at the extreme south-east end of the operation and form a barrier to mitigate the impact of noise from the operation on communities to the east of the property. The remaining S&G be impounded in OVB1, where it will be used in construction of cells to impound clay.

Approximately 17% 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 727 Mt, only 7% (52 Mt) will be impounded along with sand and gravel and clay in OVB1. The combined volume of clay, sand and gravel, and rock for this impoundment will be 101 Mm3 and it will extend approximately 3.4 km along strike and to an approximate height of 40 m (as with OVB2, it will be constructed in 4 lifts of 5 m, followed by 2 lifts of 10 m). To minimize haulage distances, OVB1 will be accessed by 4 separate ramps. The northern and southernmost will be aligned with the hanging wall pit exits, with the remaining two spaced evenly between.

Approximately 113 Mt of waste rock will be impounded in WRD2, located within the SEE portion of the pit, after mining in Phase 6 is completed. This dump will be constructed in two phases:

- The initial phase takes place while the Main Pit is still active. A stand-off distance will be left a distance of 50 metres from the crest overlooking the main pit and the dump will be constructed in lifts to just below the surface level (to allow drainage through the pit following closure) and will achieve an overall face slope of 1.6 : 1.
- The second phase takes place during mining of Phase 8, when activity in the Main Pit will be completed and all personnel and equipment will have been removed (at this point, impoundment of tailings will begin). During this phase, the existing height of the dump will be maintained, and the dump will be extended toward the Main Pit.

Once the impoundment of tailings within the pit commences, water will begin to collect. Figure 16.22 provides a cross sectional view of WRD2 when completion of the initial stage of deposition in completed in Year 19. Figure 23 then illustrates the dump (along the same section line as Figure 16.22) following completion of all tipping is completed in Year 24, as well as illustrating the progressive raising of the surface level for both tailings and water in later years of the project. It can be seen that by Year 40, WRD2 will be submerged.

Figure 16-22: Cross Section view of Inpit Waste Rock Dump (WRD2) at Year 19

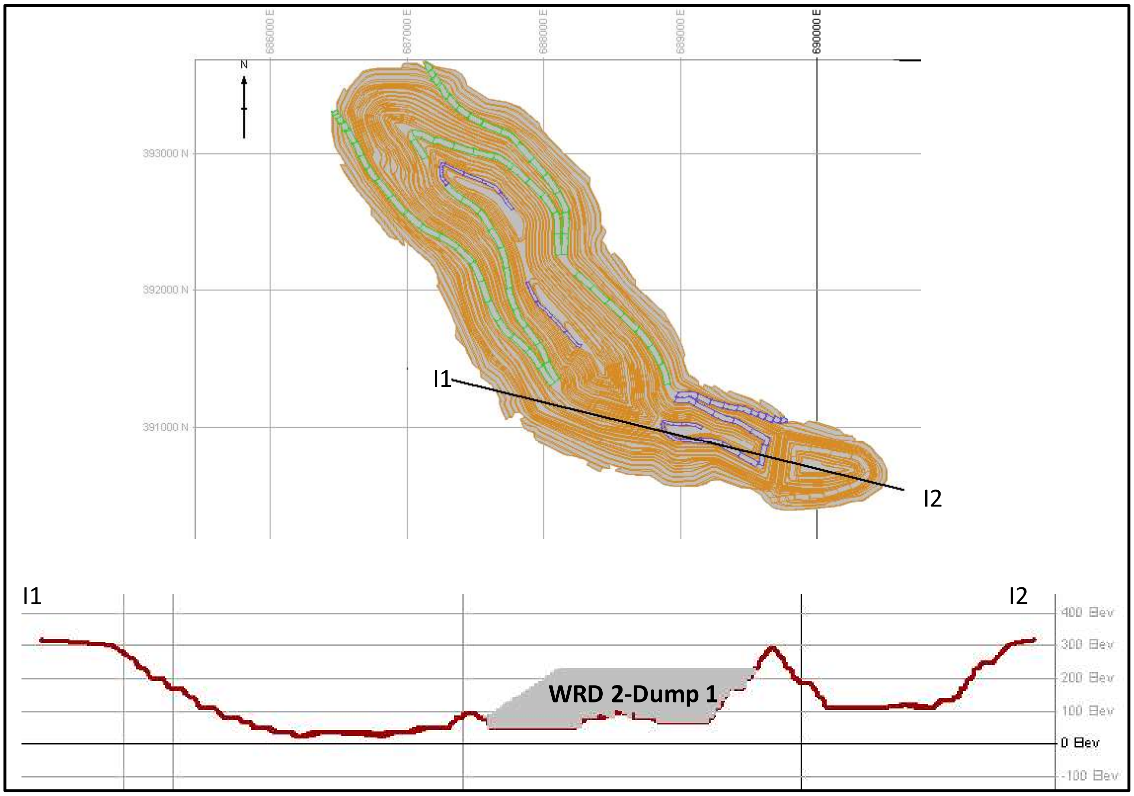
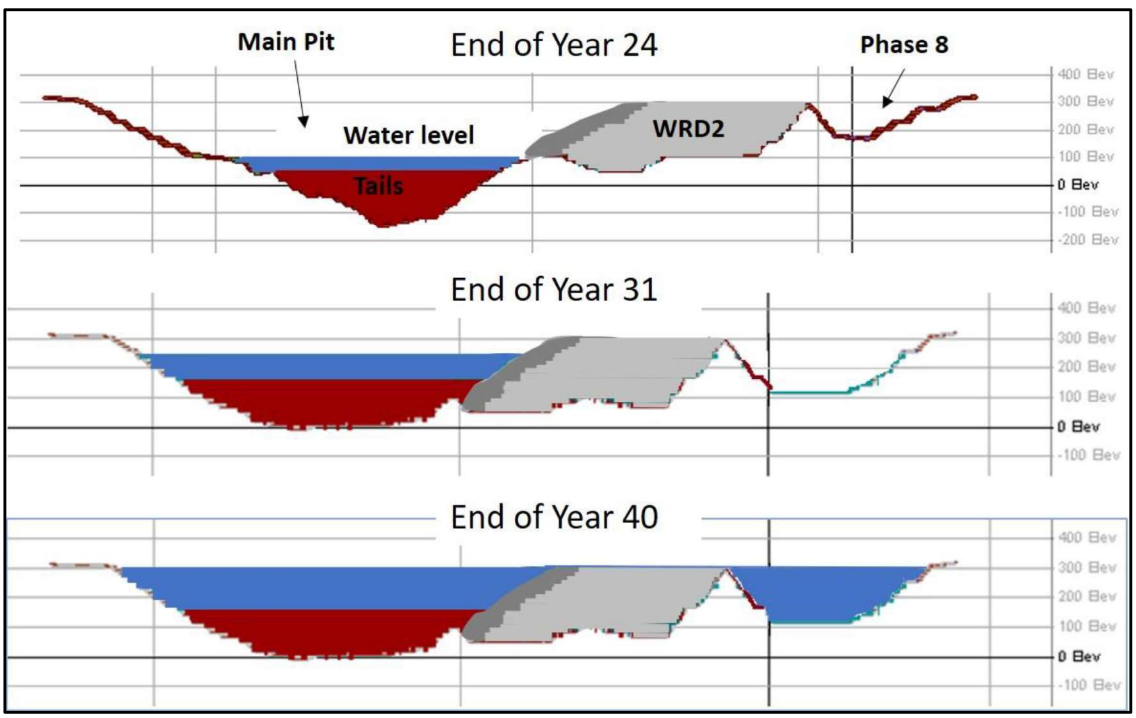


Figure 16-23: Cross Section view of Inpit Waste Rock Dump (WRD2) at Years 24, 31 and 40

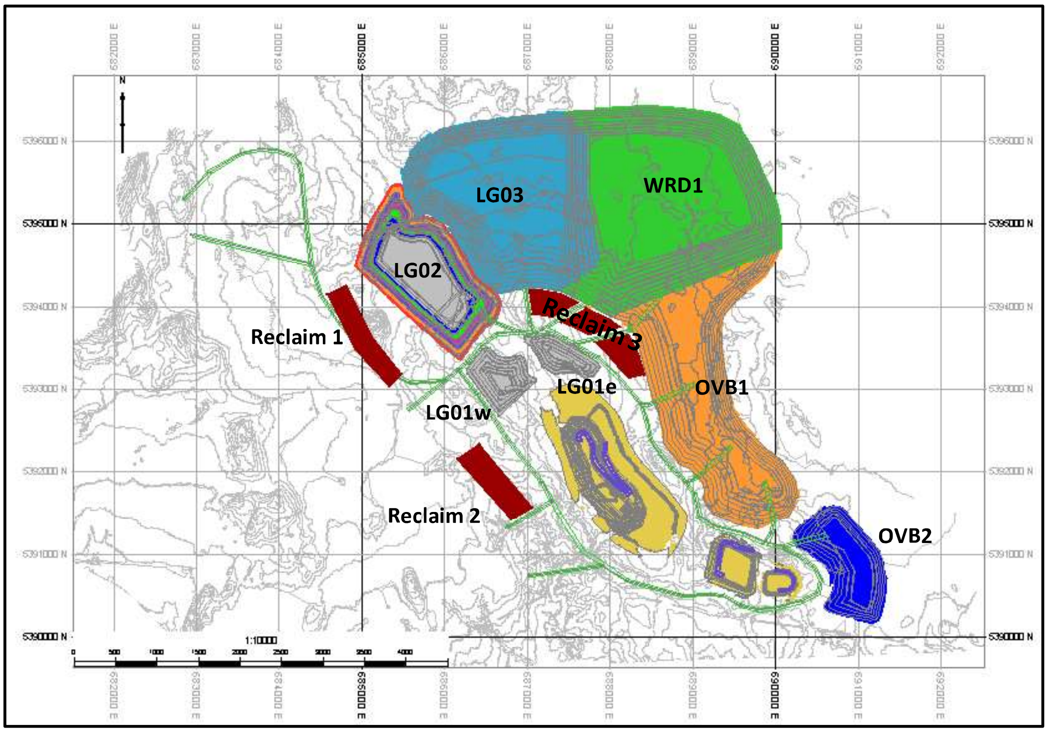


The majority of waste rock (561 Mt) will be stored in WRD1, which is located between OVB1 and LGO3 (see Figure 16.24). This dump will occupy 267 Mm3 compared to the permitted footprint of 353 Mm^3, so there is ample space to impound any waste associated with subsequent pushbacks beyond the scope described in this report. WRD1 will be constructed in 4 lifts of 5m followed by 6 lifts of 10m, reaching an ultimate height of 80 m. To ensure stability, the pit-facing slopes will be a relatively flat 6H:1V, compared to 3H:1V used on slopes not facing the pit

The boundary between WRD1 (which will be a permanent impoundment) and LGO3 (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.

Figure 16-24 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). Note that WRD2 is not included as tipping will not have commenced at the time depicted in this figure.

Figure 16-24: Overall Layout of Dumps & Stockpiles



Source: RNC.

Figure 16-25 to Figure 16-29 illustrate the evolution of dump faces over the life of project

Figure 16-25: Layout of Dumps & Stockpiles – Year 10

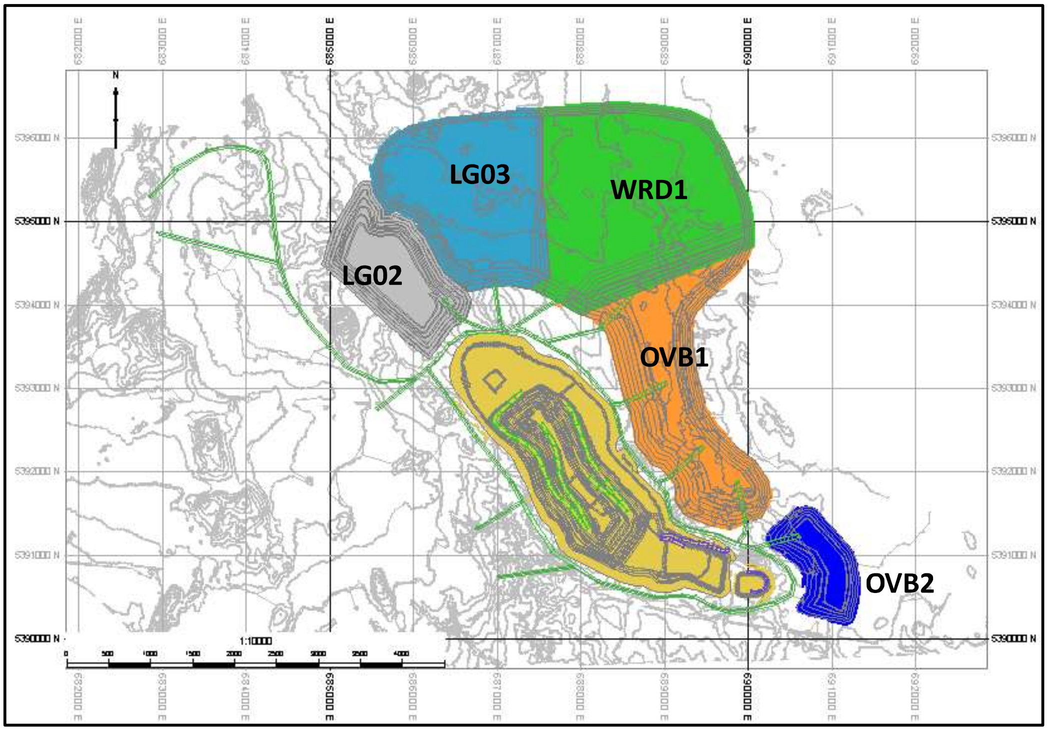


Figure 16-26: Layout of Dumps & Stockpiles – Year 15

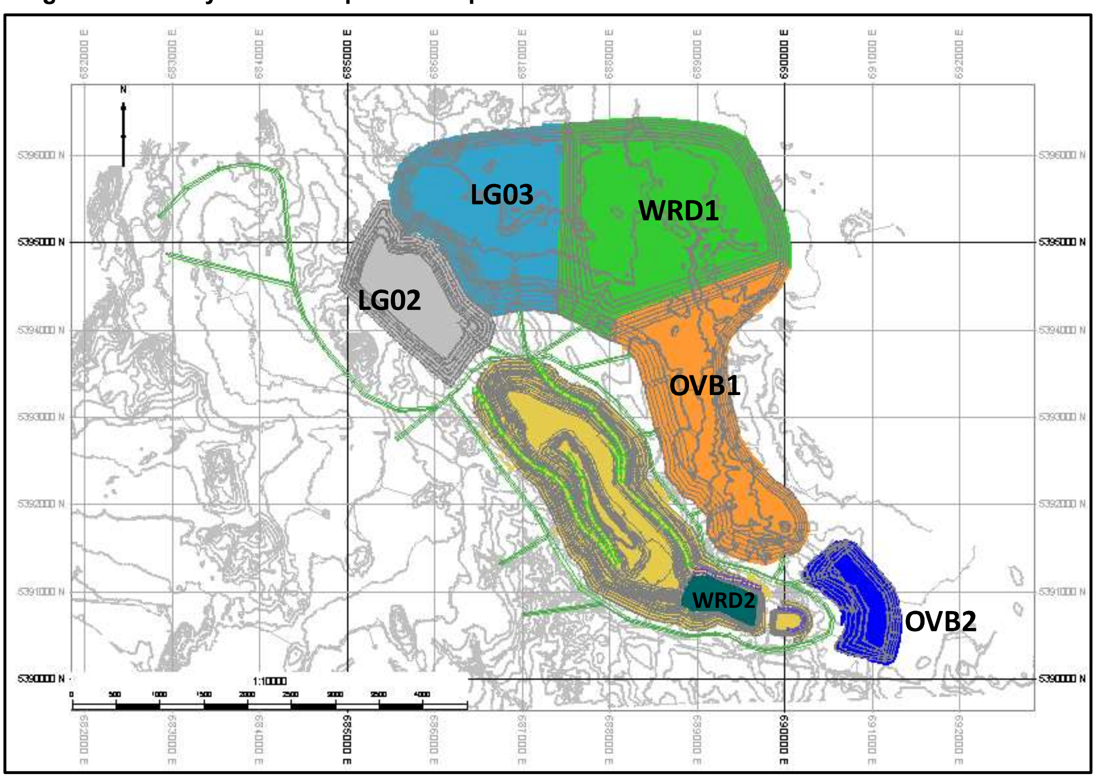


Figure 16-27: Layout of Dumps & Stockpiles – Year 19

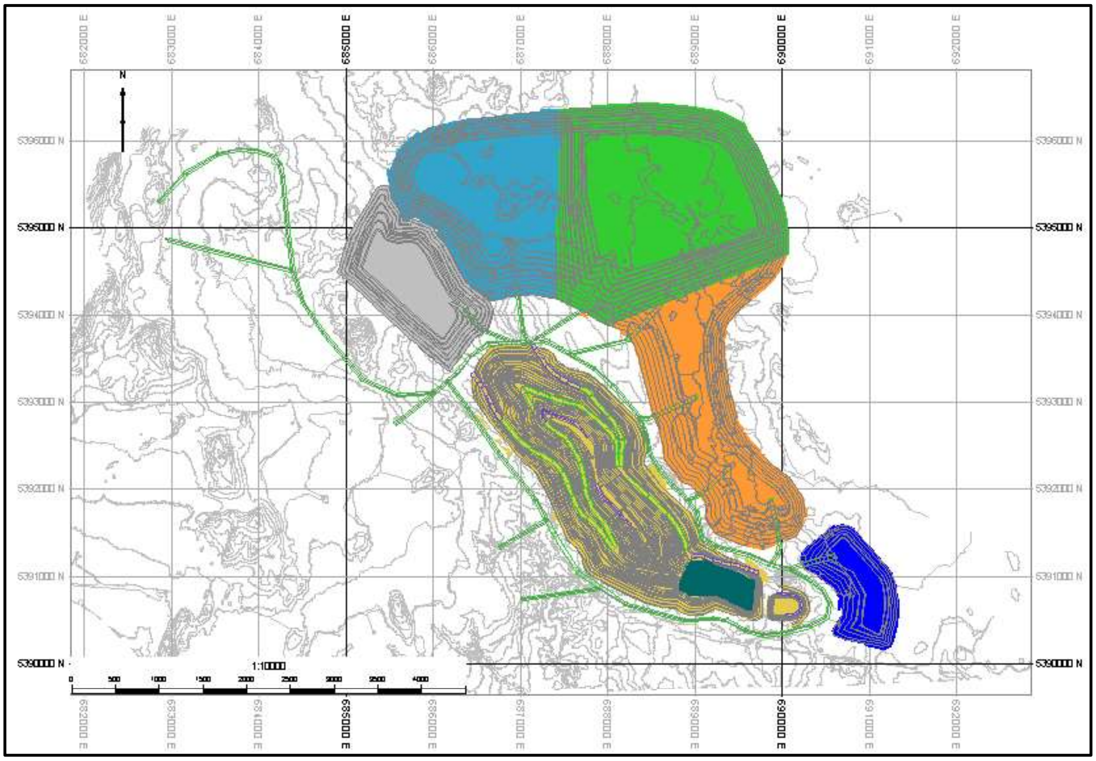


Figure 16-28: Layout of Dumps - Year 24

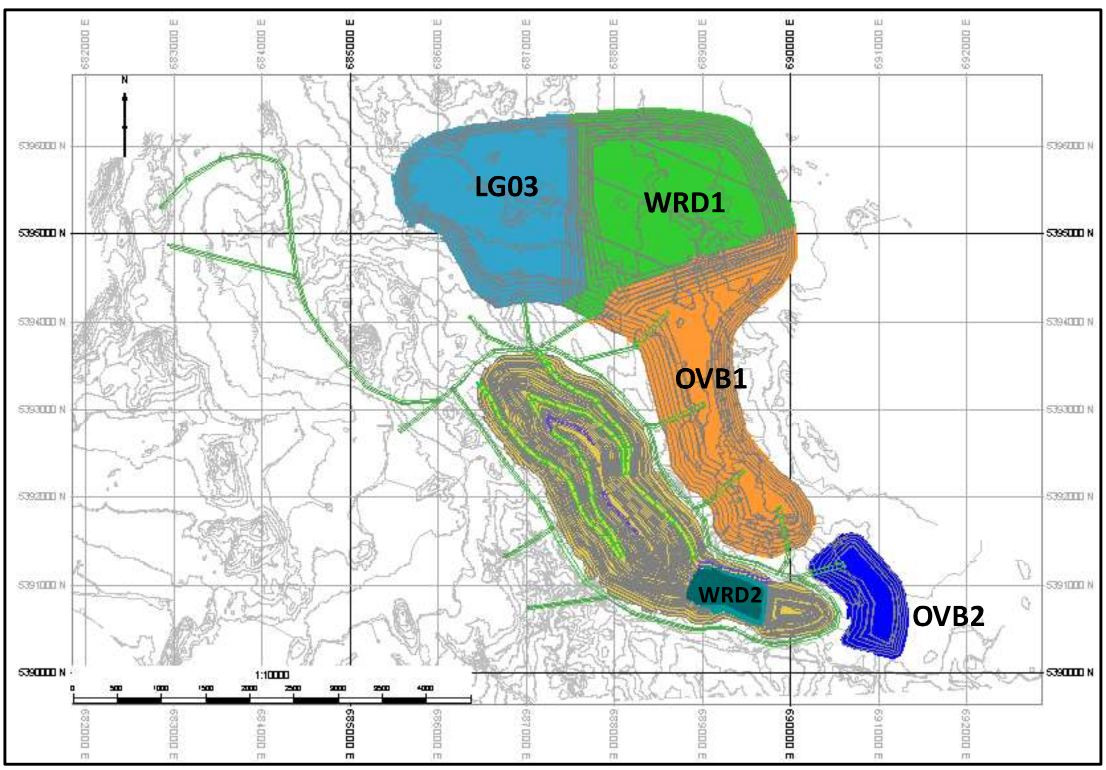
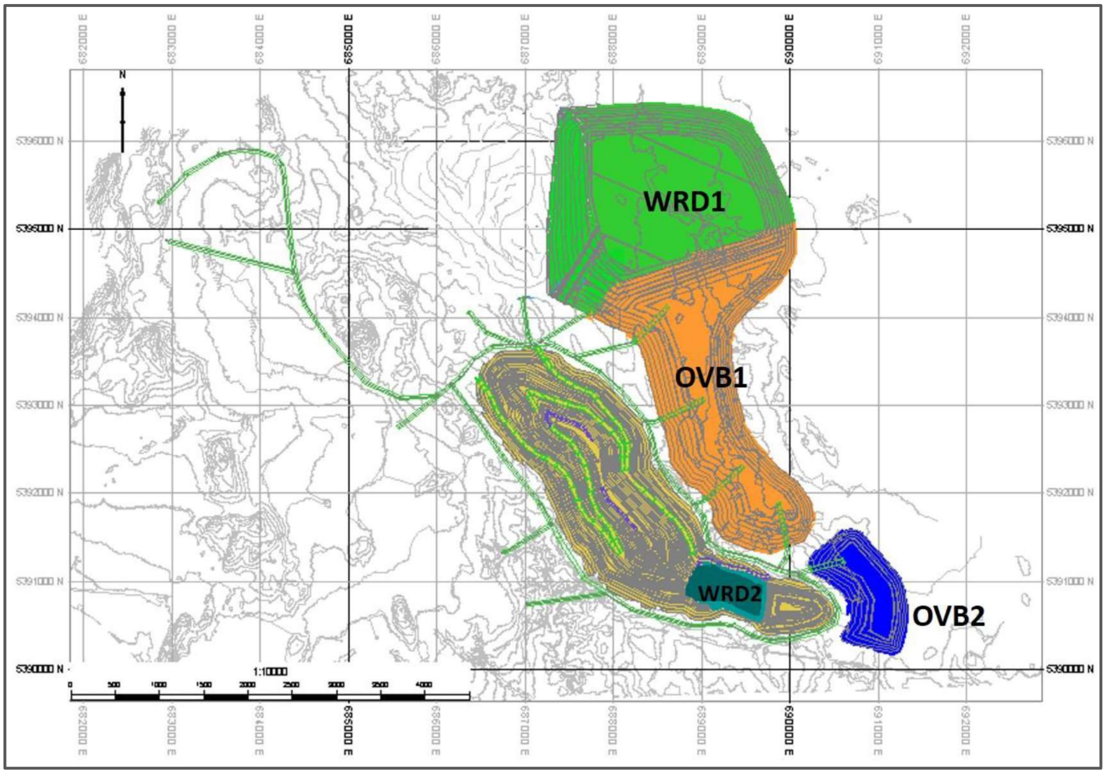


Figure 16-29: Layout of Dumps - Year 31



16.3.7 TSF

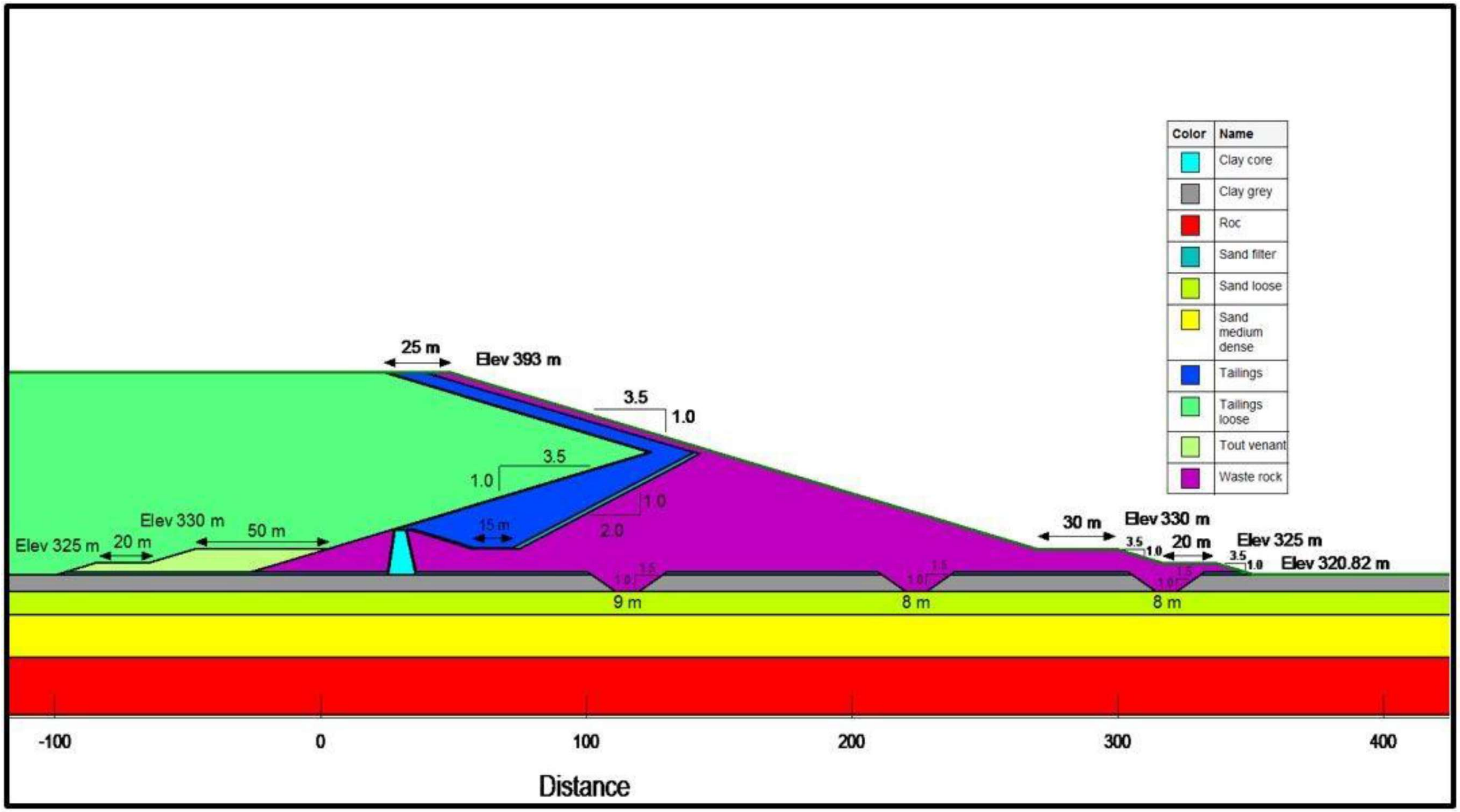
The TSF will be constructed of tailings, waste rock, sand and gravel. Starter dikes will be constructed of clay, rock and sand, and will be constructed over the first few years. Subsequence dikes raise will be constructed mainly of coarse tailings, sand and rock with rock as the principal material.

The starter dike construction will be divided into 2 phases; the starter dikes of the northern TSF will be constructed at year 0, and the southern dike will be constructed at year 1. The starter dikes will be comprised of a 4 m wide clay core, with a filter zone using sand and gravel on both upstream and downstream of the clay core then covered by a thick layer of rock. The starter dikes slopes will be constructed to 3.5H:1V, and stability berms will be required on both the upstream and downstream slopes. Shear keys will be required to be excavated for certain sections upstream of the starter dikes where the clay is thick. The crest of the northern starter dikes will be at a maximum elevation of 337 m and the crest of the southern starter dikes of the TSF will be between elevations 332 m and 332.5 m.

The dikes of the TSF will be raised annually on the downstream slopes, consisting of tailings and waste rock from year 1 to approximately year 11. Then from year 12 to 19, the dike raise will switch to an upstream construction method. The required stability berm and a shear keys will have to be constructed progressively as the dikes are raised. The downstream slope of the TSF dams will be constructed at 3.5H:1V. Figure 16-30 through Figure 16-32 provide typical cross-sections of the Eastern and Southern dams as well as the of Tailings Management Facility Recycle Water Basin dam.

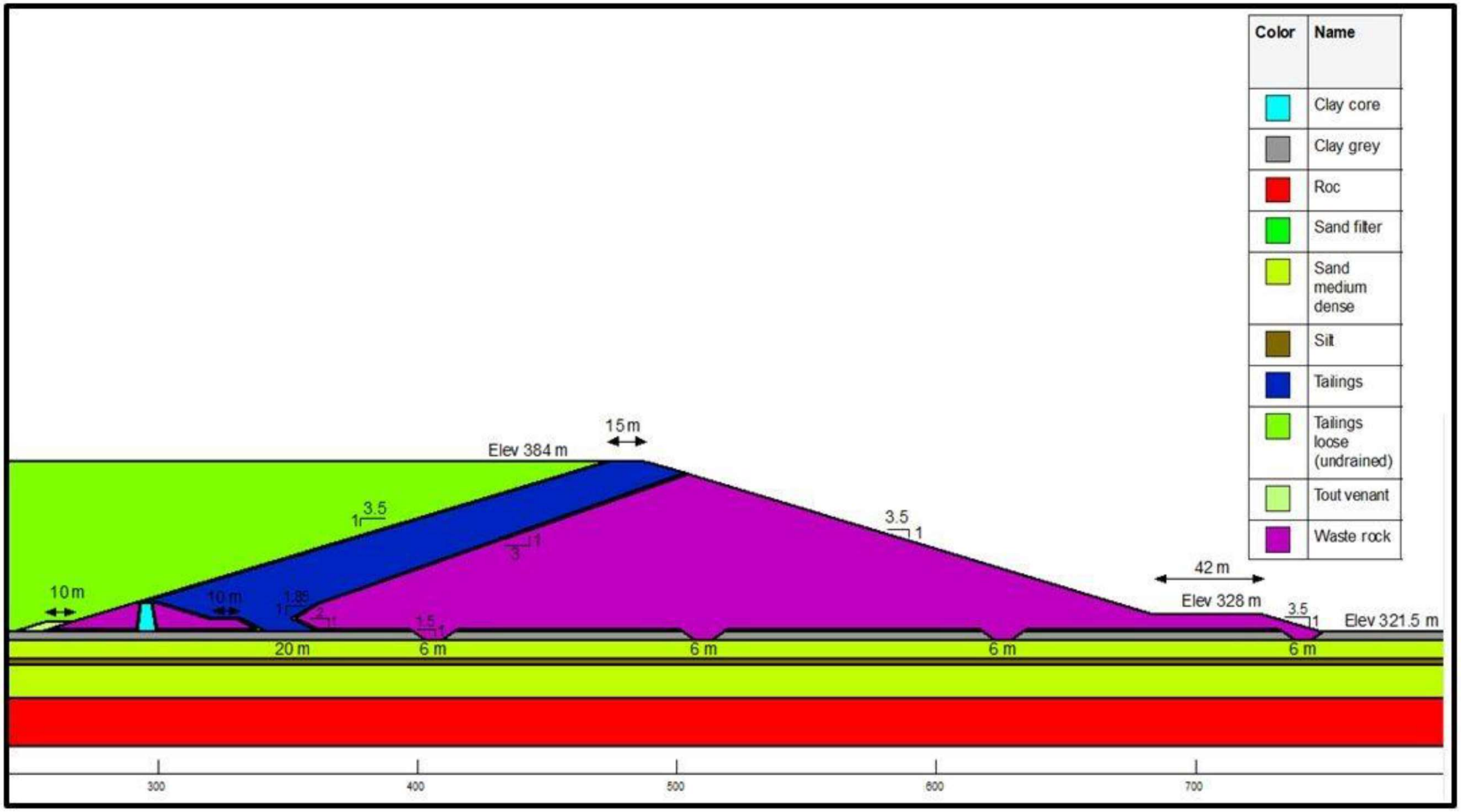
The areas where the dikes will be constructed, will first require logging, where necessary, grubbing of roots and stripping of the organic soils. The organic soils will be pushed upstream of the starter dikes and will be incorporated with the rock, where temporary stability berms are required upstream of the perimeter dikes/ dams. The clay core and filter layer would be constructed on a six-month basis (during warm months, when clay is soft enough to be handled) and peak requirements will exceed instantaneous ROM output, resulting in some stockpiling and rehandling being required. 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). The TSF will be used to impound tailings for the first 19 years of operation, after which tailings will be pumped directly into the exhausted open pit.

Figure 16-30: Typical Cross-section through TSF Eastern Dam



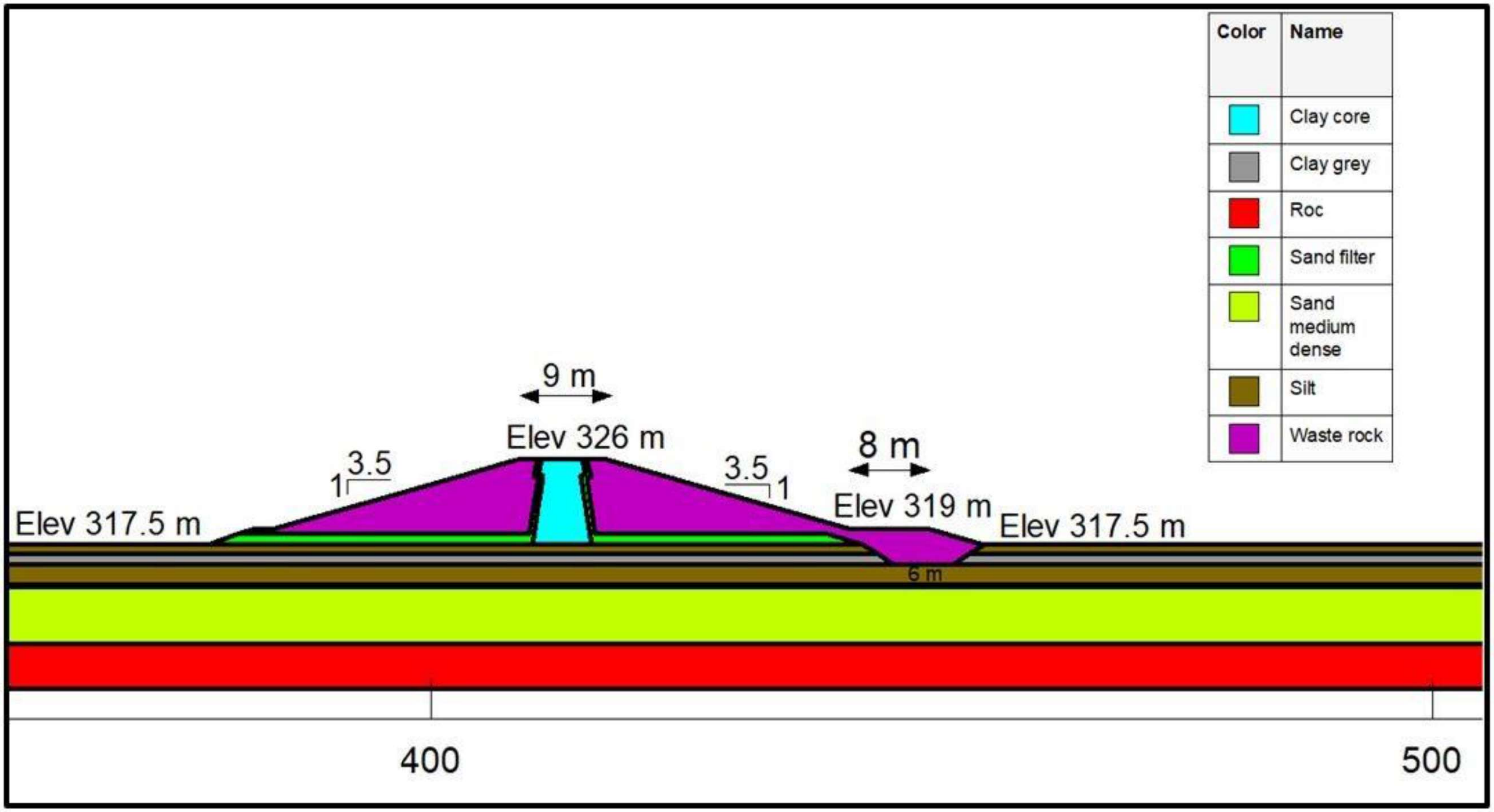
Source: Wood

Figure 16-31: Typical Cross-section through TSF Southern Dam



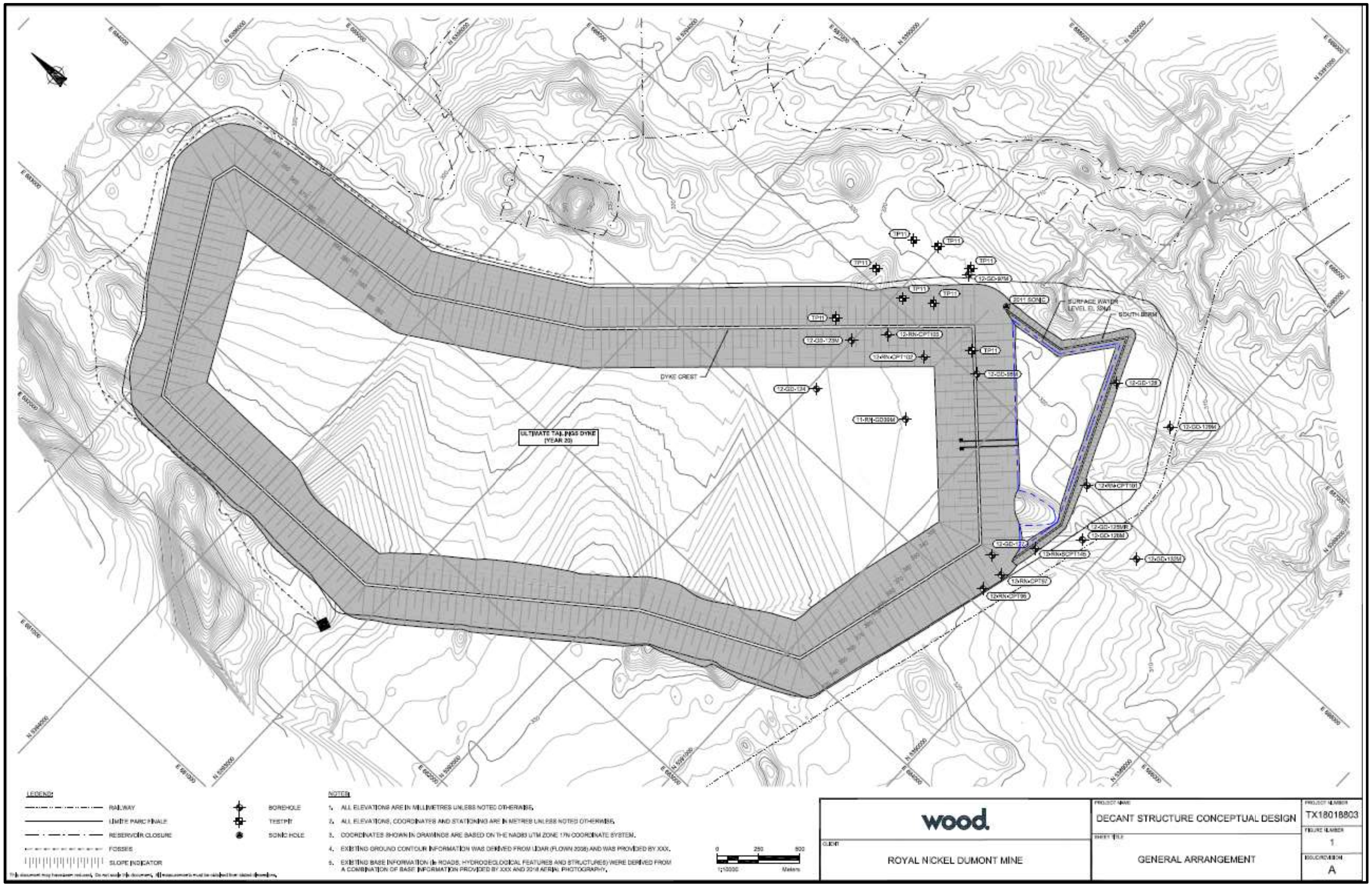
Source: Wood

Figure 16-32: Typical Cross-section through TMF Recycle Water Basin



Source: Wood

Figure 16-33: General Arrangement of TSF



Source: Wood

16.3.8 Surface Haul Roads

Pit operations will require the construction of 47.8 km of haul roads on surface, with 12.4 km being temporary and removed as the pit expands. The remainder 35.4 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 31 Mt roadstone over the life of mine.

All main haul roads on surface will be 37 m wide, which is suitable for the 290 t class trucks planned for use (ramps equipped with trolley-assist infrastructure will be 5m wider, to allow for the trolley sub-stations). 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

Expit mining operations at Dumont will be conducted by the following fleets of production mining equipment (in order of lithology that will be mined – see Figure 16.34 and 16.35):

- Areas where the depth of clay exceeds 7.5 m will be mined using a combination of 90 t class and 150 t class backhoes loading 45 t articulated trucks. The nominal application for the smaller excavator will be the actual clay while the larger excavator will be used for any associated S&G or rock in the immediate area. The backhoes will load from on top of the clay and will require the surface to be ‘armoured’ with crushed rock to prevent sinking. No drilling and blasting will be required for the overburden, while rock will be drilled using percussion drills with a nominal hole diameter of 115 mm on a bench height nominally of 5 m.
- Areas where the depth of clay is ≤ 7.5 m will be mined using a 300 t class hydraulic excavator operating in face shovel configuration. The excavator will load from the underlying S&G footwall and deliver all clay, S&G and rock into 90 t rigid body haul trucks. No drilling and blasting will be required for the overburden, while rock will be drilled using percussion drills with a nominal hole diameter of 115 mm on a bench height nominally of 5 m.
- Below the clay – S&G interface, S&G and rock will be mined on 10 m benches. Areas that are predominantly S&G will be loaded with a 600t class hydraulic excavator operating in face shovel configuration while rock will be predominately loaded with cable rope shovels. All material will be loaded into 290 t class haul trucks. Rock will be drilled using rotary drills with a nominal hole diameter of 270 mm.
- Below the rock – S&G interface, benches will be 15 m and all rock will be loaded by rope shovels into 290 t class trucks. Rock will be drilled using rotary drills with a nominal hole diameter of 311 mm.

Figure 16-34: Mining Fleets – Clay Horizon

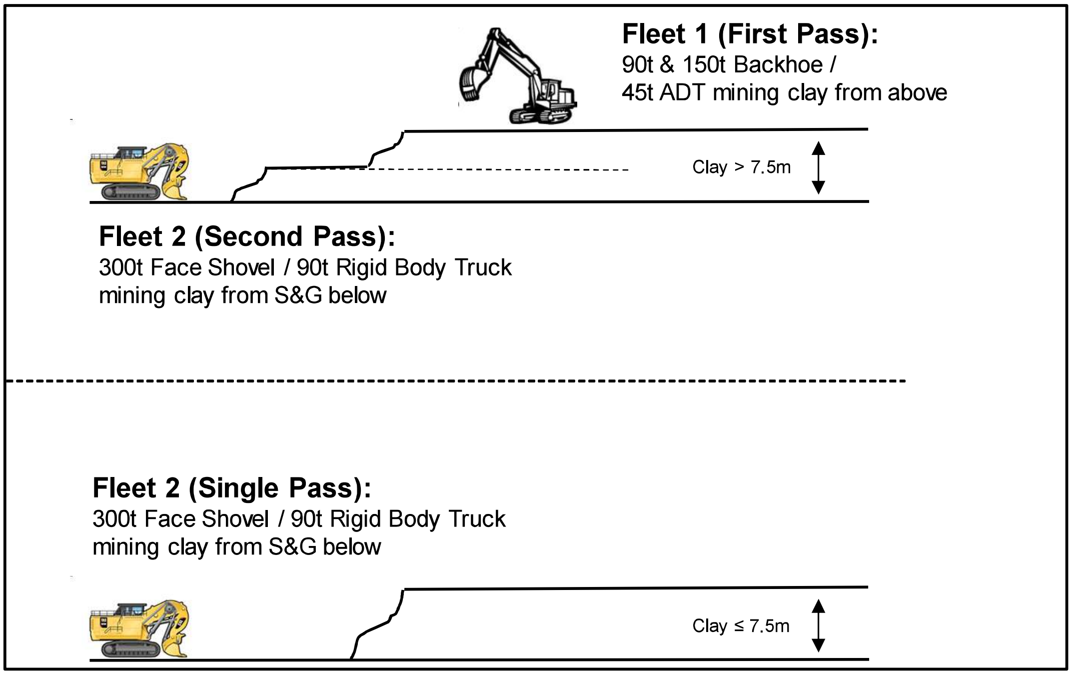
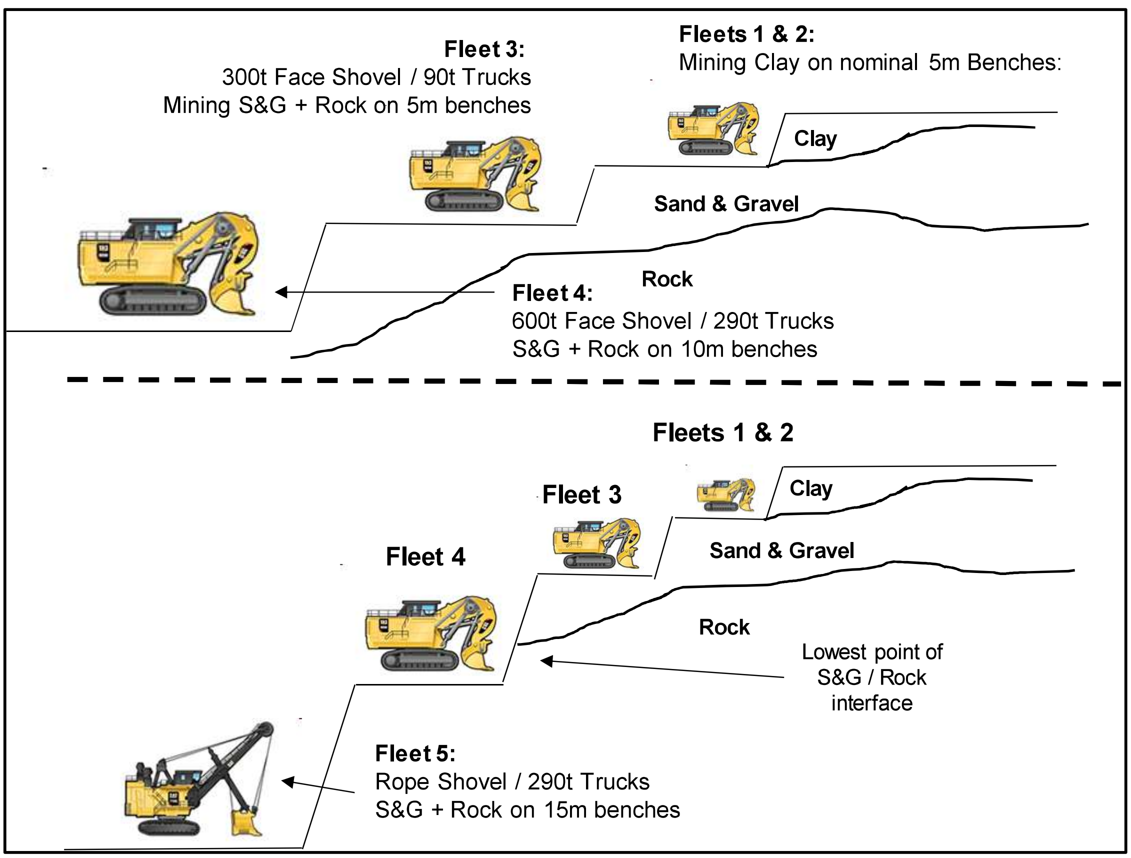


Figure 16-35: Mining Fleets – Below Clay Horizon



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.

It has been assumed that all mining fleet will be purchased by the Owner. Norascon, a local mining contractor with experience operating in similar environments has been pre-selected to assist in the mining operation, specifically to perform the clay stripping operations during the initial 2 years of pre-stripping prior to the mill be commissioned. All other equipment would be operated by the Owner.

The duty cycle for production units was estimated by first principles, based on the production 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;
- inpit sump and associated dewatering system; and
- electrical reticulation system.

The Owner's mining labour complement will average 297 persons during the life of the project, reaching a peak of 598 persons while the pit is active then dropping to an average of 89 while the low-grade stockpile is being reclaimed. The mining contractor workforce will average 91 persons over the period that the contract is active.

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 6,300 (cable shovels) to 6,000 (rotary drills, excavators and haul trucks) to 4,900 (percussion drill).

Table 16-6 summarizes the main units of the mining fleet that will be utilized, while Tables 16-7 and 16-8 summarize the size of mining fleet by year over the life of the project during expit 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.

Table 16-7: Dumont Mining Fleet

Process	Unit	Application	Size	Examples
Drilling	Percussion Drill	Rock at S&G interface	115 mm hole	Sandvik DX800
	Diesel Rotary	Rock	270 mm hole / 10 m bench 311 mm hole / 15 m bench	Cat MD 6310 Sandvik D90
	Down-The-Hole Hammer	Pre-Splitting	165 mm hole	Sandvik DI550
Loading	90 t Excavator	Backhoe Clay, 5 m bench	6 m3 bucket (8 t)	Cat 390
	150 t Excavator	Backhoe S&G + Rock, 5 m bench	8 m3 bucket (15 t)	Cat 6015 Hitachi EX 1200 Komatsu PC 1250
	300 t Excavator	Face Shovel Clay, S&G, Rock (5 m benches)	17 m3 bucket (30 t)	Cat 6030 Komatsu PC 3000
	600 t Excavator	Face Shovel S&G, Rock (10m benches)	34 m3 bucket (60 t)	Cat 6060 Hitachi EX 5600 Komatsu PC 5500 Liebherr R9600
	Electric Rope Shovel	S&G, Rock (15m benches)	60 m3 bucket (100 t)	Cat 7495 P&H 4100
Hauling	Articulated Truck	5 m bench with Backhoe	40 t payload	Cat 745 Komatsu HM 400
	90 t Truck	5 m bench with Face Shovel	90 t payload	Cat 777 Hitachi EH 1700 Komatsu HD 785
	290 t Truck	10 m and 15 m benches	290 t payload	Cat 794 Hitachi EH5000 Komatsu 930E Liebherr T276
Support Equipment	Large FEL	Rehandle to 290 t trucks	30 t payload	Cat 994 Komatsu WA1200
	Medium FEL	Rehandle to 90 t trucks	20t payload	Cat 992 Komatsu WA900
	Small FEL	Rehandle to articulated trucks	10 t payload	Cat 988
	Large Dozer	Support Large Excavator & Rope Shovel	800 HP	Cat D11 Komatsu D475
	Medium Dozer	Support 300 t Excavator	600 HP	Cat D10 Komatsu D375
	Small Dozer	Support Backhoes	300 HP	Cat D8
	Wheel Dozer	Road Clean Up	600 HP	Cat 844
	Large Grader	Maintain Permanent Roads	18 ft blade	Cat 18M
	Small Grader	Maintain Roads in Overburden	14ft blade	Cat 14M
	Water Tanker	Suppress Dust	Nominal 90t payload	Cat 777 Komatsu HD 785

Table 16-8: Dumont Mining Fleet by Year during Expit Operations

Unit	Fleet Levels During Expit Operations																								
	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Yr 18	Yr 19	Yr 20	Yr 21	Yr 22	Yr 23
Percussion Drill	1	1	1	1	0	1	1	0	1	1	1	1	0	0	0	0	0	0	0	0	0	1	0	0	0
Rotary Drill	0	1	2	2	3	3	3	4	3	3	3	4	4	4	5	4	3	3	3	2	1	0	1	1	1
Pre-Split Drill	0	0	0	0	0	0	0	0	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0	0
90t Excavator	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	0	0
150t Excavator	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0	1	0	0	0	0
300t Excavator	1	2	2	2	2	1	2	0	1	2	2	2	1	0	0	0	0	0	0	0	0	2	1	0	0
600t Excavator	0	1	2	2	2	2	2	2	2	2	2	2	1	1	2	2	1	1	0	0	0	0	1	1	1
Rope Shovel	0	0	1	1	2	2	2	2	2	2	2	3	3	3	3	3	3	3	3	2	2	1	1	1	1
Articulated Truck	15	16	3	14	2	14	11	2	9	20	19	2	2	2	2	2	2	2	1	1	2	0	0	0	0
90t Truck	6	15	12	16	10	9	11	3	6	17	17	17	7	3	4	3	4	4	4	4	3	6	3	1	2
290t Truck	0	2	11	17	20	24	24	28	25	25	29	36	40	45	45	34	42	45	45	31	16	5	7	9	11
Small Dozer	1	2	2	2	1	2	2	1	2	2	2	1	1	1	1	1	1	1	1	1	0	0	0	0	0
Medium Dozer	1	1	1	1	0	0	1	0	0	1	1	1	0	0	0	0	0	0	0	0	0	0	0	0	0
Large Dozer	0	1	2	2	2	2	2	2	2	2	2	3	2	2	3	3	2	3	3	2	1	0	0	0	0
Wheel Dozer	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Small Grader	1	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Large Grader	1	1	2	2	2	2	2	2	2	3	3	3	3	3	3	2	3	3	3	2	1	1	1	1	1
Tanker	1	1	2	2	2	2	2	2	2	3	3	3	3	3	3	2	3	3	3	2	1	1	1	1	1
Small Front End Loader	1	1	0	1	0	1	1	0	1	1	1	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Medium Front End Loader	0	0	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Large Front End Loader	0	0	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

Table 16-9: Dumont Mining Fleet by Year during Stockpile Reclaim

Unit	Fleet Levels During Reclaim Operations							
	Yr 24	Yr 25	Yr 26	Yr 27	Yr 28	Yr 29	Yr 30	Yr 31
150t Excavator	0	0	0	0	0	0	0	0
300t Excavator	0	0	0	0	0	0	0	0
600t Excavator	1	1	1	0	0	0	0	0
Rope Shovel	1	1	1	1	1	1	1	1
90t Truck	1	2	2	1	1	1	1	1
290t Truck	9	11	8	7	7	7	7	7
Wheel Dozer	1	1	1	1	1	1	1	1
Large Grader	1	1	1	1	1	1	1	1
Tanker	1	1	1	1	1	1	1	1
Medium Front End Loader	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. Drilling patterns will range from approximately 4m (square) for 5 m benches, increasing to 8 – 9 m square for 10m benches and 9 – 10 m square on 15 m benches. In all cases, the smaller dimension is for higher S.G. gabbro and basalt while the increased dimension is for lower S.G. peridotite and dunite. Over the life of mine, an average of 206 tonnes is yielded per metre drilled.

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:

- 115 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 76% 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 41.3 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 these delays, overall rotary drill productivity is estimated at 26.4 m/h. With this productivity, the production plan can be achieved with a fleet that reaches a maximum strength of five rotary units, supplemented by a single percussion drill. The mine plan requires a total of 311,000 engine hours for the rotary units, which will be achievable without purchasing any replacement units.

The productivity calculations take account of the planned implementation of a High Precision GPS (HPGPS) guidance, monitoring and rock recognition system for the fleet of rotary drills. A key benefit of the system is elimination of the requirement for surveyors to stake the X-Y collar positions of holes and conduct subsequent checks of drilled holes. Other benefits of the system include:

- A more correct determination of collar elevation, improving footwall control and minimizing over-drilling.
- Recognition of rock types encountered down the hole will allow more accurate mapping of lithologies, which will improve planning – including the design of blast patterns to reduce dilution and ore loss at contacts.
- The technology ensures more consistent drill performance, particularly when lithologies change down the hole. In turn, this ensures closer adherence to the drill instructions, regardless of the operator experience and level of skills.

The HPGPS guidance, monitoring and rock recognition system is also an essential building block for operation of Autonomous Drill Systems (ADS), which is an opportunity that will be discussed in Chapter 24.

Rock would be blasted using emulsion produced at an on-site facility operated by the explosives OEM.

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, hole spacings ranging from, a minimum of 1.75m in basalt to a maximum of 2.7 m in dunite. A weighted average of 2.37 m was then estimated, based on the volumes of the four different rock types.

The total pre-splitting requirement was based on a measured 270.2 km of total final wall perimeter over the 36 benches that would be mined. The resulting 114,075 presplit holes would require 1,758 km drilling. Pre-splitting was assumed to start following the completion of Phase 4, when the initial final walls are established in the southeast extension. The total metreage of pre-splitting was assumed to be divided evenly into the remaining duration of expit mining (with pre-splitting assumed to be finished 6 months before the final production blast), resulting in quarterly rates of 2,377 holes or 36.7 km.

Pre-split drilling would be accomplished by a single percussion rig equipped with down-the-hole hammers. Similar penetration rates as estimated for the percussion drill have been assumed. The pre-split rig would be equipped with the same HPGPS guidance, monitoring and rock recognition system as the rotary drills.

16.4.2.3 Loading & Hauling

As outlined previously, multiple fleets of load and haul equipment will be employed to ensure the various rock types at Dumont will be mined most productively. Areas where the clay thickness exceeds 7.5 m will be stripped using 90 t and 150 t backhoe excavators while the remaining areas where mining will be conducted on a nominal 5 m bench height will be stripped using 300 t excavators in face shovel configuration. All three of the smaller excavators will be diesel powered and they will load a combined 7% of the 2,080 Mt expit tonnage, or 6% of the total tonnage that includes a further 511 Mt reclaimed from stockpiles. The 600 t excavators, which will operate predominantly on 10 m benches as well as the stockpiles, will be electrically powered and load 22% of the total tonnage. The remaining 72% of total material will be loaded by rope shovels, operating predominantly on 15 m benches and the stockpiles. Criteria used to calculate the productivity of various loading units are given in and in Table 16-11.

Table 16-10: Loading Design Criteria – Excavators Operating on Nominal 5 m Benches

Parameter	units	Clay 90t Excavator	S&G + Rock 150t Excavator	 300t Excavator
Average Bucket Factor	Tonnes	8	15	30
Average Truck Payload	Tonnes	38	38	88
Theoretical Passes per Load	Number	4.80	3.00	3.00
Additional Passes per Load	Number	0.25	0.25	0.25
Total Passes per Load	Number	5.05	3.25	3.25
Cycle Time per Bucket	Seconds	30	30	30
Spot Time	Seconds	30	30	30
Total time to Load Truck	Seconds	182	128	128
Engine hrs per year	Hours	5,800	5,800	5,800
Non-productive time per year1	Hours	1,400	1,400	1,800
Max productivity per unit	**Mt/a**	**3.3**	**4.7**	**10.0**

Notes: 1. includes blast delays, equipment moves and waiting for trucks

Table 16-11: Loading Design Criteria – Equipment Operating on 10 m & 15 m Benches

Parameter	units	600t Excavator	Rope Shovel
Average Bucket Factor	Tonnes	61	100
Average Truck Payload	Tonnes	285	285
Theoretical Passes per Load	Number	4.60	3.00
Additional Passes per Load	Number	0.25	0.25
Total Passes per Load	Number	4.85	3.25
Cycle Time per Bucket	Seconds	30	30
Spot Time	Seconds	30	30
Total time to Load Truck	Seconds	176	128
Engine hrs per year	Hours	5,800	6,300
Non-productive time per year1	Hours	2,400	1,850
Max productivity per unit	**Mt/a**	**20.0**	**36.0**

Notes: 1. includes blast delays, equipment moves and waiting for trucks

The usage and size of fleets for each of the various loading units will be as follows:

- 90 t Excavator usage = 36 k engine hours, maximum fleet: = 1 unit
- 150 t Excavator usage = 50 k engine hours, maximum fleet = 1 unit
- 300 t Excavator usage = 102 k engine hours, maximum fleet = 2 units
- 600 t Excavator usage = 134 k engine hours, maximum fleet = 2 units
- Shovel usage = 303 k engine hours, maximum fleet = 3 units

The life of mine engine hours that will be accumulated by various loading units falls within the economic life for each class of machine and it will not be necessary to purchase any replacement units.

In response to the poor footwall conditions that will be experienced in areas of deeper clay, both the 90 t and 150 t excavators will load articulated trucks with a 40 t payload. The 300 t excavator will be operating in areas where the footwall of 5 m benches is situated in S&G and it will be more productive to load 90 t rigid body trucks.

The optimal size of truck that 600t excavators and shovels will load was subjected to a trade-off study during the Penultimate LG Optimization. This trade-off concluded that project economics could be improved by upsizing the 230 t trucks selected for the 2013 FS to 290 t. While the tare weight-to-payload ratio for the larger trucks remains generally inferior and results in slower uphill tramming speeds and/or increased diesel consumption, these shortcomings are negated with use of trolley assist. The larger truck has added benefits of a more productive shovel match and reduced labour intensity.

Trolley assist is a proven technology that, on uphill hauls, supplies power to the wheel motors of a diesel-electric truck from an overhead line rather than the onboard generator. A more complete description of the technology and its application at Dumont is given under the later section on Mining Infrastructure.

Criteria used to calculate the productivity of various hauling units is given in Table 16-12.

Table 16-12: Hauling Design Criteria

Parameter		40t ADT	90t	290t
Payload	tonnes	38	88	285
Loading Time1	min / load	4.64	2.13	2.13
Dumping Time	min / load	2	1	1
Queuing Time	min / load	2	2	2
Speed - inpit flat (empty & full)	km/h	19	25	25
Speed - expit (empty & full)	km/h	26	35	35
Speed - uphill loaded conventional	km/h	10	12	13
Speed - uphill loaded trolley	km/h	n/a	n/a	23
Speed - downhill empty	km/h	23	30	30
Average Cycle Time[2]	**min / load**	**31.9**	**23.6**	**30.3**
Average Fuel Burn[2]	**L/h**	**28**	**87**	**239**
Average productivity per unit[2]	**Mt/a**	**0.4**	**1.3**	**3.3**

Notes: 1. Loading time varies as a function of the loading unit 2.Averages achieved over the life of mine, given selected mine plan.

All three sizes of truck were assumed to achieve average availability and utilization of 85% and 80%, respectively. This results in a total of 5,960 engine hours per unit annually. The usage and size of fleets for each of the sizes of trucks will be as follows:

- 40t Articulated Truck usage = 705k engine hours, maximum fleet: = 20 units
- 90t Truck usage = 1.033k engine hours, maximum fleet = 17 units
- 290t Truck usage = 3,883k engine hours, maximum fleet = 45 units

Maximum fleet sizes will be dictated by the surge in mining rate and lengthening hauls (mainly due to expanding waste dump dimensions) experienced between Q46 – Q54 of mill life, which corresponds to years 14 – 16 of an overall 33 year project life (including pre-stripping and stockpile reclamation). It will thus be possible to retire units with high engine hours in the later years of project life and replace them with units purchased for the surge in mining and it will not be necessary to purchase any replacement units.

The efficiency of the load and haul operation will be maximized through purchase of a number of technology systems, including:

- Fleet Management System (FMS), which assign, track and monitor the fleet of mobile equipment. Assignments are continually updated to take account of the current status of all equipment and thus ensure the plan is achieved with the minimal resources and operating expenditure.

- Tire Monitoring Systems, which entail deployment of sensors within the tires on haul trucks, FEL and wheel dozers. These report the real-time pressure and heat data that can be used to generate assignments that maximize both safety and tire life. For example, a haul truck with tires approaching the thermal limit that would lead to a heat separation could be re-routed on profiles where lower speeds are realized, allowing the tires to cool.
- HPGPS Guidance and Monitoring, which is similar to the system that will be deployed on drills. These will allow loading units to load more closely to planned X-Y-Z boundaries, reducing dilution and ore loss while improving footwall conditions. The system would also be employed on support equipment such as dozers and graders, to ensure footwall and ramp conditions are maintained to the highest standard. For example, the FMS may identify a section of road where haul trucks are reducing speed. A dozer and/or grader would then be automatically dispatched to the exact location where cutting and/or filling was required to return the road to grade.
- Loading Unit Tooth Detection, which use cameras to monitor the status of dipper teeth and send an alarm when a tooth goes missing. This allows identification of the haul truck containing the broken tooth before the tooth is delivered to a crusher.
- Payload Monitoring, which provide operators with the dipper-by-dipper mass delivered to a haul truck to ensure over- and under-loading is minimized. In turn, this maximizes utilization of haul trucks while minimizing excessive wear on the haul truck frame, suspension and tires.

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. The backhoe excavators would be supported by smaller 35 t class units (e.g., Cat D8), the 300 t excavator would be supported by 50 t class units (e.g., Cat D9) while the large excavators and rope shovels would be supported by 100 t class units (e.g., Cat D11).
- Rubber-tired dozers, for lighter construction and general cleanup. The fleet requirements were estimated based on the empirical relationship of 0.25 operating dozers for every operating loading unit. Equipment of the 35 t class would be selected (e.g. Cat 844).
- Graders. The fleet requirements were estimated based on the empirical relationship of 1 grader for every 20 trucks. Two size graders would be employed. Units with a 14 ft blade would be used by the Contractor initially. The Owner fleet would subsequently utilize larger units with an 18 ft blade.
- Water tankers, for suppressing dust. Requirements were based on the empirical relationship of 1 tanker for every 20 trucks. Tankers would be modified 90 t haul trucks.
- Front end loaders (FEL), for construction and clean-up activities including the loading of roadstone into trucks. A small FEL with bucket capacity of 10 t would be employed to support operations in clay, while a larger unit with a 20 t payload would support other operations. This unit would also be used to load roadstone onto 90 t trucks. A large FEL with 35 t payload would be available to as a supplemental loading unit, it required.
- Utility excavators, for activities such as construction, scaling highwalls and breaking oversize.

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 empirical factor of one bay for every five haul trucks.

At start-up, the workshop will comprise six bays. As the fleet expands in response to increased production rate and longer hauls, the workshop will subsequently be expanded to 10 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

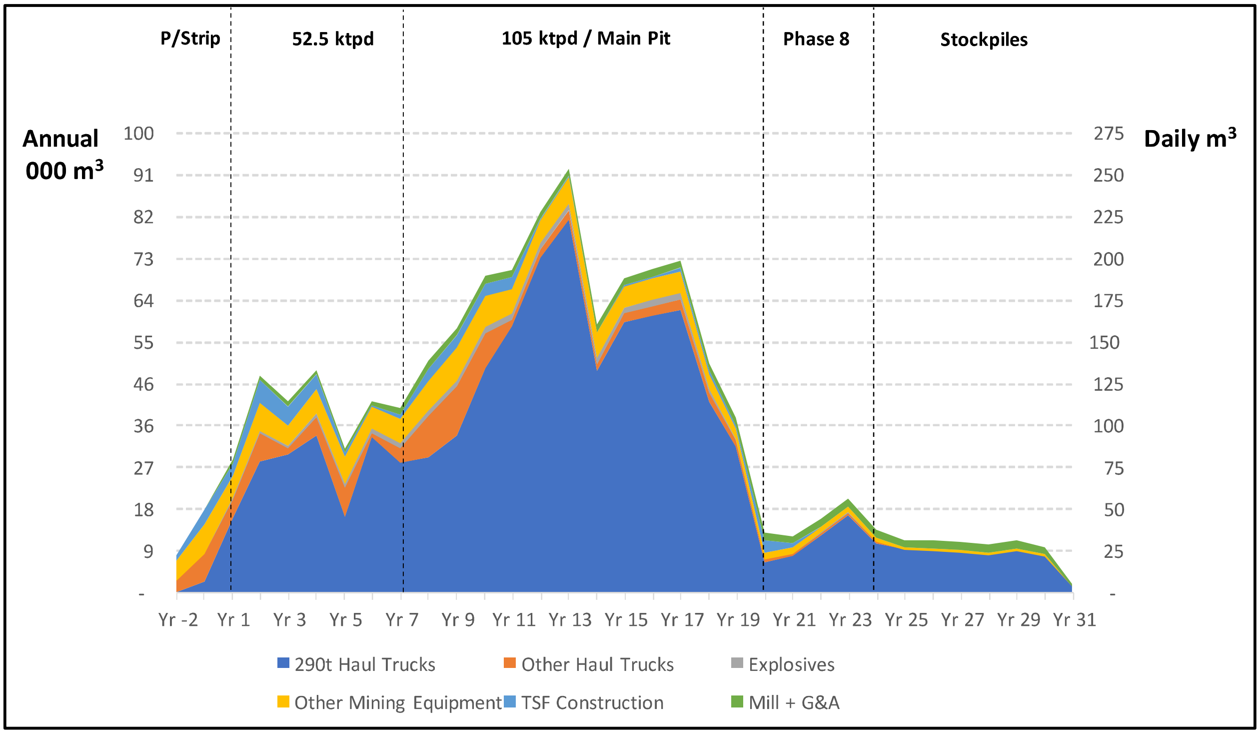
Fuel consumption has been estimated from first principles, based on the burn rate for the various pieces of equipment that would be operated and specific duty cycle. Figure 16-36 illustrates that the 290 t haul trucks are expected to consume 75% of the 1.23 Mm^3 fuel required over the life of project. In the event trolley-assist were not employed, diesel consumption of the large trucks would increase by 45% and life of project diesel consumption would total 1.65 Mm^3. The combined consumption of smaller haul trucks and other mining equipment, including that employed in the construction of the TSF, will be 20% of the total or 0.24 Mm3. One opportunity that was not investigated during this study is the use of Battery Electric Vehicles (BEV) for these applications instead of conventional diesel powered mining equipment. Diesel is a key constituent of the Ammonium Nitrate – Fuel Oil (ANFO) based explosives that will be used and will total 2% of total fuel consumed. The remaining 3% will be used by smaller vehicles operating at the mill and administration areas. These machines could also possibly be replaced by BEV.

Figure 16-36 also illustrates that average daily consumption rises from a rate of 50 m3/day during the pre-strip to peak at 125 m^3/day prior to the mill expansion. The fuel farm has been sized to allow for surge, with the initial capacity of 900 m^3 providing 6 days capacity during the period of peak pre-expansion consumption. Following expansion, daily consumption rises steadily to a maximum of 250 m^3/day during year 13. The fuel farm is correspondingly increased to 1,650 m^3, providing over six days storage for this year of peak consumption. During mining of Phase 8, consumption reduces to a peak of 50 m^3/d before dropping to a consistent 30 m^3/d following the end of pit operations.

Equipment would be fuelled at a diesel fueling station located adjacent to the workshop complex. A modified 90 t 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 16-36: Diesel Consumption



Source: RNC.

16.4.3.3 Explosives Manufacture Plant

Rock will be blasted using emulsion explosives. There are two prospective OEMs that could supply explosives to the project, Dyno-Nobel or Orica. For the initial 15 months of operation (during pre-stripping), while an explosives facility is being constructed on site, finished bulk explosives products will be sourced from one of the two locations:

- Orica – Plant located at the Canadian Malartic mine, approximately 90 km from the Dumont mine site; or
- Dyno-Nobel – Plant in North Bay, approximately 400 km from the Dumont mine site.

Average daily consumption during this period would be 7 tonnes, with a peak of 10 tonnes. As explosives would be trucked using 12.5 t bulk delivery trucks, daily traffic would be a maximum of 1 truck. On-site storage facilities will be erected for the product and a bulk re-pump truck will be used to deliver the product to blast holes. Two magazines will also be erected for the storage of boosters and detonators.

Following commissioning of the on-site facility, emulsion will be manufactured at site. The main ingredient is Ammonium Nitrate Solution (ANSOL), which is non explosive and can be delivered by conventional rail tankers of 90t payload. At the peak production rate of approximately 360 ktpd rock, explosives demand would be 90 tonnes per day. As rail service along the spur line is not daily, the peak delivery rate would be bi-weekly shipments of 4 – 5 tankers, with the excess accounting for surge requirements.

The technology under consideration for Dumont would result in explosives manufacture in the bulk delivery truck that is used to charge the holes, meaning that materials stored on site would not be considered explosive. In turn, this would allow the explosives storage facilities to be located no further than 270m from other buildings, reducing the footprint of surface operations and cycle times for the bulk delivery trucks.

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 the capital cost associated with all equipment and facilities will be borne by the explosives OEM and recovered by way of a service charge applied once the project begins generating cashflow. Decommissioning of this facility at the end of mine life will 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 7,000 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 crushed rock has been estimated at 32.7 Mt over the life of project, comprising:
  - 0.4 Mt of crushed rock used for blast hole stemming.
  - 1.0 Mt of crushed rock used for ‘armouring’ the loading surface in areas of deeper clay, where 90 t and 150 t backhoes would load articulated trucks.
  - 31.3 Mt of roadstone on inpit roads that will be reloaded as the roads are mined out

Feed to the roadstone plant will be delivered using 90t haul trucks, while crushed roadstone will be loaded into the same size haul trucks with a front-end loader.

16.4.3.5 Trolley Assist

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 does 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 on the truck to enable collection of electric power from overhead lines. The lines are supported by rigid poles, and electricity is fed to the line by a direct current (DC) 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 16-37 provides views of a trolley-equipped truck and infrastructure at Boliden’s Aitik mine in Sweden.

Figure 16-37: Trolley-Assist at Aitik

Source: RNC.

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.

Savings realized from trolley assist can be categorized as follows:

- Energy cost savings – which occur as power is supplied to wheel motors 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 74% possible for the class of truck planned for use at Dumont. This allows the mine plan to be achieved with fewer trucks and an associated reduction in labour. The reduction in truck fleet has additional benefits by reducing congestion associated with 'bunching' of units following shift change and other stoppages.
- 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 hydrocarbons.

The savings associated with trolley-assist are partially offset by costs associated with operating the system that include:

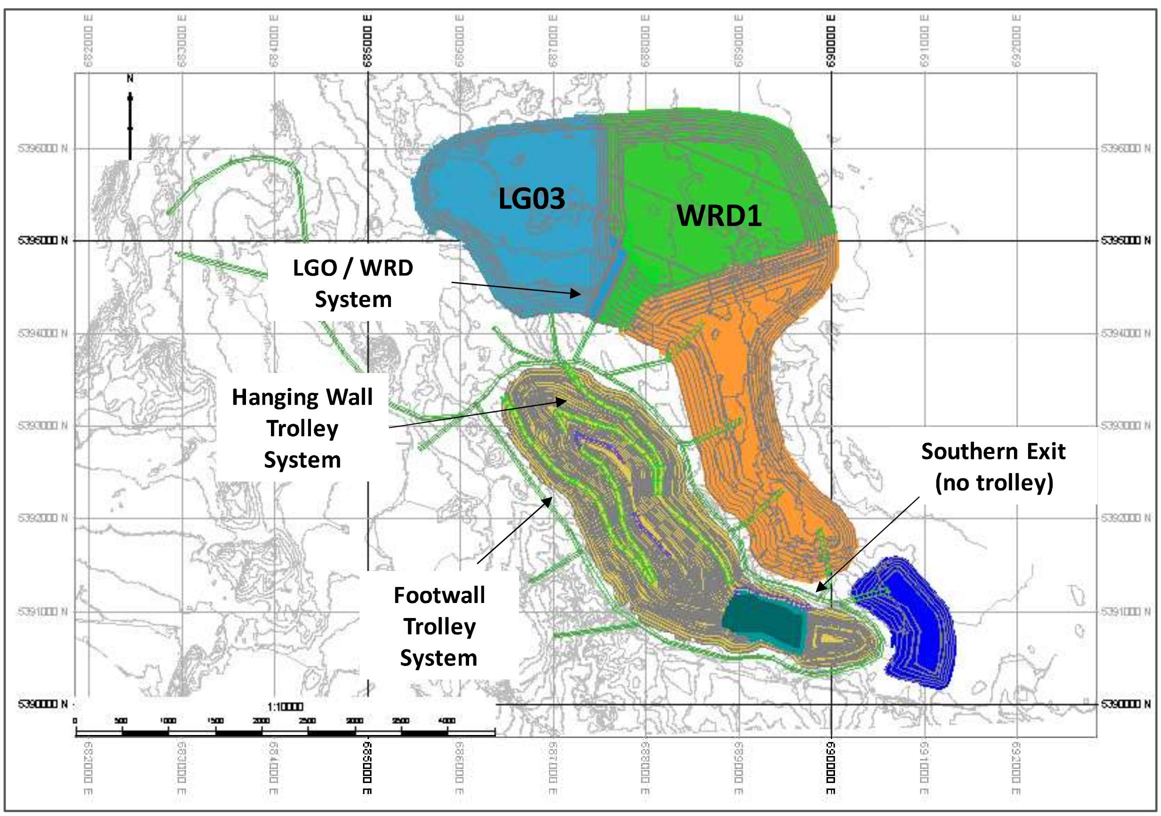
- Fixed infrastructure – including the trolley line, pole and substation.
- Truck infrastructure – including the pantograph and associated on-board control devices.
- Ongoing maintenance of fixed and truck-based infrastructure.
- Wider ramps –to accommodate trolley-assist infrastructure (primarily the sub stations), the width of equipped ramps would be increased by 5 m. This could result in flatter overall slopes and increased waste stripping.

Application of Trolley-Assist at Dumont

The current pit design was optimized for use of trolley assist. Compared to design in 2013, there is one fewer pit exits, which results in a higher density of traffic on the remaining ramps with an associated improved utilization (measured in tonnes x kilometres) of the trolley infrastructure. Each of the ramps was evaluated and the following found to be optimal (see Figure 16-38):

- The Main Hanging Wall ramp will be equipped for trolley starting in Year3. The ramp will ultimately extend to Bench 26 (RL 660 m) or two benches above its ultimate depth at RL 630 m. The 2.7 m tonnes that will be hauled from the lower benches does not justify the installation of a line and the ramp width was correspondingly reduced to 37 m. Maximum length of this system is 3.3 km.
- The Main Footwall ramp will be equipped for trolley starting in Year5. The ramp will ultimately extend to Bench 33 (RL 585 m) or six benches above its ultimate depth at RL 495 m. While the 36 Mt that will be hauled from the lower benches would justify a partial extension of the line, it was found to be more economic to reduce the ramp width and total waste stripping. Maximum length of this system is 5.0 km.
- WRD1 will be equipped for trolley starting in Year5. Material destined for LGO3 will also use this ramp, which will reach an ultimate height of 70 m, for a maximum system length of 800 m. When LGO3 is reclaimed, empty trucks en-route to the loading face will continue to use the trolley system.
- The southern pit exit would achieve break-even economics if equipped for trolley. For this reason, it has been left unequipped with a reduced ramp width and associated stripping requirements.

Figure 16-38: Trolley-Assist Equipped Ramps at Dumont



Source: RNC.

A key assumption in the design, that was based on operating experience at Palabora as well as at Rossing and Goldstrike, is that steady-state utilization of each trolley equipped ramp (measured in percentage of potential tonnes x equipped km) would be 90%. It was also assumed each new inpit segment would take 15 months to reach this utilization, with a key constraint being the time required to open a bench sufficiently that fly rock from blasting would not damage the system. For the dump, where no blasting would take place, the time required to reach steady-state was assumed to be 9 months. Over the life of mine, 60% of total uphill tonnes x km travelled by the 290 t trucks would be on trolley-assist. The smaller 90 t and articulated trucks would not be equipped for trolley assist.

The single most important issue to be addressed in order for trolley to be successfully implemented at Dumont is the impact of weather – particularly the spring frost – 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 using 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). Boliden's Aitik mine in Sweden, which experiences similar amounts of snowfall as Dumont, began operating a trolley assist system in Q4 2018. 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.

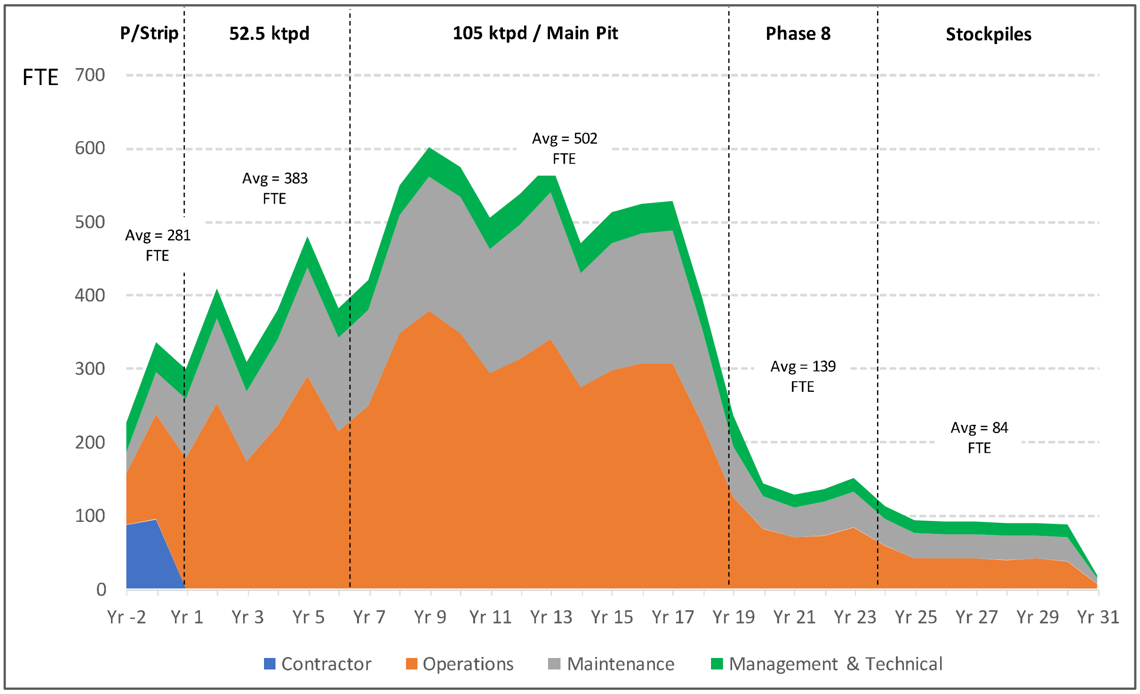
16.4.4 Labour

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. Included in the numbers illustrated are personnel associated with construction of the TSF as well as contractors.

During the period of pre-stripping, the complement will average 281 full time equivalent personnel (FTE), then increase to 383 during the initial years of commercial production while the concentrator is operating at 52.5 kt/d. After the concentrator is expanded to 105 kt/d, increases in mining rate and the length of haul result in average labour strength rising 502 FTE and reaching a peak of 602 FTE. Following closure of the Main Pit and during mining of Phase 8, the complement averages 139 FTE. When activity is solely focused on stockpile reclamation, the complement decreases further, to 84 FTE.

Figure 16-39: Labour Complement



Source: RNC.

17 RECOVERY METHODS

17.1 General

The process plant and associated service facilities will process ROM ore delivered to primary crushers to produce nickel concentrate and tailings. The proposed process encompasses:

- Crushing and grinding of the ROM ore;
- Desliming via hydrocycloning;
- Slimes rougher and cleaning flotation;
- Nickel sulphide rougher, scavenger and cleaning flotation;
- Magnetic recovery of sulphide rougher tailings and sulphide cleaner tailings;
- Regrinding of magnetic concentrate; and
- 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 processing facilities.

The magnetic separation tailings and awaruite rougher tailings will be combined in the coarse tailings thickener. The majority of the thickened coarse tailings will be sent to the TSF, while a small portion will be mixed with the slimes flotation tailings to help settle the material in the slimes tailings thickener. The thickened slimes tailings will also be sent to the TSF in a dedicated pipeline.

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 Plant Design Basis

The key criteria selected for the plant design are:

- Nominal base plant treatment rate of 52.5 kt/d;
- Nominal expansion plant treatment rate of 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.

17.3 Design Criteria Summary

The overall 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.

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
Mill Throughput		Mt/a	19.2	38.3
Mill/Flotation Availability		%	92	92
Mill Throughput		t/h	2,378	4,755
Physical Characteristics	BWi	kWh/t	21.0	21.3
(Design Values)	RWi	kWh/t	15.6	15.6
	CWi	kWh/t	15.3	15.3
	SMC	kWh/m³	5.33	5.33
	JK Axb	-	54.2	50.4
	Specific Gravity	t/m³	2.57	2.57
Grind Size	P_{80}	µm	180	180
Head Grade (Design)		% Ni	0.37	0.37
		% S	0.22	0.22
		% Magnetite	4.37	4.37
Metal Recovery (Design Values)	Overall Nickel	%	60.5	60.5
Ni Concentrate Filtration Rate		kg/m²/h	450	450
Concentrates Thickening Flux		t/m²/h	0.25	0.25
Tailings Thickening Flux	Slimes	kg/m²/h	0.5	0.5
	Coarse	kg/m²/h	1.0	1.0
Tailings Thickener Underflow Density	Slimes	% w/w	35	35
	Coarse		55	55
KAX20 Consumption		g/t	98	98
MIBC Consumption		g/t	112	112
Cytec 65 (Frothing Agent) Consumption		g/t	2	2
Calgon Consumption		g/t	254	254
CMC Consumption		g/t	22	22
Sulphuric Acid Consumption (H_2SO_4)		g/t	3,888	3,888
Flocculant Consumption	Concentrate	g/t	10	10
	Slimes	g/t	60	60
	Coarse	g/t	30	30
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 this application. Ausenco has sized this circuit to be suitable to achieve a throughput of 52.5 kt/d per plant 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 test work performed at SGS-Lakefield. Test work 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 75th 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 phase 2 plant design will need to be revisited once phase one is in operation 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 test work. Figure 17-1 shows a process schematic for the Dumont plant (only 52.5 kt/d plant shown).

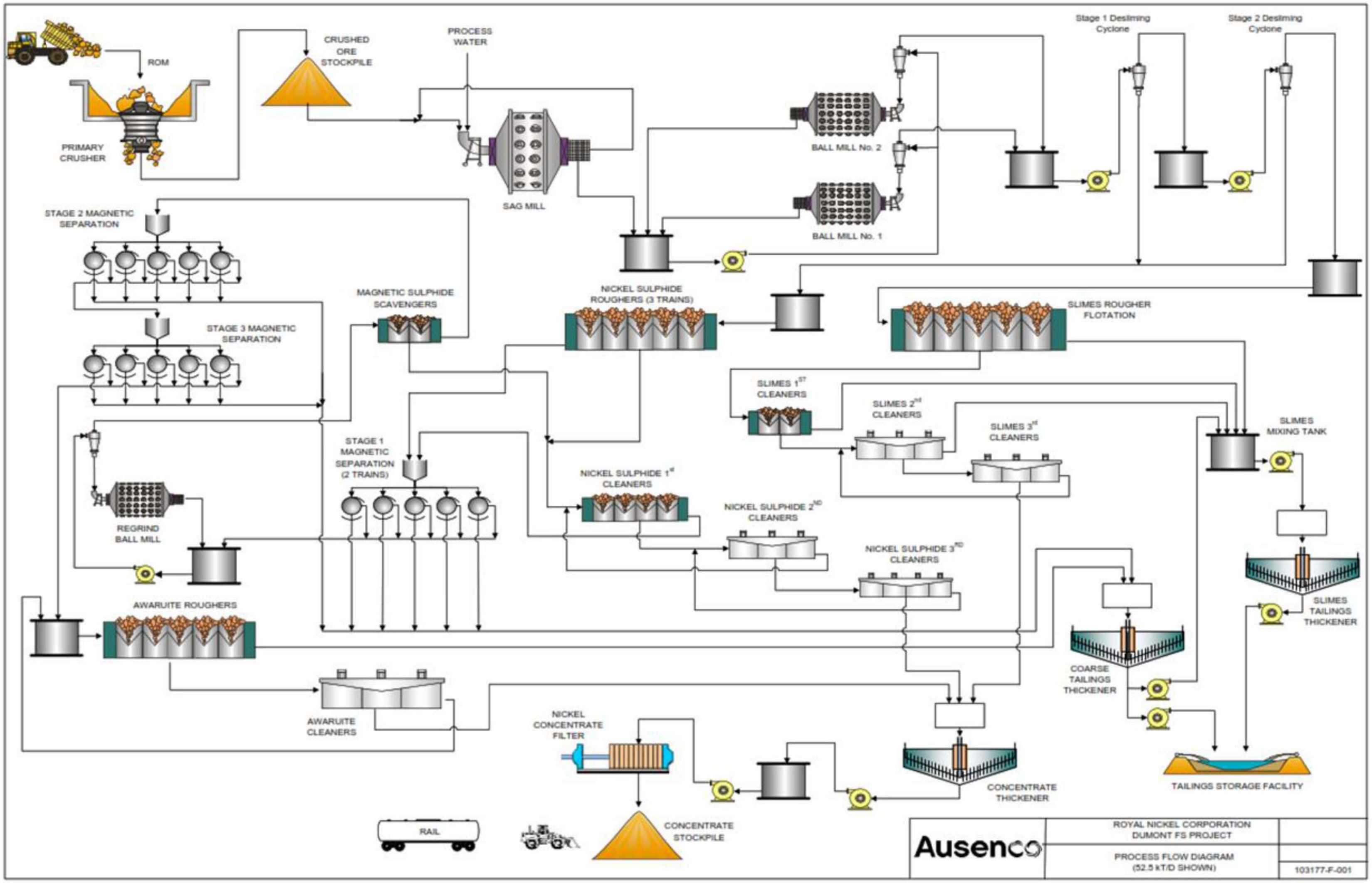
Details of the flowsheet design and the selection of major equipment for the process plants are discussed in the sections below.

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 sacrificial conveyor, which then feeds 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 22 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 12.3 m EGL (40.5 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



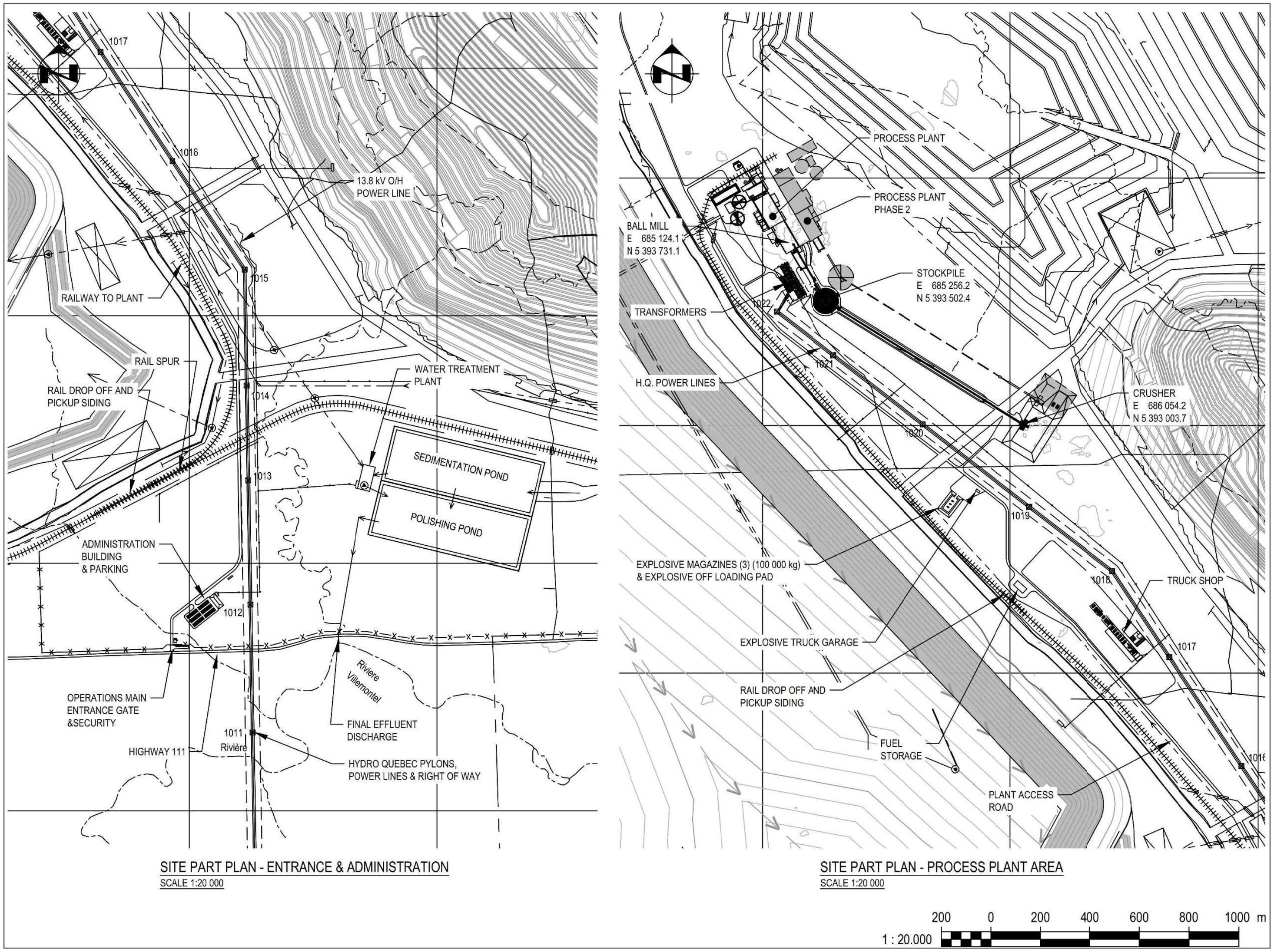
Source: Ausenco.

- Two-stage desliming circuit via hydrocyclones, targeting an overall mass split to slimes of about 20%, with the first stage to split mass according to an overflow particle size (P_{80}) of approximately 35 μm. Second stage to split mass to obtain an overflow with a P_{80} of 12 μm. The 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 six 5 m^3 forced air tank flotation cells to provide 45 minutes, 13 minutes, and 10 minutes of retention time, respectively.
- Magnetic separation (1st stage) 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 12% of sulphide rougher and cleaner flotation on plant feed.
- Magnetic concentrate regrind stage in an 8 MW ball mill, 6.7 m diameter (22.0 ft) with 9.6 m EGL (32 ft), operating in closed circuit with hydrocyclones, grinding to a product size of nominally 80% passing (P_{80}) of 41 µm.
- Magnetic sulphide scavenger flotation of the reground ore consisting of six 200 m^3 forced air tank flotation cells to provide 62 minutes of retention time.
- Two stages of magnetic separation (2nd and 3rd stage) on magnetic sulphide scavenger flotation tailings, consisting of five 3.6 m long LIMS magnetic separators for the 2nd stage and an additional five 3.6 m long LIMS magnetic separators for the 3rd stage, for a nominal stage mass recovery of 5.4% on plant feed.
- Awaruite rougher flotation consisting of five 70 m^3 forced air tank flotation cells per train to provide 70 minutes of retention time.
- Awaruite cleaner flotation consisting of four 1.5 m^3 forced air tank flotation cells to provide 21 minutes of retention time.
- Nickel concentrate (the combination of the slimes cleaner concentrate, the nickel sulphide cleaner concentrate and the awaruite cleaner concentrate) thickening in a 14 m diameter high-rate thickener followed by dewatering in a horizontal recessed-plate diaphragm pressure filter.
- Thickening of the combined magnetic separation tailings and awaruite rougher tailings in a 55 m diameter high rate thickener to an underflow density of 55% solids.
- Thickening of the slimes tailings, dosed with a small portion of the thickened coarse tailings to improve settling properties, in a 55 m diameter high rate thickener to an underflow density of 35% solids.
- TSF for process tailings deposition that will impound tailings for the first 19 years of operation. Thickened coarse tailings and thickened slimes tailings are fed to the TSF using dedicated pipelines.
- Reagent mixing facilities for KAX20 (collector), Calgon (depressant), CMC (depressant) and both concentrate and tailings flocculants.
- 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 thickener overflows and tailings storage facility. Other sources include pit de-watering operations.
- Potable water is generated by treatment water from the fresh water 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, filtered using sand filters, distribution services to supply cooling water, gland water, reagent mixing water, firewater, etc.
- Plant, instrument and flotation air services and associated infrastructure.

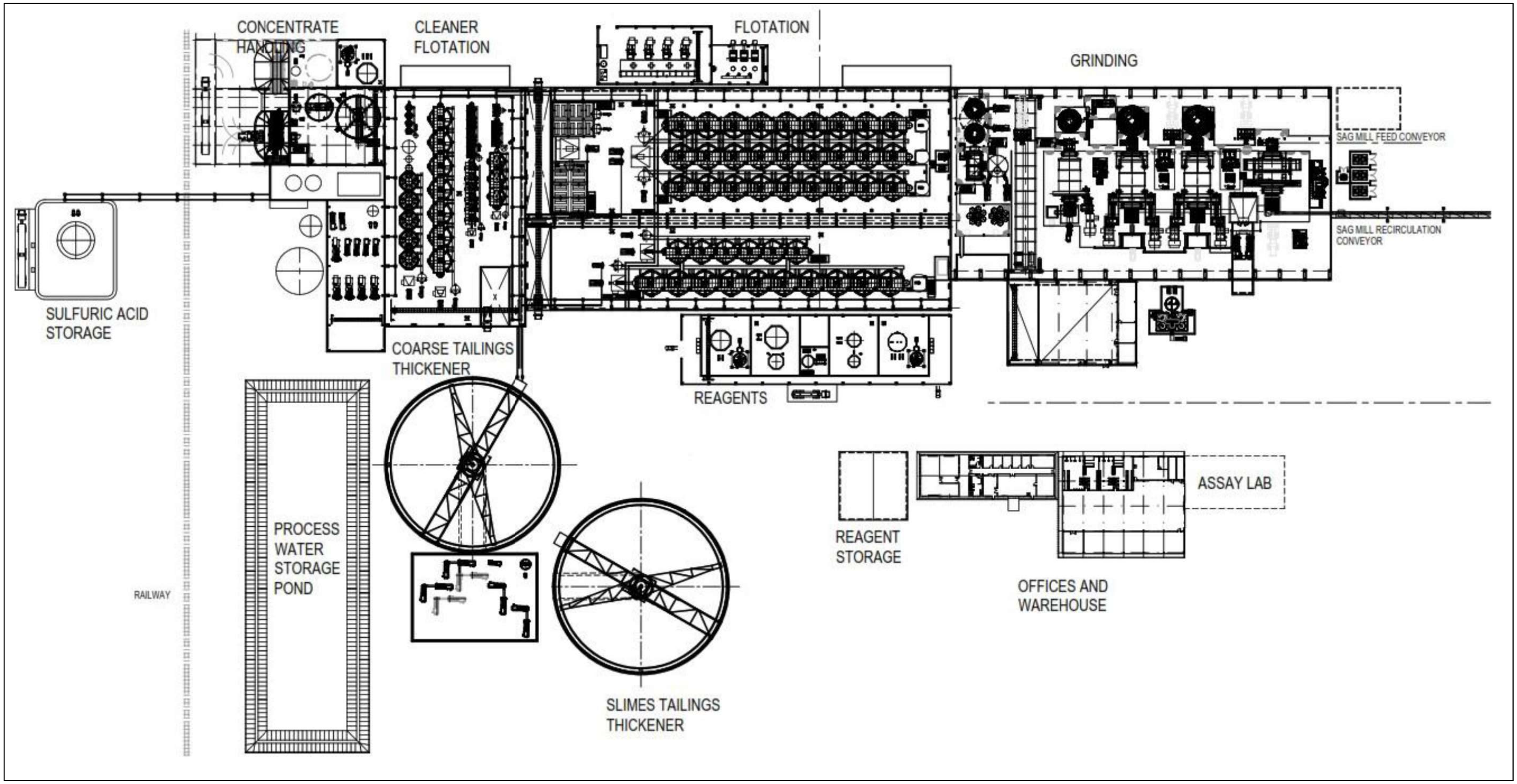
Layouts of the process plant area and process plant are shown in Figure 17-2 and Figure 17-3, respectively.

Figure 17-2: Layout of Process Plant Area



Source: 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 1600 x 2900 TS 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 323 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. An overhead crane will be installed in the building for maintenance of the crusher. Auxiliary crusher equipment includes a mobile rock breaker and dust suppression system. Water sprays are used to minimise dust at the crusher bin, crusher discharge and crusher belt feeder.

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 the 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. A cross belt self-cleaning electromagnet, followed by a metal detector, is fitted over the sacrificial conveyor to detect and remove tramp steel prior to discharge onto the stockpile feed conveyor.

The 98 m diameter and 37 m high stockpile will provide a minimum of 12 h live capacity at the nominal SAG mill fresh 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 150 kt. In the 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 nominal mill feed rate.

The stockpile will be enclosed to minimize fugitive dust emissions. The cover will be a dome of galvanized structural steel construction and cladding.

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 75th 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 test work 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 75th percentile for all design data. This would indicate that the current circuit could achieve higher throughputs when processing less competent ore. However, the selection of a design percentile that is higher than the average is necessary to allow the required throughput to be achieved on an annualised basis, since other non-comminution criteria will also restrict plant throughput, particularly on softer ores.

Flotation test work 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. This P_{80} has been selected because it is typical for sulphides to be found in finer size fractions under plant conditions when compared to laboratory test work (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 test work. 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 test work 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 22,000 kW.

Table 17-2: Mill Design Criteria

Criteria		Units	Design 52.5 kt/d	Design 105 kt/d
Throughput (nominal)		t/h	2,378	4,755
Mill Type			SAG	SAG
Shell Power Required		kW	17,357	34,714
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	28	28
	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	22,000	22,000
Mill Type			Ball	Ball
Grind Size	P_{80}	µm	180	180
Pinion Power Required		kW	23,064	46,128
Number of Mills			2	4
Pinion Power Required per mill		kW	11,532	11,532
Mill Speed		% Nc	75	75
Ball Charge Volume	Nominal	% vol	26	26
	Maximum (design)	% vol	35	35
Mill Diameter	Inside shell	m	7.9	7.9
Mill Length	EGL	m	12.3	12.3
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. Oversized pebbles from the trommel screen (scats) will be recycled back onto the mill feed conveyor. A cross belt self-cleaning electromagnet removes broken and worn mill balls and other tramp steel from the scats stream. 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.

A pebble circulating load of 15 % of the fresh 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 fresh feed.

The SAG mill 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.

Three 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 200%. 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 feed pumps and cyclone clusters, the SAG mill discharge slurry first passes through a trommel screen with 12 mm x 55 mm slotted apertures to remove pebbles and grinding media; the undersize flows into the hydrocyclone feed hopper.

SAG and ball mill discharge slurries will be combined in the ball mill hydrocyclone feed hopper and then pumped to two clusters of 650 mm diameter hydrocyclones to a target overflow P_{80} of 180 µm. Each hydrocyclone cluster will be fed with a dedicated hydrocyclone feed pump.

The milling circuit will require the installation of two clusters of fourteen 650 mm hydrocyclones per cluster, of which twelve will be in operation with two on standby. A slurry knife gate 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.

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 be combined 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 will deslime the hydrocyclone overflow from ball mill No. 1 and No. 2 to remove fine fibrous particles. This is critical to achieve optimal flotation kinetics in the sulphide flotation circuit. The deslime circuit accomplishes this with two-stage hydrocyclone clusters. The two-stage desliming circuit targets an overall mass split to slimes of about 20%, with the first stage to split mass according to an overflow particle size (P_{80}) of approximately 35 µm. The second stage to split mass to obtain an overflow with a P_{80} of 12 µm. The hydrocyclone sizes for each stage are 400 and 100 mm.

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 desliming stage.

The underflows from both the first and second stages 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 test work 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 180 µm. However, the use of magnetic recovery stages and regrind has shown that additional nickel sulphide recovery is achievable.

The magnetic recovery stage has the main purpose of recovering nickel that is in various alloy forms, predominately awaruite, a naturally occurring nickel/iron alloy. A subsequent regrind stage and magnetic recovery is required to increase liberation of the nickel and allow for higher rates of gangue rejection. The addition of a flotation stage on the regrind circuit product allows for recovery of additional nickel sulphides and allows 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 and magnetic sulphide scavenging;
- Secondary and tertiary magnetic separation; and
- Awaruite rougher and cleaner flotation.

Slimes, sulphide and awaruite concentrates are all combined as final concentrate. Slime rougher and cleaner flotation tailings report to the slimes tailings mixing tank. Combined magnetic tailings and awaruite rougher tailings report to a common coarse tailings thickener. The residence times for the nickel flotation circuit have been based on the test work performed on various Dumont ore types and composite ore type samples.

The test work 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	3.3	33
Slimes 2nd Cleaners	4	3.3	13
Slimes 3rd Cleaners	3	3.3	10
Sulphide Roughers	30	3.0	90
Sulphide 1st Cleaners	15	3.0	45
Sulphide 2nd Cleaners	4	3.3	13
Sulphide 3rd Cleaners	3	3.3	10
Magnetic Sulphide	20	3.1	62
Awaruite Roughers	20	3.5	70
Awaruite Cleaners	6	3.5	21

17.11 Flotation Circuit Configuration

17.11.1 Slimes Flotation

Stage 2 deslime hydrocyclone overflow will be fed by gravity to the slimes rougher conditioning tank, where flotation reagents will be added. The slimes flotation circuit consists of ten 300 m^3 tank flotation cells. The cells will be in a single cell arrangement with an elevation change between each cell. 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 slimes tailings mixing tank.

The slimes 1st cleaner stage consists of three 70 m^3 tank flotation cells, operating in an open circuit configuration. Slimes 1st cleaner flotation tailings are pumped to slimes tailings mixing tank and the concentrate is pumped to the slimes 2nd cleaner flotation stage.

The slimes 2nd cleaner stage consists of three 5 m^3 tank flotation cells. 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 tailings flow by gravity back to the 1st cleaners. 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 KAX20 (PAX, collector), MIBC and/or Cytec 65 (frother) and Calgon (depressant).

17.11.2 Sulphide Rougher Flotation

Stage 1 and 2 deslime hydrocyclone underflows are 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 single cell configuration with an elevation change and level control between each cell. 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 one of two nickel rougher concentrate hoppers. Concentrate from the 3rd train will have its own hopper which will pump the contents to 1st and 2nd train hopper. The combined concentrate will 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 KAX20 (PAX, collector), MIBC and/or Cytec 65 (frother) and Calgon (depressant).

17.11.3 1st Stage Magnetic Separation

A low-intensity magnetic separation (LIMS) 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 contained in magnetic alloys (primarily awaruite) that can be found in the sulphide rougher and cleaner tails.

Test work performed using magnetic separation established mass recovery and approximate concentrate nickel grade design criteria. 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

Magnetic Separation Stage Feeds	Magnet Strength Gauss	Magnet Linear Loading t/h/(m drum)	Magnet Volumetric Loading m^3/h/(m drum)	Magnet Configuration
1st Stage: Nickel Sulphide Rougher & 1st Cleaner Tailings	1,000	26	80	Counter-current
2nd and 3rd Stages: Magnetic Sulphide Scavenger Tailings	1,000	26	80	Counter-current

Nickel sulphide rougher and cleaner flotation tailings will be pumped to two trains, each consisting of seven 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 gravity flow to a common hopper and be pumped to the regrind circuit, specifically the regrind mill cyclone feed hopper.

The 1st stage magnetic tailings will gravity flow to a common hopper for all three stages of magnetic tailings (non-mags) which will be pumped to the coarse tailings thickener.

17.11.4 Magnetic Concentrate Regrind

A closed regrind circuit is used to grind the stage 1 magnetic concentrate stream. The regrind mill discharge first passes through a trommel screen to remove scats. The underflow is fed to the regrind mill cyclone feed hopper and then pumped to a cluster of seventeen (fifteen operating and two standby) 400 mm hydrocyclones to achieve a product size of nominally 80% passing (P_{80}) of 41 µm. Hydrocyclone overflow will report as flotation feed to magnetic sulphide scavenger flotation, while the underflow will report to the regrind ball mill for further grinding.

The regrind mill nominal fresh feed rate is 285 t/h. The mill was sized using a design BWI of 21.6 kWh/t. The regrind circuit classification circuit has been designed for a nominal circulating load of 400 %. On this basis, a single 8,000 kW regrind ball mill has been selected for this study.

17.11.5 Magnetic Sulphide Scavenger Flotation

The magnetic sulphide scavenger flotation stage consists of six 200 m^3 tank flotation cells arranged in a single train. The cells will be in a single cell configuration with an elevation change and level control between each cell. Magnetic sulphide scavenger tailings will be pumped to stage 2 magnetic separation, while recovered nickel sulphide concentrate will be pumped to the nickel sulphide cleaner flotation circuit.

The reagents added will consist of a combination of KAX20 (PAX, collector), MIBC and/or Cytec 65 (frother) and Calgon (depressant).

17.11.6 2nd and 3rd Stage Magnetic Separation

Two stages of low-intensity magnetic separation cleaning circuits are used to treat the tailings of the magnetic sulphide scavenger flotation. This material is the 1st stage magnetic separation circuit concentrate which is first passed through the regrind mill and the sulphide scavenging flotation stage. The function of this circuit is to recover nickel contained in magnetic minerals (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.

The magnetic sulphide scavenger flotation tailings will be first pumped to a train of five single-drum magnetic separators (3.6 m long and 1.2 m diameter) via a feed distributor (2nd stage). The recovered concentrate is then pumped to a second train of five single-drum magnetic separators (3.6 m long and 1.2 m diameter) via a feed distributor (3rd stage). The number of magnetic separators selected will allow for variations in throughput and magnetic mass recoveries.

Magnetic 3rd stage concentrate will be pumped to the awaruite flotation circuit. Magnetic separation tailings (non-mags) from both stages will gravity flow to a common hopper for all three stages of magnetic separation tailings. The combined slurry will be pumped to the coarse tailings thickener.

17.11.7 Nickel Sulphide Cleaner Flotation

Three different 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 regrind circuit described above, 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 consists of six 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 reagents added will consist of a combination of CMC (depressant), KAX 20 (PAX, collector), MIBC (frother) and Calgon (depressant).

17.11.8 Awaruite Flotation

Stage 3 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 coarse tailings thickener). and a single cleaner flotation stage (awaruite cleaner tailings are recirculated to the head of the awaruite rougher stage).

Before the awaruite flotation circuit, the feed is conditioned in a series of tanks, the slurry then overflows to a second conditioning tank, where reagent addition takes place.

The reagents added will consist of a combination of KAX 20 (PAX, collector), MIBC and/or Cytec 65 (frother) and Sulfuric Acid (pH modifier).

The rougher flotation stage consists of five 70 m^3 tank flotation cells. The awaruite cleaner flotation stage consists of four 1.5 m^3 flotation cells.

17.12 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. A settling rate of 0.25 $t/m^2/h$ has been selected as the basis of design for nickel concentrate thickener based on the test work results.

The concentrate storage tank prior to filtration will have a live volume of 720 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 selected is a horizontal recessed-plate diaphragm pressure filter. A sample of concentrate from the pilot plant was tested and the results used as the design basis.

Filter cake is stockpiled and loaded onto rail cars via a front-end loader.

Expansion to a plant throughput of 105 kt/d will require an additional concentrate storage tank and horizontal recessed-plate diaphragm pressure filter.

17.13 Tailings Disposal

The design basis chosen for this level of study includes a coarse tailings thickener, a slimes tailings mixing tank and a slimes tailings thickener. Disposal of thickened tailings from each thickener will be fed to the TSF in dedicated pipelines. Water will be recovered from the TSF surface and recycled to the plant as reclaim water.

The coarse tailings thickener design has been sized on a settling rate of 1.0 $t/m^2/h$ and results in the selection of a 55 m high rate thickener. The feed slurry to this thickener consists of combined magnetic separation tailings and awaruite rougher tailings pumped to the thickener. The slurry will be thickened to a target density of 55% w/w solids.

The coarse tailings thickener underflow will be split into two streams. One pump will send a small portion, approximately 10%w/w, to the slimes tailings mixing tank. The majority of the underflow will be sent to the TSF. There will be two sets of two pumps in series for the first year of operation and will increase to two sets of four pumps in series for the subsequent years of operation (total of 8). The phase 2 coarse tailings thickener will require all 8 pumps at start-up. The thickened coarse tailings slurry will be pumped to the TSF via a 7.5 km pipeline pipe. Process water from the coarse thickener overflow will flow by gravity to the plant process water storage pond.

Slimes rougher and cleaner flotation tailings will be combined with a small portion of the thickened coarse tailings in the slimes tailings mixing tank to promote settling in the slimes thickener. The mixed slimes will be pumped to the slimes tailings thickener, which has been sized using a settling rate of 0.5 $t/m^2/h$ and results in the selection of a 55 m high rate thickener. The slurry will be thickened to a target density of 35% w/w solids. Process water from the slimes thickener overflow will flow by gravity to the plant process water storage pond.

For phase 2, additional pipelines will be required for both coarse and slimes tailings.

Reclaim water from the TSF will be pumped back to the process water pond via barge pumps, and a 4 km HDPE water pipeline.

17.14 On-Stream Analysis

The on-stream analysis (OSA) system will provide online nickel and other supporting assay analysis on the following 20 streams:

- Combined ball mill hydrocyclone overflow;
- Stage 2 deslime cyclone overflow;
- Sulphide rougher flotation feed;
- Slimes rougher flotation tailings;
- Slimes cleaner flotation concentrate;
- Slimes cleaner flotation tailings;
- Nickel sulphide rougher flotation concentrate;
- Nickel sulphide rougher flotation tailings (three streams, one per train);
- Nickel sulphide 1st cleaner flotation tailings;
- Nickel sulphide 3rd cleaner flotation concentrate;
- Magnetic sulphide scavenger flotation concentrate;
- Stage 3 magnetic separator concentrate;
- Combined magnetic separator tailings (non-mags);
- Awaruite rougher flotation tailings;
- Awaruite cleaner flotation concentrate;
- Final combined concentrate thickener feed;
- Final slimes tailings thickener feed; and
- Final coarse tailings thickener feed.

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, the concentrate filter feed box, or the awaruite rougher conditioning tank.

17.15 Sampling

Several samplers are provided throughout the plant to generate on-stream analysis and composite shift samples from key process streams. Two types of sampling will be performed; metallurgical and process control sampling.

Metallurgical samplers will be used to generate shift composite samples that will be assayed for plant metallurgical accounting. The following process streams are equipped with metallurgical samplers:

- Combined ball mill hydrocyclone overflow;
- Final combined concentrate thickener feed;
- Final slimes tailings thickener feed; and
- Final coarse tailings thickener feed.

These four samplers will allow the feed and final products to be sampled, to allow an accurate metal balance of the plant to be completed. Metallurgical samplers will be either cross stream or static riffler samplers. Samplers will consist of multiple stages to sub-sample the streams to produce a statistically accurate sample for metallurgical accounting purposes. The target mass recovery from the final stage of sampling is 15 L of slurry for transport to the sample preparation and assay laboratory (by others). Typically, a process control sample will also be generated from the penultimate sampling stage.

Process control samplers will typically be either pressure pipe or launder samplers and will be used to generate samples for process control of the plant. These samples will be used to feed the on-stream analyser to generate near real time assay data to allow operations to optimise the flotation circuit operation. The on-stream analyser will also sub sample the control sample stream to generate shift composite samples on these other process streams that are not required for metallurgical accounting, but that will provide valuable insight into plant operations. The following process streams are equipped with pipe or launder samplers:

- Stage 2 deslime cyclone overflow;
- Sulphide rougher flotation feed from feed distributor;
- Slimes rougher flotation tailings;
- Slimes cleaner flotation tailings;
- Nickel sulphide rougher flotation concentrate;
- Nickel sulphide rougher flotation tailings (three streams, one per train);
- Nickel sulphide 1st cleaner flotation tailings;
- Nickel sulphide 3rd cleaner flotation concentrate;
- Magnetic sulphide scavenger flotation concentrate;
- Stage 3 magnetic separator concentrate;
- Combined magnetic separator tailings (non-mags);
- Awaruite rougher flotation tailings;

17.16 Reagents

Reagents for the project are listed below.

Collector – Potassium Amyl Xanthate (KAX20 (PAX**) –** Potassium Amyl Xanthate is a sulphide mineral collector and will be supplied in 1000 kg bulk bags as a dry reagent. KAX20 will be shipped by road to site and offloaded by forklift. KAX20 will be stored in the reagent's storage area of the warehouse facility and delivered to the KAX20 mixing area. KAX20 bulk bags will be lifted by the common reagents area overhead crane 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 KAX20 storage tank by way of pump. The KAX20 solution is stored in a day tank, where it is reticulated around the plant in a ring main system using the 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 dedicated metering pumps.

Frother 2 – Cytec 65 – Cytec 65 is a trademarked frother that will be supplied in drums 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 dedicated metering 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 the common reagents area overhead crane 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 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 the common reagents area overhead crane 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 0.5 % 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 dedicated metering 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. The storage tank will have capacity for 125 hours of consumption at design flow rates. The sulphuric acid will be distributed to the flotation circuit dosing points by multiple centrifugal pumps.

Flocculant – Magnafloc 342 – A flocculant mixing, storage and dosing system located in the reagent preparation area will be provided to facilitate concentrate thickening. Magnafloc 342 will be supplied in 25 kg bulk bags and will be shipped as a dry reagent. The flocculant will be manually loaded into the concentrate flocculant storage hopper and fed via screw feeder to the concentrate flocculant mixing tank where it is diluted to 0.25 % w/w. The diluted mix is transferred to the concentrate flocculant storage tank by way of a pump. 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.

Flocculant – 913 VHM – A flocculant mixing, storage and dosing system located in the reagent preparation area will be provided to facilitate coarse tailings thickening. 913 VHM will be supplied in 750 kg bulk bags and will be shipped as a dry reagent. The bulk bags will be lifted by the common reagents area overhead crane and loaded into the coarse tailings storage hopper. Loose flocculant is transported via screw feeder to the coarse 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 coarse tailings flocculant storage tank by way of pump. 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 coarse tailings thickener.

Flocculant – Magnafloc 333 – A flocculant mixing, storage and dosing system located in the reagent preparation area will be provided to facilitate slimes tailings thickening. Magnafloc 333 will be supplied in 750 kg bulk bags and will be shipped as a dry reagent. The bulk bags will be lifted by the common reagents area overhead crane and loaded into the slimes tailings storage hopper. Loose flocculant is transported via screw feeder to the slimes flocculant mixing tank. Water is added to the slimes tailings 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 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 slimes 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.

17.17 Air Services

17.17.1 Process Air

The flotation blowers will supply low pressure process air to the flotation cells. The blowers will generate air at the highest pressure required by the flotation cells. Pressure reducers will be used to step-down the pressure to the flotation cells requiring lower 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. 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.

A fourth air compressor is dedicated to the concentrate filter press requirements. It is equipped with its own dedicated air dryer and receivers.

Valving will allow the standby plant compressor to be used as a standby filter compressor as required.

17.18 Process Control Philosophy

The control philosophy to be implemented for the Dumont Nickel and Cobalt 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 of the process plant is located 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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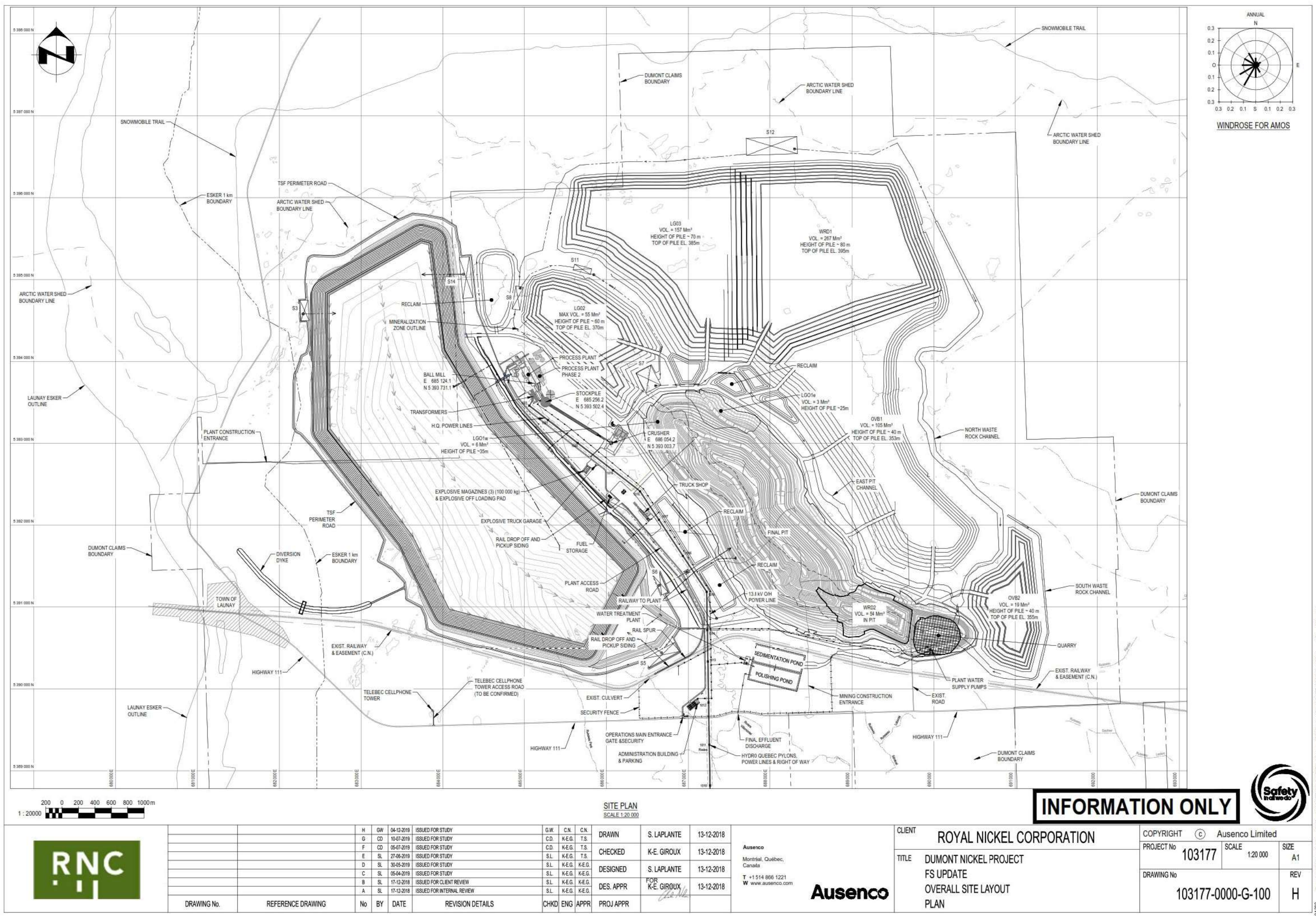
Figure 18-1: Overall Site Layout



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 Propane Gas

The use of Propane gas is considered for heating buildings (Process plant and workshops) in this study and is included as part of the OPEX, as deliveries will be by tanker truck.

For future supply considerations, an existing natural gas pipeline extends to within approximately 25 km from the south edge of the property. The tie-in and construction costs to supply gas from this location and units conversion cost have not been included.

18.4 Rail Spur

A rail spur that services the process plant is proposed for the project. The total length of the rail spur is 6.0 km. A fuel tanker drop-off and pickup siding is located beside the fuel storage area, near the mining truck shop, and the main track extends north of the process plant to load concentrate. A rail car drop-off and pickup siding is located north of the main security entrance, northwest of the water treatment plant for dropping off and picking up consumable rail cars and concentrate.

18.5 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 expit operation, when stockpiles are being reclaimed) to provide aggregate for continually resurfacing haul roads.

18.6 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 located 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 rougher and cleaner 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 two 55 m diameter coarse and slimes tailings thickeners are adjacent to the process water pond, on the south side.

During the plant expansion in Year 7 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 thickeners (coarse and slimes) 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.

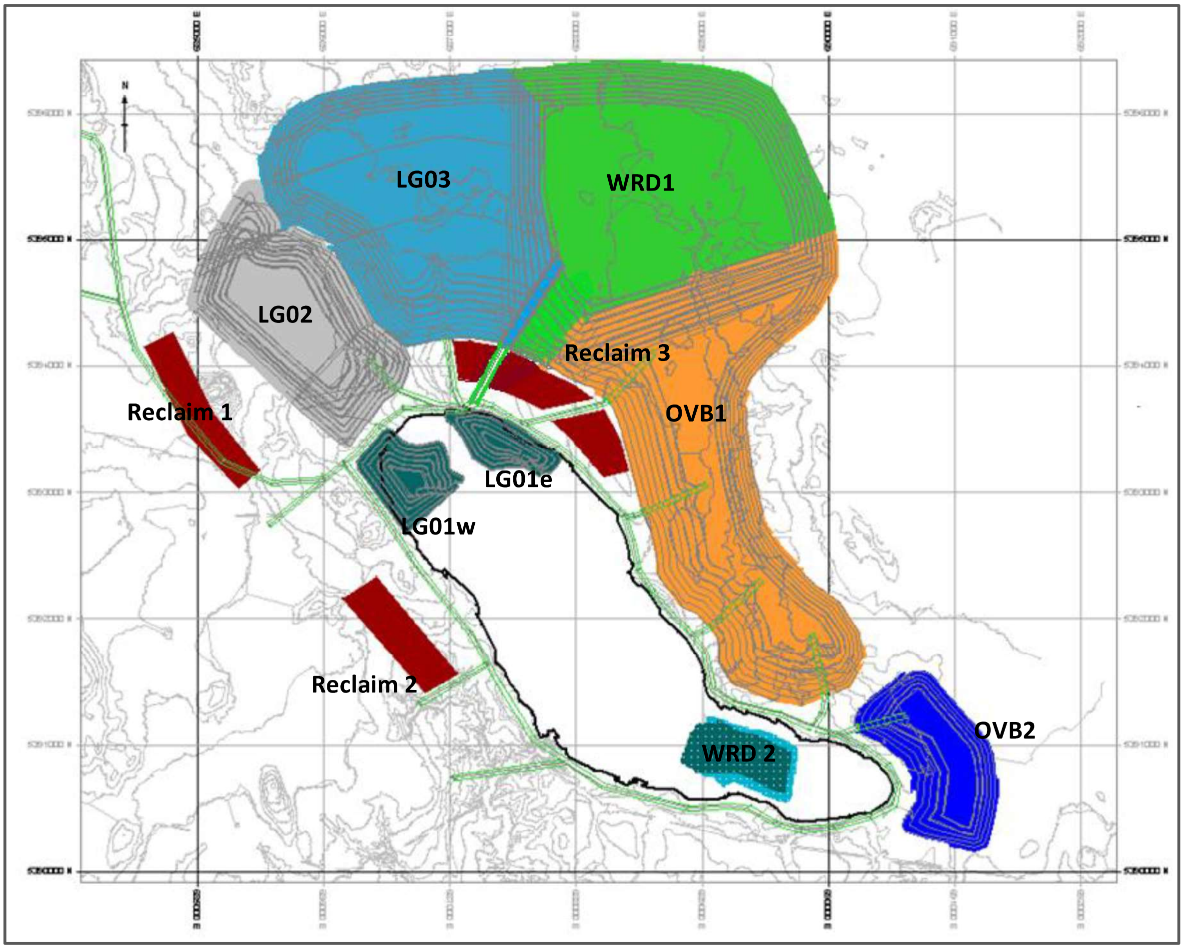
18.7 Waste Rock & Overburden Dumps, Low-Grade Ore & Reclaim Stockpiles

The open pit mining operation will generate 1,052 Mt of overburden and waste rock and 511 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 such as the tailings storage facility and mine roads. The balance will be stored in two waste rock dumps (WRD1 and WRD2) and co-disposed with overburden in OVB1. Overburden from the open pit stripping will be used for reclamation, where applicable, with the balance stored in two overburden dumps (OVB1 and OVB2). Low-grade ore will be intermittently processed or stored in three low-grade ore stockpiles (LGO1, LGO2, LGO3).

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 18-2 shows the location of the various impoundments. Note that the impoundments never co-exist as presented in the figure – for example, LGO1 will be completely reclaimed before tipping on WRD2 commences.

Figure 18-2: Dumont Open Pit Impoundments of Waste, Reclamation Material and Low Grade Ore



18.7.1 Waste Rock Dumps

The total waste rock volume, which includes a compacted swell factor of 32%, is expected to be 417 Mm^3. This material will be impounded as follows:

- 68 Mm^3 will be used for construction of roads and the TSF dike along with backfilling of the TSF key trench
- 28 Mm^3 will be co-disposed with overburden in OVB1. This dump will reach a maximum height of 40 m, being constructed in 5 initial lifts of 5 m followed by 2 of 10m.
- 54 Mm^3 will be impounded in the inpit waste dump WRD2. The initial 39 Mm^3 will come from Phase 7 of the pit, and will lift the dump to RL 1000 m, or 1 m lower than the lowest point on the pit perimeter and will therefore not impede drainage from the north following pit closure. The remaining rock will be sourced from Phase 8, following cessation of activities in the Main Pit, and will extend the dump to the north-west.
- 267 Mm^3 will be impounded in WRD1, which will reach a maximum height of 80m.

18.7.2 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 (OVB1) will reach a maximum height of 40 m high and will store 77 Mm^3 of mixed clay and S&G, along with the 28 Mm^3 of waste rock previously discussed. The southeastern pile (OVB2) will be 40 m high and will store 19 Mm^3 of predominantly S&G material.

A further 5 Mm^3 of overburden will be used as cover material during site reclamation and will be temporarily stored in three stockpiles (Reclaim1 – 3). These will each reach a maximum height of 10 m.

18.7.3 Low-Grade Ore Stockpiles

Three stockpiles, LGO1, LGO2, LGO3 will be developed to temporarily store low-grade ore on a regular basis during the period of active pit mining. The maximum volume of low-grade ore that will require storage is estimated to be 207 Mm^3.

LGO1 will be located within the final perimeter of the pit and will impound the highest value stockpiled ore during the early years of pit development. This stockpile will have been completed depleted before mining expands into the area following Year7. The stockpile will comprise two lobes (LGO1w and LGO1e), being divided by a small stream. The smaller eastern lobe will reach a maximum height of 30m and have a capacity of 3 Mm^3. This lobe will only be required for a 12 month period during Year3. The larger LGO1w will reach a maximum height of 40 m and have a capacity of 6 Mm^3. It will be the preferred location for impounding material due to its shorter haul to the primary crusher.

LGO2 will contain the next highest value stockpiled ore and will be located 211 m closer to the primary crusher than LGO1 (but will, on average, require a 765 m further haul from the pit). This stockpile will reach a maximum height of 60m and have a maximum capacity of 55 Mm^3. The stockpile will be depleted approximately 2 years before mining in Phase 8 is completed.

LGO3 will contain the lowest value stockpiled ore and reclamation will only begin when LGO3 has been depleted. The design provides for a maximum height of 70 m, allowing it to impound up to 168 Mm^3.

18.8 Tailings Storage Facility

The TSF will be used to impound tailings for the first 19 years of operation.

Starter dams, consisted of clay, rock and sand, will be required to be built in the first few years. Subsequent dam raises will be carried out mainly using coarse tailings, sand and rock, with rock as the principal material. The starter dam of the Northern TSF will be built first at Yr0, the southern starter dam will be built during Yr1. The objective of splitting the construction of the starter dams into 2 years is to reduce traffic and noise for the local population as there will be construction works for the others mine infrastructures on the East and South side during Yr0.

The starter dams of the TSF will provide storage during the first two years of operation for approximately 27.5 Mm^3 of tailings. Dam raises will be completed annually from Yr1 to approximately Yr11 based on the downstream construction method. From Years 12 to 19, the dams raise will be switched to an upstream construction method.

Deposition will progress from the north to the south, pushing the supernatant pond southward as the tailings advance. Two (2) lines of tailings (coarse and mixed) will be required. The coarse tailings will be deposited along the perimeter dam to form a tailings beach and provide construction material for the perimeter dams. While the mixed tailings will be deposited in the middle of the TSF. The purpose of the tailings separation is to provide a faster consolidation rate and quicker drainage of the higher permeability coarse tailings for:

- The earlier use of the tailings beaches as borrow source for construction;
- Ease of tailings transport by truck;

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- An increase in constructability by lowering the risk of a construction ‘bottleneck’ (i.e. not being able to raise vs tailings deposition).

During operations, the supernatant pond inside the TSF will be maintained as low as possible. The objectives are to minimize risk of overtopping during operations as well as to minimize impacts during a potential dam failure (in an unlikely event one should occur) as the tailings run-out and inundation area will be minimal, due to the minimal water volumes stored in the TSF.

18.8.1 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
Project Life Main Pit mining period Phase 8 and Stockpiles period	30 years 19 years (first of 30 years) 11 years (remainder of 30 years)	RNC
Tailings production Year 1 Year 2 to Year 6 Year 7 Year 8 to Year 30 Year 31	 45.7 kt/d 52.2 kt/d 71.8 kt/d 104.6 kt/d 20.9 kt/d	RNC
Required total TSF tailings storage Mass Volume	 596 Mt 458.5 Mm^3	RNC
Required inpit storage Mass Volume	 428 Mt 329.2 Mm^3	RNC
Dam Classification(1)	Variable between High and Very High(2)	SRK/WOOD
Maximum Design Earthquake	1:5000 year, PGA = 0.1g	SRK/WOOD
Freeboard above supernatant water pond	2.5 m(3)	SRK
Environmental Design Flood (EDF) (4)	Snow accumulation (1:100 years) melt in 30 days and 24-hour rainfall, (1:1000 years)	SRK/WOOD
Inflow Design Flood (IDF)	PMF	WOOD
Stability Factor of Safety (FOS) (5) Static, drained Static, undrained Pseudo-static	 1.5 1.5 1.1	SRK/WOOD
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 2013). 2. Dams at the TSF were designated with two classifications based on their corresponding failure consequences. 3. 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). 4. EDF is to be managed without release of untreated water to the environment). 5. The minimum factor of safety follows the Canadian Dam Safety Guidelines 2007 Edition (CDA 2013).

18.8.2 Site Selection

The selection of the 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.

18.8.3 Foundation Preparation beneath the Perimeter Dams

The geotechnical database for the TSF area indicates that the typical soils foundation of the TSF generally consisted of a thick layer of fine grained lacustrine clays, overlaying a sand and gravel layer of variable thickness. The clays are overlain by top soils approximately 1 meter thick. The grey clay in the lacustrine layer has a consistency from very soft to firm and is present along some sections of the proposed perimeter dams. Foundation preparation will consist of clearing the entire area. The clearing will include the stripping and stockpiling of organic soils. Some of the organic soils will be pushed upstream of the starter dams and will be used with rock to build, where required, a temporary stability berm upstream of the perimeter dams.

Stability analyses indicate that shear keys will be required in some locations as the dams are raised. The shear keys will consist of excavating the grey clay and replacing it with waste rock from the mine operations. The typical shear key section will be between 3 to 6 m deep, and 8 m wide across the base, with cut slopes at 1.5H:1V. The area of the shear key is shown in Figure 18-2. Material excavated from the shear key will placed inside the TSF or used for reclamation.

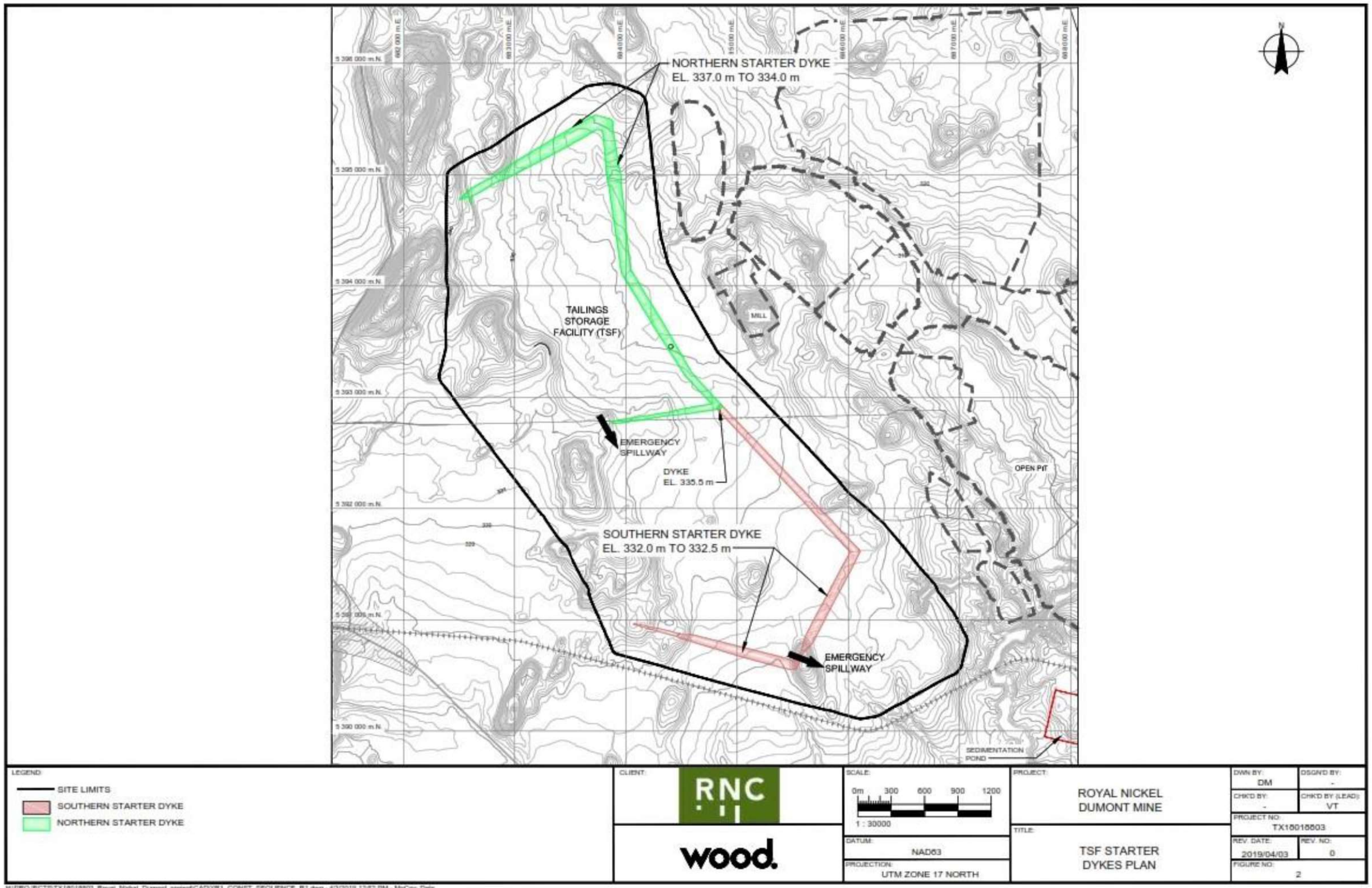
18.8.4 Starter Dam Design

A plan view of the TSF starter dam, to be built in Yr0 and Yr1, is shown in Figure 18-3.

The northern starter dam of the TSF will be constructed to a maximum elevation of 337m. Both the upstream and downstream slopes will be constructed to 3.5H:1V, with a 4 m wide, vertical clay core that will tie into the clay stratum and extended to the top of the starter dam. A filter zone will be constructed upstream and downstream of the clay core. Stability analysis have shown that a stability berm will be required to be built on both the upstream and downstream sides. Shear keys will be required for certain section of the starter dams where clay is thick.

The configuration of the southern starter dam will be similar to northern starter dam, with a clay core, filter zone, and shear key. The crest of the southern starter dam of the TSF will be between elevations 332 m and 332.5 m.

Figure 18-3: TSF Starter Dams Plan



Source: Wood.

18.8.5 Subsequent Dam Raises & Final Elevation

As mentioned in section 18.7.1, the dams will be raised annually mainly with tailings and waste rock using the downstream construction method from Yr1 through approximately Yr11. From Yr12 through Yr19, the dams will be raised with tailings using the upstream construction method. The required stability berm and shear keys will be progressively constructed as the dams are raised. The downstream slopes of the TSF dams will be constructed at 3.5H:1V and the perimeter dams are designed to promote seepage. Post TSF filling, there will be no ponding on top of the TSF and tailings will be mostly drained. This allows for the TSF to convert to a lower risk structure or possibly be reclassified to a mine waste landform from a tailings impoundment.

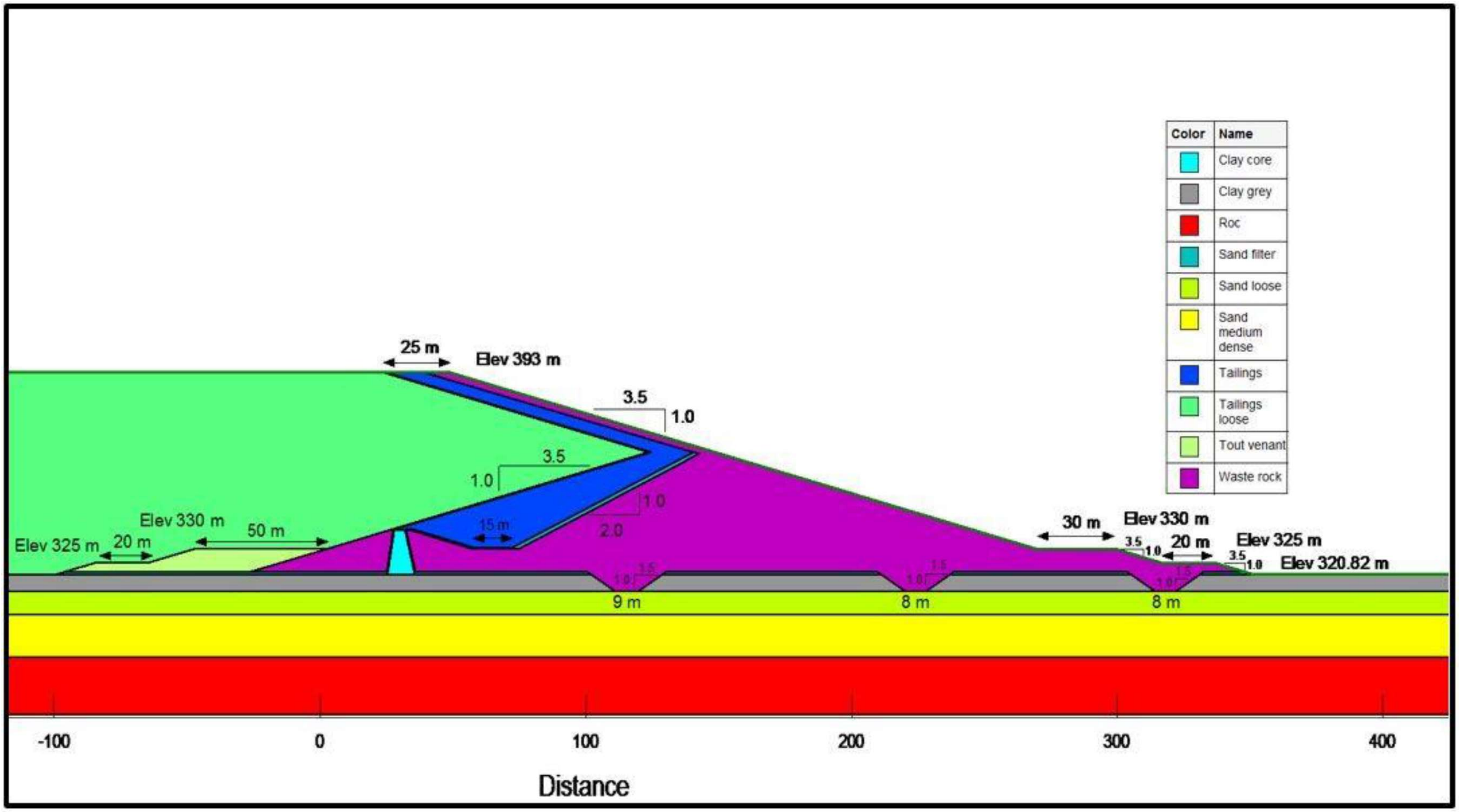
The final crest elevation of the perimeter dams will be 392 m. The total volume of material needed to construct the TSF, including perimeter dams, core, filter, stability berm, and shear key will be 95 Mm3. This does not include the 7.5 Mm3 of grubbing and shear key excavation required. Typical cross-sections through the TSF dams are presented in the Figure 18-4 to Figure 18-6. The typical cross-section through the Recycle Water Basin is presented in Figure 18-6

18.8.6 Inpit Tailings Disposal

18.8.6.1 General

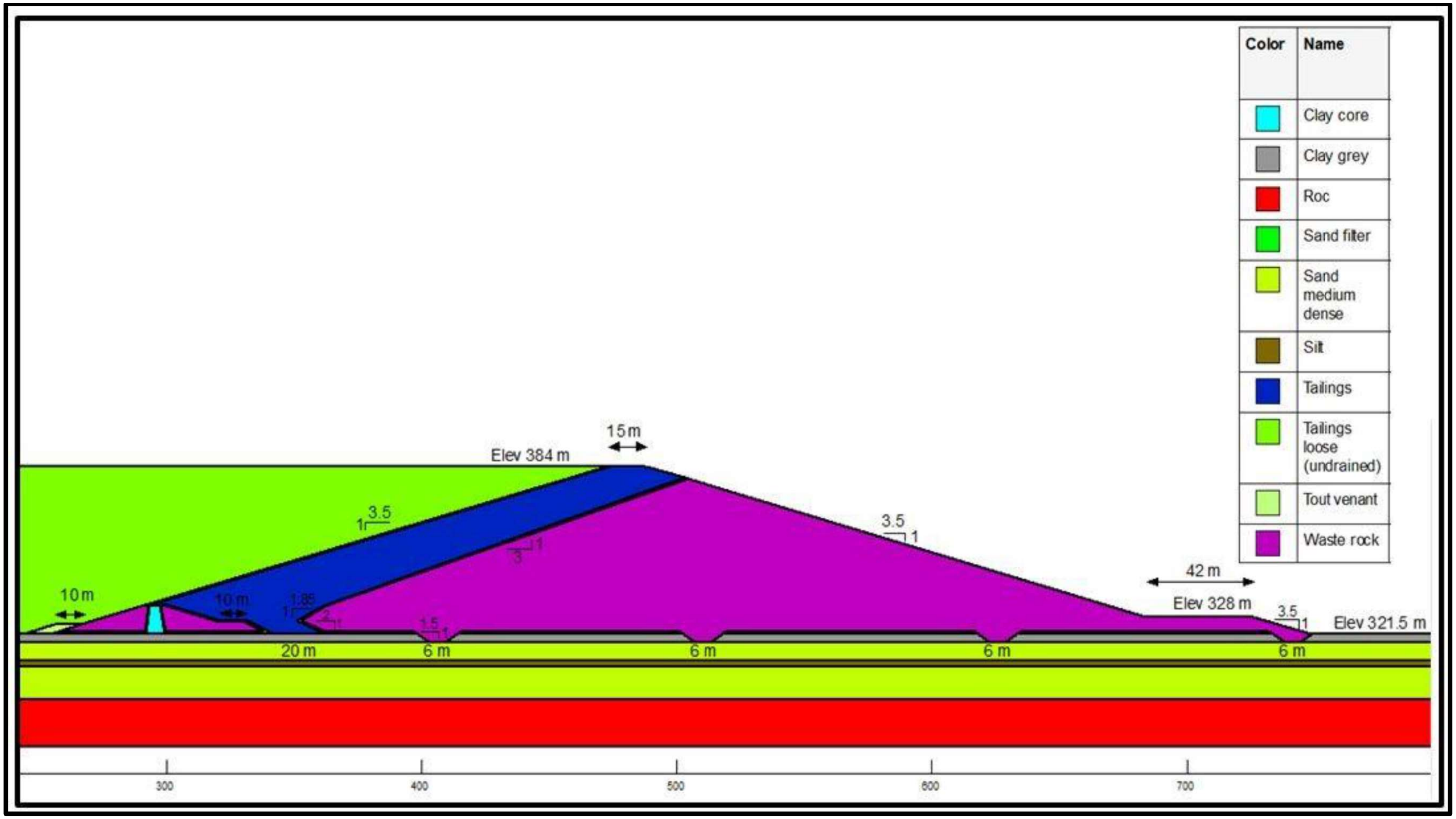
Once mining operation has been completed, the mill will continue to process the stockpiled ore for another 12 years. Approximately 428 Mt of tailings will then be deposited into the open pit for permanent storage.

Figure 18-4: Typical Cross-section through TSF Eastern Dam



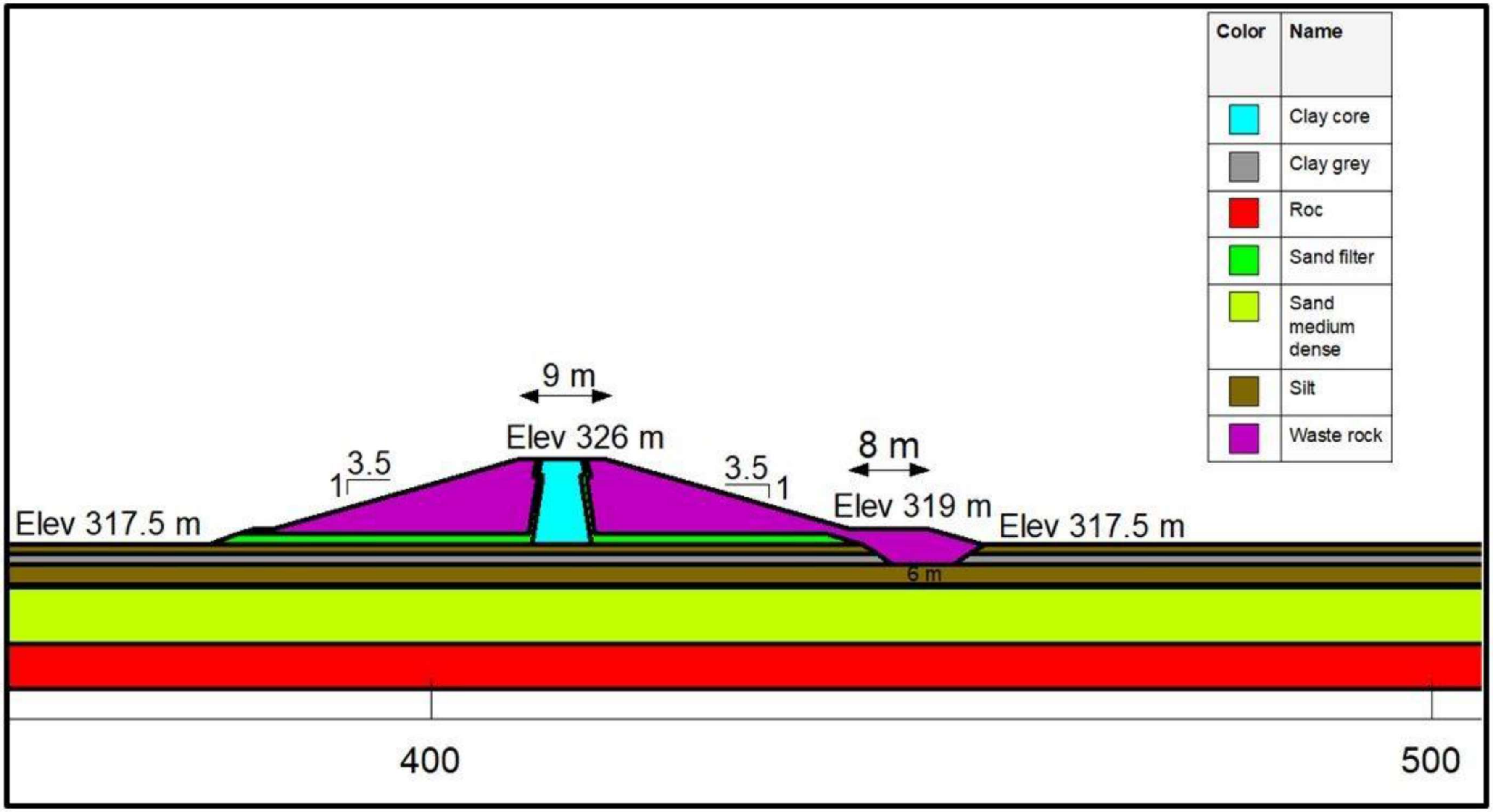
Source: Wood.

Figure 18-5: Typical Cross-section through TSF Southern Dam



Source: Wood.

Figure 18-6: Typical Cross-section through TMF Recycle Water Basin



Source: Wood.

18.8.6.2 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 Pit will then be allowed to fill with direct precipitation and runoff, and overflow into the Villemontel River. Figure 16-23 shows a typical section at end of milling (approximately Yr31). Although, if required due to water quality concerns, the Pit water will be pumped and treated prior to discharge to the river.

18.8.7 Water Management

18.8.7.1 General

The water management plan at the TSF is largely controlled by the following factors:

- The supernatant pond within the TSF will be separated from the perimeter dam by a tailings beach, except at the south-eastern dam were the water will be ponded against it. As mentioned in section 18.7.1, the supernatant pond will be maintained as low as possible to minimize risk of overtopping during operations as well as to minimize impacts in the unlikely event of a dam failure. Water from the supernatant pond will be transferred, likely via siphon system, to the recycle water basin (RWB), and will provide recycle water for the plant site.
- Seepage rates through the perimeter TSF dam are expected to be high as the perimeter dams are design to promote seepage.
- An emergency spillway will be constructed on both TSF and RWB, to safely convey a Probable Maximum Flood (PMF). For the TSF, the emergency spillway is required until year 10, and it will be located besides the siphon, discharging towards the RWB. From year 10, the minimum freeboard between maximum water level and dike lowest crest (at siphon location) will be higher than 1.3 m, increasing the TSF water storage capacity. The emergency spillway will then be replaced by additional siphons to evacuate the PMF. The emergency spillway of RWB will discharge towards the open pit.

18.8.7.2 Water Pool & Water Return

During operations, coarse tailings will be deposited around the perimeter of the dam using a conventional spigot method to develop a tailings beach to keep the supernatant pond away from the perimeter dams. The mixed tailings stream will be deposited in the centre of the TSF as mentioned in section 18.7.1.

Water for recycle will be obtained using a floating barge, pipeline and siphon system.

18.8.7.3 Seepage Collection

Within the footprint of the TSF, 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 porewater from seeping into the natural groundwater.

A series of ditches leading to four seepage collection sumps will be established around the external perimeter of the TSF. Pumps will be set up at each sump to convey this water back into the TSF, and thereby prevent seepage from potentially entering the environment. Typical, sumps are rectangular, with length to width ratios of approximately 3:1, with dimension ranging from 50 m to 500 m, and depths of 2 to 4m. The seepage collection ditches range from 1.0 m to 2.0 m deep, a bottom width between 1.0m to 2.0m and 2.5H:1V side slopes. Seepage collection ditches alignments were designed to minimize excavation to required depth, but final excavation depth varies accordingly with terrain topography. The layout of the seepage collection facilities is shown on Figure 18-7.

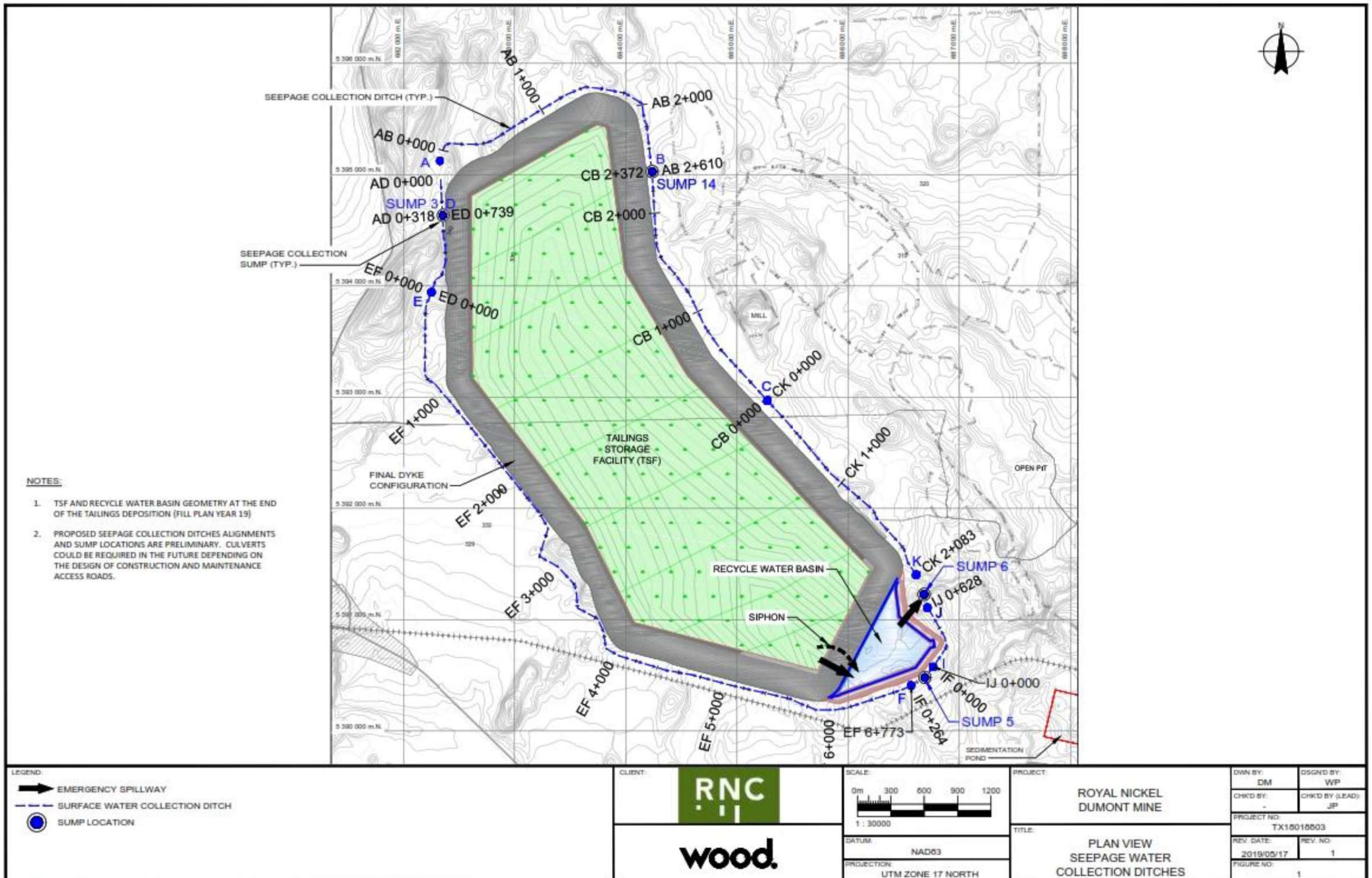
18.8.8 Tailings Delivery System

The tailings delivery system will transport separated coarse and slime slurried tailings from the processing plant to the TSF. The delivery system will be sized initially on the basis of a 52.5 kt/d operation and increased to 105 kt/d. The coarse tailings will be discharged along the perimeter of the TSF to build the dikes. The coarse tailings pipeline will initially consist of a DN630 HDPE (high density polyethylene) pipeline, approximately 4 km long. At year 2, this pipeline will be expanded in two 7.5 km long branches built with DN550 DN630 HDPE. A carbon steel section will be added at the pump discharge when the pressure caused by the dikes height will be too high for HDPE (around year 10). This pipeline will initially transport 2,041 m^3/h of coarse 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 4,082 m^3/h of coarse tailings to the TSF. The slimes tailings will be discharged in the center of the TSF. The initial slimes tailings pipeline will consist of a DN710 HDPE pipe approximately 4.5 km long. At year two a second branch approximately 3 km long consisting of a DN710 HDPE pipe will be installed. This pipeline will initially transport 1,656 m^3/h of slimes tailings to the TSF. A second line of the same size and length than 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 3,312 m^3/h of slimes tailings to the TSF. Both tailings thickener underflow pumps will be on emergency power to prevent the lines from freezing in case of a power loss.

18.8.9 Return Water Delivery System

The return water delivery system for recycle water from the TSF has been sized on the basis of 1,886 m^3/h of water being pumped from the TSF or Recycle water basin to the Process Water Pond, for the initial 52.5 kt/d operation. This system will consist of barge pumps and a DN600 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 3,772 m^3/h. The pipelines will be heat traced at low points to prevent freezing.

Figure 18-7: Plan View of TSF Seepage Collection System



Source: SRK.

18.8.10 Closure

The TSF reclamation starts at the end of Yr19. The surface of the TSF will be reshaped and the supernatant pond eliminated. A minimum soil cover of 0.5m thick will be established on the sides of the TSF perimeter dams and 0.15 m thick over the tailings surface.

Drainage swales will be established on top of the TSF to convey natural run-off water toward the seepage collection ditches and sumps at the toe of the TSF. Post TSF closure, the water quality in the sumps will be monitored and pumped to the open pit, during the pit in-filling, until water quality meets the effluent criteria. Once the water quality is deemed acceptable, where possible, the channels will be re-graded towards the open pit. An engineered outflow channel will be constructed to control the water level within the Pit and convey excess water to the Villemontel River. The inpit tailings storage and waste rock dump will permanently maintain passive water cover.

The soil required for TSF reclamation will be stored in two nearby stockpiles from active open pit and TSF stripping.

18.9 Truck shop & 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, in year two, 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 by an additional two truck bays in year 8 to a total of twelve bays. The building type will be structural steel and covered in architectural cladding. The tire yard is located beside the truck shop.

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.

18.10 Assay Laboratory

An area has been set aside for trailers or/a building to be supplied by the analytical services provider. A proposed location to the south of the office and warehouse buildings has been reserved. The construction of a building or rental of trailer(s) costs have not been included. The building/trailer costs are captured in the operating costs for sample processing by the analytical service provider. The labs will process samples from the mining and exploration operations, as well as the process plant.

18.10.1 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.

18.10.2 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.

18.11 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. The process water pond is fed from overflow from the tailings thickener and concentrate thickener, as well as from return water from the TSF or the Pit (during the inpit tailings disposal 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.

18.11.1 Raw Water

Raw water is retrieved mainly from the Quarry or from the pit (during inpit tailings disposal phase) 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.

18.11.2 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.

18.11.3 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.

18.12 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.

18.13 Transportation & Shipping

The concentrate loadout area is located at the north end of the process plant. A capacity of two days of concentrate production can be stockpiled in the filter discharge area building under cover, prior to being loaded on railcars on the concentrate loadout rail spur adjacent to the concentrate handling building. 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.

A drive through truck loading bay to the north end of the concentrate loading building complete with rolling doors at the west entrance and east exit has also been provided. Trucks can be loaded by front end loader while remaining under the cover of the building.

Nickel concentrate initial peak throughput will be 145 kt/a (based on 16 t/h nominal plant capacity and 92% availability) and 177 kt/a in phase 2 with a peak production of 230 kt/a. Based on six services per week, outgoing traffic will consist of seven 99 tonne wagons to be loaded per day during phase 2. A normal FEL (e.g., CAT 980) will have a productivity of 300 t/h. Therefore, only two to three hours of operation six days per week will be required to load seven 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 lowest cost 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 will be 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.

18.14 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.

18.15 Site Security

All entrants to the mine and 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.

18.16 Communications

18.16.1 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.

18.16.2 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.

18.16.3 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.

18.17 Surface Water Management System

18.17.1 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 are to:

- provide a reliable water supply to the concentrator, maximizing the use of recycled water from TSF;
- 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 divided 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, ditches, sump and pump systems, sedimentation ponds, and reservoirs that manage contact water and impacted contact water (water released from tailings) separately as the overall surface area or footprint of the mine expands.

18.17.2 Contact Water Diversions

Surface water runoff that 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 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 western and eastern branches of the unnamed creek around the ultimate footprint of the open pit. These drainage areas consist of two streams that are not very large, therefore, the diversions implemented could be either a pump and pipeline system or an open channel.

The western and eastern branches of the unnamed creek will be replaced by three major open channels that will route surface water away from the open pit and towards the Quarry. 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 11 sumps will be situated in low elevation areas throughout the catchment area of the western and eastern branches of the unnamed creek, 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 water 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 Quarry.

The contact water collected in the Quarry can be pumped to the concentrator as reclaim or as a raw water source. The open pit will also pump water to the Quarry through an oil separator.

Starting year 19, all water will be diverted to the main pit to accelerate filling and reclamation of the excavation.

18.17.3 Impacted Contact Water Diversions

Water released from tailings and surface water runoff that comes into contact with tailings is considered “impacted contact water”. This water includes runoff and seepage from the tailings dams collected by a network of channels and four sumps situated around the TSF, which will be pumped back into the TSF. The concentrator will reclaim as much of the TSF water as possible to minimize the requirement for treatment of the impacted contact water. Excess impacted contact water will be pumped to the water treatment plant for treatment and discharge to the polishing pond, which discharges to the Villemontel River.

18.17.4 Sedimentation Pond

The sedimentation pond is located south of the TSF and, for modelling purposes, is assumed to have a capacity of approximately 1 Mm³. Excess water from this pond reports either to the water treatment plant (if additional treatment is required) or Villemontel River. The sedimentation pond will be in place early in the construction phase, to capture and treat runoff for high suspended solids (TSS) throughout from the disturbed areas during the construction phase.

The sedimentation pond also receives excess water from the open pit area during construction, the Quarry prior to the start of low-grade ore (LGO) stockpile milling, water from sump 9 during construction, local runoff and direct precipitation. The pond allows high TSS (total suspended solids) to settle to acceptable concentrations. In addition, a CO_2 sparging system will be installed adjacent to the sedimentation pond to treat the water for high pH, before discharging to the polishing pond, which discharges to the Villemontel River.

The sedimentation pond was 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 pond has been set at 6 m, and the length to the width ratio is 3 to 1, respectively. The pond is situated south of the railway and open pit, and north of the administration building.

18.17.5 Tailings Management Facility (TSF and RWB)

The tailings management facility (TMF), which includes the TSF and the recycle water basin (RWB), will serve three 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 RWB will serve as the live water storage during operations to meet the demand of the concentrator (i.e., temporarily store water during wet periods for subsequent use in the concentrator during the winter). Finally, the TMF will provide enough storage capacity to manage the environmental design flood (defined as "crue de projet" by Quebec directive 019) or excess water that exceeds the treatment rate capacity, and that will be sent to the WTP between April to November.

The largest inflow to the TSF will be the water released from tailings and the largest withdrawal will be outputs of reclaimed water for the concentrator and excess water to be treated at WTP. 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.

TSF will operate with a supernatant pond maintained as low as possible to ensure the settlement of tailings particles. It is assumed for modelling purposes that the minimal required volume for the settlement is 1 Mm³, which will ensure approximately 5 days residence time based on expected flow of water released from tailings.

Operational procedures will be implemented to ensure that:

- Prior to spring freshet, water storage in both, TSF and RWB, will be maintained at the minimal to manage the freshet.
- Prior winter freeze up, sufficient water will be storage on the RWB to compensate for unavailable tailings water during the winter, to meet the recycle water demand for the concentrator.

Both, TSF and RWB, will maintain a minimum freeboard of 1.5 metres between environmental design flood water level and the dam crest.

Excess water from TSF and RWB will be pumped to the water treatment plant prior to discharge to the Villemontel River via polishing pond.

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 on the TSF water management are provided in Section 18.7.7.

18.17.6 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 that their runoff will not require treatment prior to discharge to the environment. 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 Quarry, 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 Quarry.

The low grade ore (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 Quarry.

It is assumed that infiltration into the stockpiles from rainfall will eventually report to the Quarry and excess water from the Quarry will be pumped towards the sedimentation pond.

18.17.7 Water Treatment

A CO_2 sparging system will be located adjacent to sedimentation pond to treat high pH contact water as required. 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 polishing pond.

Excess impacted contact water from TSF will be directed to the water treatment plant (WTP). Treatment will be required for possible elevated arsenic concentrations as well as other potential metals. The WTP will operate from April to November and provides management of the TSF pond elevations to maintain the minimal operational target levels as well as to manage a design flood. The treatment rate was optimized with the water balance model to prevent uncontrolled discharges of untreated impacted contact water to the Villemontel River. The plant will operate at a capacity of 0.7 m^3/s.

18.17.8 Polishing Pond

A polishing pond is located east of the lower reach of the Unnamed Creek, in parallel of sedimentation pond, and is assumed for modelling purposes to have capacities of approximately 1 Mm^3.

Polishing pond will receive treated water from WTP and water from sedimentation pond. Water quality will be tested to ensure it meets the effluent criteria prior to discharging into the Villemontel River.

19 MARKET STUDIES & CONTRACTS

19.1 Nickel & Stainless Steel Market Outlook

According to the long-term outlook (May 2019) by Red Door Research global nickel consumption is forecast to increase by 5.0% in 2019 to 2.33 Mt; and by 4.6% per year to 2.96 Mt in 2023; and 4.4% per year thereafter to 4.00 Mt in 2030. Both of the two main consumption sectors, stainless and non-stainless, are expected to grow in the future with the non-stainless sector being primarily driven by rapid growth in the use of nickel in lithium-ion batteries.

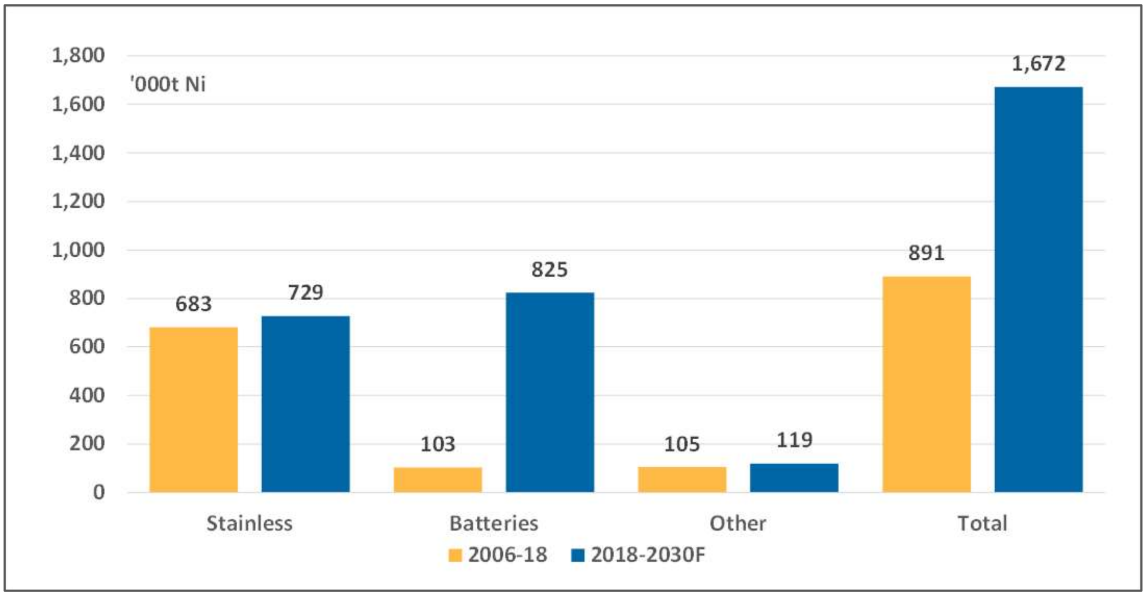
In 2018, stainless steel made up 70% of total world nickel use. Primary nickel demand in stainless steel is projected to increase to 2.4 Mt by 2030 driven by growth in global stainless melt output of 4.4% per annum to 63.5 Mt until 2023 with further growth of 3.4% per annum to 80.2 Mt until 2030. The bulk of this growth will be supported by the continued expansion of the Chinese stainless steel industry.

The fastest growing sector for nickel in recent years and for the foreseeable future is the use of nickel in lithium ion batteries for the booming electric vehicle market. Driven by governmental policy (to ban sales of internal combustion engine vehicles in the coming decades) and environmental concerns, the switchover of the existing car fleet from internal combustion engines to hybrid and ultimately fully electric vehicles (EVs) is now under way.

EVs are still a small portion of the vehicle market and in 2018 accounted for 5 million vehicles, only 2.2% of global vehicle sales, but the growth rate from 2014-2018 was just under 60% a year. With consensus trend growth rates of 25-35% a year in sales, the share of EVs will grow steadily with forecasts of the global market share of electric vehicles being 10-20% by 2025 and 30-50% by 2030.

According to Red Door Research, the forecast trend annual growth rate for nickel use is 17.9% a year for all battery types to 958 kt per annum by 2030, noting that the growth rate for lithium ion batteries for cars alone (which is only around 50% of 2018 total nickel use in batteries) is more than double the overall rate.

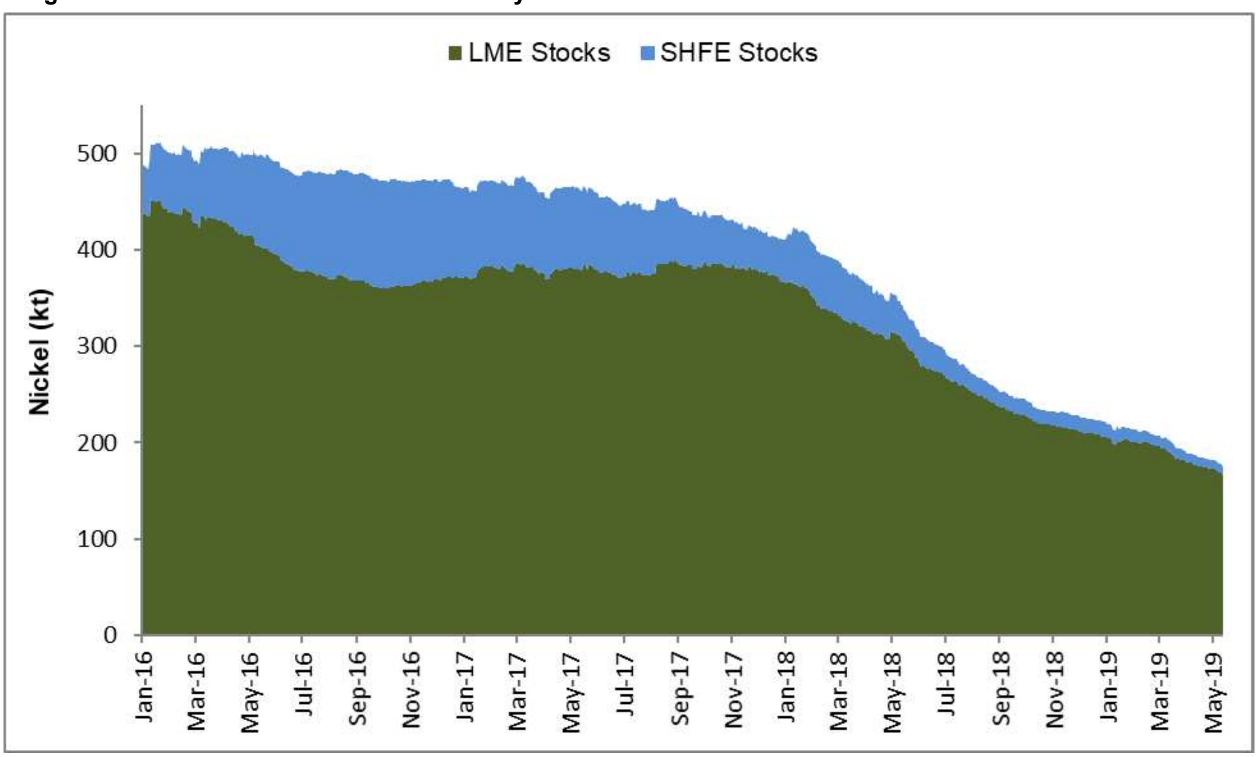
Figure 19-1: Nickel Consumption Growth Drivers, Stainless Steel and Batteries



Source: Red Door Research, INSG.

According to Red Door, ongoing deficits are expected between supply and demand, albeit at reduced rates from the large deficits experienced recently. Market inventories are expected return to normal levels by the end of 2020 and potentially fall to critical levels in 2022/2023, placing strong upward pressure on prices.

Figure 19-2: LME and SHFE Nickel Inventory Levels



Source: Red Door Research, INSG.

19.2 Price Assumptions

Pricing assumptions were developed for nickel and the cobalt, platinum, and palladium by-products contained in the Dumont concentrate based on forecasts as of April 2019. As the expected timing for the project falls within the long term forecast of those forecasts, a single nickel price was used for all production years. For the by-product 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

		Long-Term
Nickel	US$/lb	$7.75
Cobalt	US$/lb	$25.00
Platinum	US$/oz	$1,000
Palladium	US$/oz	$1,000

A long-term nickel price assumption of $7.75 per pound was utilized in the study which is consistent with the average long-term nickel price of forecast given by two leading independent nickel industry analysts.

The metal price assumptions for both platinum and palladium of $1,000 per ounce were consistent with the forecast long-term average prices published on April 15, 2019 by Consensus Economics Inc. of $1,1272 per ounce for platinum and $1,088 per ounce for palladium. The metal price assumption for cobalt of $25 per pound was consistent with the average forecast long-term price published by a leading independent cobalt industry analyst. 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, is ideally suited for use as a high quality raw material feed for ferronickel or nickel pig iron producers through a roasting process. The high nickel content of the concentrate means that lower amount of power, reductants (coke), and energy are required for processing resulting in lower costs and payabilities that are higher compared to traditional smelting and refining. CRU, a leading, provider of analysis, prices and consulting in the mining, metals and fertilizer markets, prepared a value-in-use study and market analysis that looked at toll-processing in Asia for a range of nickel concentrates with nickel content ranging from 14% to 29% through to a final ferronickel product. It found that net payabilities utilizing this approach for all concentrates were significantly higher than current market terms based on CRU's estimate of long-term nickel prices, product premiums, and tolling costs. For a 29% nickel concentrate, which is the grade expected to be produced from Dumont, the nickel payability for concentrate was estimated to be 94%, or 25% higher than the top end of the estimated current 70-75% market range for nickel sulphide concentrates.

CRU has considered a scenario under which Asian processors would process nickel concentrate on a tolling basis. This means that the concentrate producer would retain ownership of the ferronickel product, and simply pay the processor to cover their costs, plus a margin.

The tolling costs were derived by estimating the operating costs and margins at a typical NPI producer using CRU's Nickel Cost Model and the estimated operating costs and margins at a typical roasting facility. CRU adjusted the NPI producer costs by assuming that the higher nickel content of the concentrate means that lower amounts of power, reductants, and energy would be required in the furnace for each tonne of nickel produced. The assumption is made that this relationship is linear, and that other costs remain constant. An additional margin to incentivise the switch to using nickel concentrate rather than their typical feed was then applied to these costs. Due to the current and forecasted thin margins at some NPI producers, only a relatively small incentive payment would be required to encourage such NPI producers to use a nickel concentrate for feed instead of their current laterite feeds.

The technical viability of using roasted nickel sulphide concentrate has been successfully demonstrated. After successfully initially demonstrating the potential of roasted nickel concentrate as a more valuable alternative to traditional smelting and refining in 2011, RNC worked with the Tsingshan Group ("Tsingshan"), beginning in 2012, to validate the concept. In 2014, Tsingshan began construction of the first plant to directly utilize nickel sulphide concentrate as part of the stainless steel making process and has since built an additional plant utilizing the roasted nickel concentrate approach. Additionally, Tsingshan signed an offtake agreement with Western Areas Ltd. in late 2016.

Additionally, RNC's work with a large Japanese trading house indicates that roasters in Asia are able to process feed at an approximate cost of $30/tonne. When combined with the average integrated NPI/stainless conversion cost of approximately $80-$90/tonne (according to Wood Mackenzie), the implied conversion cost is approximately $110-$120/tonne of concentrate (equivalent to approximately $400 per tonne of contained nickel for a 29% concentrate or approximately 3% of the recent LME price of $11,900 per tonne). This compares very favourably to the 25-30% of the concentrate value believed to be currently captured by traditional smelters/refiners.

For purposes of the feasibility study update, a more conservative payability of 91.5% was assumed.

19.4 Smelter Options

With roasting, no payment will be realized for the cobalt and PGMs contained in concentrate. At higher prices for cobalt and/or PGMs, it could be more economic to treat the concentrate or a portion of the concentrate via conventional smelting and refining or by alternate processes to allow the nickel and cobalt to be utilized by the battery industry. RNC continues to evaluate and discuss with potential partners a range of market alternatives for concentrate treatment.

There are various nickel smelters globally. Brief profiles of the most likely smelters are provided in the subsections below.

19.4.1 Glencore

The Glencore 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. As such, it should be possible to treat Dumont concentrate at the smelter in Falconbridge without exceeding MgO limits.

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 twelve years. Biophysical data come mainly from three distinct fieldwork programs performed from 2007 to 2009, with some complementary information extracted from the baseline studies designed to support the Environmental and Social Impact Assessment in 2011 and 2012. RNC has hired consultants over the past 5 years to optimize the project and consequently, additional data were acquired from 2013 to 2018. Table 20-1 summarizes the sources of information for the various biophysical and social components described in this section.

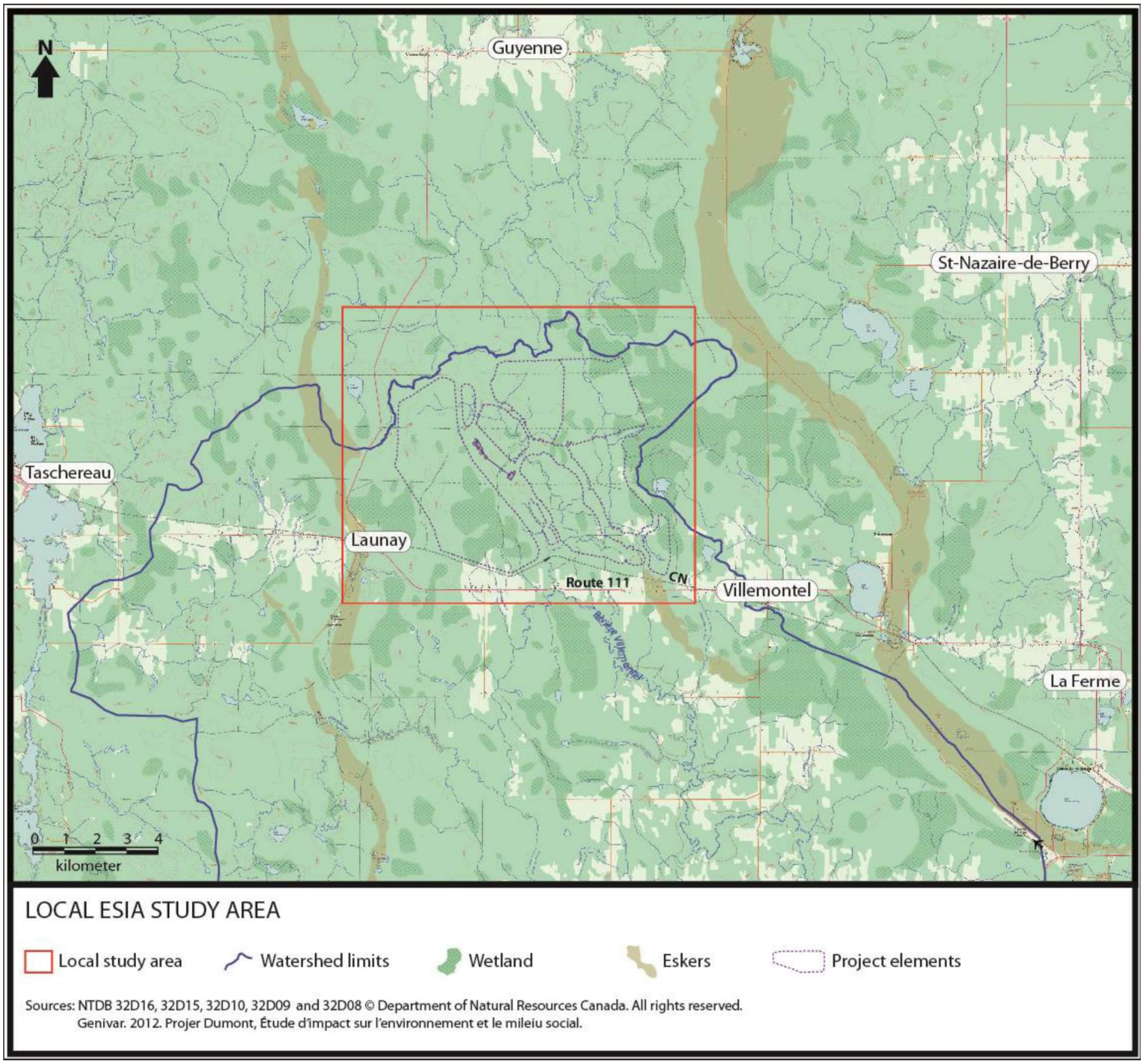
Table 20-1: Studies used to describe Biophysical & Social Components included in the Feasibility Study and up to year 2018

Type of Study	2007	2008	2009	2011	2012	2013	2014	2015	2016	2017	2018
Climate				√	√	√	√	√	√	√	√
Air quality							√	√	√	√	
Hydrology and bathymetric survey				√	√	√	√	√	√		
Water and sediments quality	√	√	√	√		√	√				
Groundwater quality				√	√	√					
Soil characterization					√	√					
Rare and protected plants	√			√							
Environmental geochemistry				√	√	√					
Vegetation and wetlands		√		√			√				
Wildlife	√	√	√								
Small mammals				√							
Fish	√	√	√	√	√		√				
Benthic invertebrates	√	√	√								
Birds		√		√				√			
Reptiles and amphibians				√		√		√			
Ambient noise				√		√					
Infrastructures								√			
Archaeology		√				√					
Public and Stakeholders				√	√		√	√	√	√	√

Notes: 1. References are specified in the sections 20.1 to 20.4 and 20.7. RNC **Source:** RNC.

The study zone for the ESIA encompasses an area larger than the footprint of the project, as shown in Figure 20 1.

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 range 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 on site 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. A second weather station

Ausenco

(station Lacroix) installed in December 2014 registers data every year and shows similar data. No detailed analysis has been performed on these data.

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, i.e. a succession of water bodies 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. Where unnamed stream 1 empties into the Villemontel River, it 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 discharge directly into the Villemontel River, just upstream from unnamed stream 1.

From 2011 to 2016, hydrological data were collected concerning water current velocity, water level variation and flow in the Villemontel River and unnamed streams 1 and 2.

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 footprint.

The groundwater of the bedrock aquifer and the overburden aquifer of the study zone, other than the groundwater from the eskers, is considered as Class II hydrogeological formations (Class I being of high importance and Class III of lower importance), the later being used only locally to supply water to private properties along Highway 111.

However, the Saint-Mathieu-Berry (outside of the study zone area and on the other side of the drainage basin limit), 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 water flow.

In the environmental impact assessment study, 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/year to 1.1 m/year in the overburden and 7.8 m/year to 15.3 m/year in the near-surface bedrock. The flow velocities do not exceed 0.06 m/year 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 was slightly alkaline (pH most often slightly higher than 7.0) and moderately hard (total hardness most often between 17 and 57 mg/L) during measurement campaign. It was rich in organic carbon, which was mainly found in dissolved form, at concentrations ranging between 4 and 28 mg/L. The turbidity was 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 were separated into three distinct groups based on surface water characteristics. The Villemontel River differed from the other watercourses in several regards. Its water was harder, and its major ion concentration was higher, which translated into a specific conductance about twice as high.

Among the measured nutrients, the total phosphorus concentrations were 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 MELCC, which is intended to prevent eutrophication of water bodies. Observed exceedances were observed at all stations, which are evidence of eutrophic aquatic environments.

Among the measured metals, the aluminium concentrations were especially high. They generally exceeded the MELCC ‘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 iron concentrations regularly exceeded chronic aquatic life toxicity criterion.

In 2013, similar results were obtained. In all samples, aluminium concentrations exceeded the chronic aquatic life toxicity criterion. Arsenic, copper and iron concentrations exceeded the CCME criteria in two sampling stations. Lead concentrations exceeded the chronic aquatic life toxicity criterion in four out of 5 sampling stations.

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 are no evidence that the site could have been contaminated by past activities. A soil characterization program was planned in 2013 to evaluate baseline conditions prior to project implementation. All samples showed metal concentration under the level “A” criteria except for three samples who were classified as level “A-B” because of high tin and/or chromium concentrations.

Geochemical characterization of the overburden that will be manipulated and stockpiled was performed in 2012 and results are presented in section 20.7 of this chapter.

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 deforestation activities have fragmented several natural environments.

The majority of the terrestrial environments has medium ecological value. However, intolerant deciduous trees, uncultivated grassland, scrubland and recent deforestation areas 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, they form a diversity of interesting natural 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. A precise cartography was made in 2014 to develop an offsetting plan for wetlands that will be disturbed by the Dumont project (see section 20.5.3.1).

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 Quebec Ministry of Natural Resources and Wildlife (MNRF now MFFP) 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 MFFP, Lac à la Savane, slightly outside of the Dumont property, is considered a protected muskrat habitat.

20.1.10 Small Mammals

A field survey designed under the MNRF (now MFFP) 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 mixt forest located on rock outcrops. This habitat located west of "Lac à la Savane, 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.

In 2014, a survey was completed to select potential sites to offset fish habitat perturbation associated with the Dumont Project and an offset plan was designed and approved by the MELCC (see section 20.5.3.2).

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.

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 water bodies 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.

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 the mining infrastructures was conducted. No wood turtles were observed. This species is likely to be designated threatened or vulnerable in Quebec.

Furthermore, real time recordings performed onsite 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) in order to confirm the absence of this species on the Dumont property. This species is considered threatened by both provincial and federal governments. An exhaustive survey was performed in 2013 confirming the absence or extremely rare presence of the species (Génivar 2013).

20.2.3 Birds

2011 field inventories 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. The rusty blackbird is of special concern and the olive-sided flycatcher and the common nighthawk are considered threatened under the Species at risk federal law. In 2015, these three species were surveyed during breeding period. Seven recordings were made, one from an olive-sided flycatcher and six from common nighthawks. This study concludes that these three species are not abundant in the study zone and the projet footprint would not affect their population (WSP Canada Inc. 2015).

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 24,400 (MAMH, 2019). 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 2016	12,823	538	218	538	963	136	1,223	8,164,361
Total population of 15 years old and over in 2016	84 %	78.5 %	88.6 %	69.2 %	81.3 %	85.2 %	82.8 %	83.7 %
Area in 2016 (km^2)	430.3	577	258.5	1	250.7	720.8	197.1	1,356,625
Population density per km^2 in 2016	29,8	0.9	0.84	538	3.8	0,2	6.2	6.0
Average age of the population in 2016 (yrs.)	42.7	37.6	44.8	30.8	42.3	43.3	41.8	41.9
Employment rate in 2016	60.6 %	62.8 %	67.6 %	41.7 %	46.4 %	51.9 %	60.2 %	59.5 %
Unemployment rate	7.4 %	15.4 %	8.7 %	16.7 %	14.3 %	13.3 %	12.5 %	7.2 %
Total population 15 years old and over without certificate, diploma or degree in 2016	26.4%	49.2 %	33.3 %	51.4 %	32.7 %	57.1 %	25 %	19.9 %
Median income in 2015 – All private households ($)	43,425	35,111	N.D.	26,314	34,769	N.D.	47,455	42,546
Dwellings requiring major repair - as a % of total occupied private dwellings in 2006	7.9 %	17.1 %	18.2 %	34.4 %	14.1 %	N.D	5.2 %	6.4 %

Sources: Statistics Canada, 2016 Census of population - Community profiles.

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 218 inhabitants and 118 private dwellings in the municipality (Statistics Canada, 2016). Most of its territory is located on public (Crown) lands. The limits of the Dumont project 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, which was exacerbated by the closure of a saw mill, its only industry, in 2006.

On September 26th, 2012, RNC and the municipality of Launay entered into 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. A permanent collaboration and partnership agreement was signed on December 15th, 2015. This agreement will be implemented when project construction starts.

20.3.1.3 Trécesson (Villemontel)

The township of Trécesson, with 1,223 citizens and 558 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 occurring within this township that is experiencing an increase in population.

On October 13th, 2013, RNC and the municipality of Trécesson entered a provisional collaboration and partnership agreement. The main objective of this agreement is to formalize the collaboration between RNC and the municipality of Trécesson to the benefit of the community and the advancement of the Dumont project. A permanent collaboration and partnership agreement was signed on December 18th, 2015. This agreement will be implemented when project construction starts.

20.3.1.4 Unorganized Territory of Lac Chicobi (Guyenne)

The town of Guyenne, with 136 inhabitants, is located 10 km north of the project site in one of the two unorganized territories managed by the regional municipality. Economic activity is mainly related to agriculture and forestry. Lac Chicobi is located in this area and hosts cottages and a summer camp.

20.3.1.5 Berry

The project touches the southwest corner of the municipality of Berry. This municipality of 538 citizens and 264 private 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 has a total of 963 inhabitants and 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 permanently in 2011. The economy is based on agricultural and forest activities and on a new tourist and recreational project. Taschereau is located at the northern limit of the Aiguebelle Provincial Park. It offers lodging and restaurants, and benefits from its location beside the lake Loïs.

20.3.1.7 Pikogan (Abitibiwinni First Nation)

The First Nation reserve of Pikogan is located along the Harricana River and occurs 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 1,075 persons registered as Abitibiwinni band members, of which 610 live in Pikogan (Aboriginal Affairs and Northern Development Canada, Indian registry – 2017). 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 5th, 2013, RNC and the local Algonquin First Nation Conseil de la Première nation Abitibiwinni ("PNA") signed a memorandum of understanding (MOU). Subsequently, an Impact and Benefit Agreement (IBA) for the Dumont Nickel Project was signed on May 2nd, 2017, between the PNA and the Dumont JV. The IBA serves as a framework to govern the relationship with the PNA and lays out the commitments of the parties regarding the impacts and benefits 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 main rights granted by the provincial government in regards with this territory are related to forestry uses. 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 MERN there are five leases for hunting camps and two registered traplines within the Dumont project boundaries. RNC has reached lease assumption agreements with registered hunting camp owners. Exchanges are ongoing with the local trapping association and government authorities to remove the Dumont Project footprint from registered traplines.

20.3.2.2 Private Lands

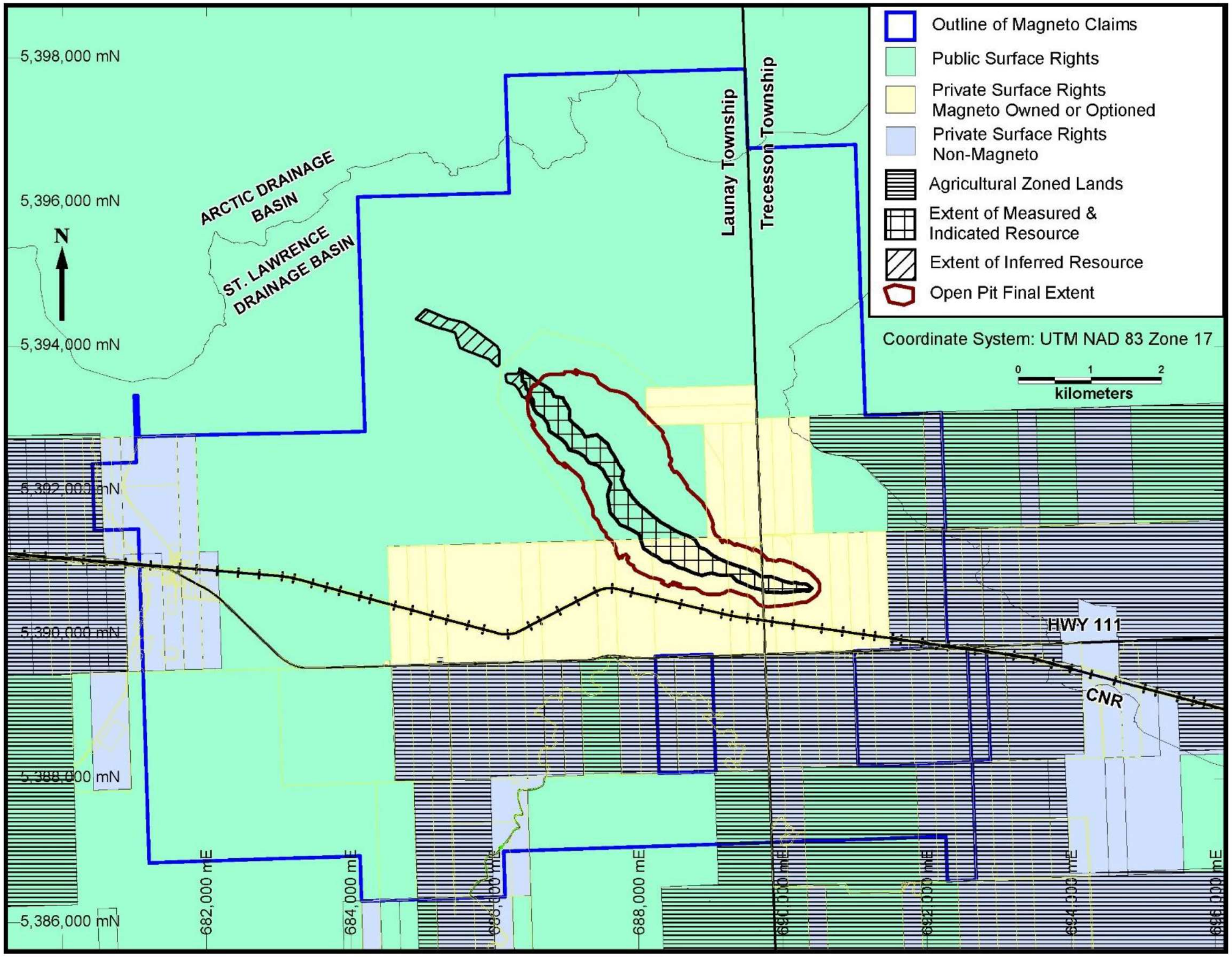
Part of the land proposed for project surface infrastructure development is privately held (Figure 20-2). These lands are assigned to agroforestry uses in the regional municipality land development plan. RNC has either purchased or concluded purchase options for all required properties (see Section 5.5).

20.3.2.3 Agricultural Area

A portion of the private and public lands on the southern portion of the project were previously located in the provincial agricultural zone (See Figure 20-2). Uses other than 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 20.6.4).

Exclusion of these lands from the agricultural zone was granted by the CPTAQ in 2013 and 2015.

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. It states that very little is known regarding the archaeology of the surroundings of the Dumont property. 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 was performed in the summer of 2013. No artefacts or archaeological sites were found.

20.3.4 Ambient noise

A first measurement campaign was held in 2011 to assess the ambient noise and the maximal noise level authorized depending of each zone related to MELCC instruction note (NI 98-01). Measurements were taken at 6 different stations corresponding to sensible receptors.

The principal noise was related to road traffic and, when no vehicle was present, natural environment sounds (birds, wind). A second campaign was held in 2013 to update 2011 results. The measured noise level range was 36 to 63 dBA ($LA_{eq\ 24\ h}$). Therefore, the maximum limits to be respected depending to the receiving point are from 42 dBA (night time) to 62 dBA (day time) ($LA_{eq\ 24\ h}$).

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 feasibility study where possible and help define the content of the environmental impact study.

This approach comprises three main stages:

- an information and consultation process associated with the pre-feasibility study;
- a consultation process associated with the ESIA;
- a consultation and information process following the ESIA submission.

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 development 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 was produced by Transfert Environnement in 2013 and submitted to the relevant authorities, as well as being filed as a public document on the company's website.

Following the ESIA submission in 2013, the BAPE (bureau d'audiences publique en environnement) consultation process took place in 2014. Also, meetings were held with municipalities representatives and public information sessions were planned and realized between 2014 and 2018.

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

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

Table 20-3 and Table 20-4 respectively present the main concerns and the location selection criteria discussed during the information and consultation activities.

Table 20-3: Main Issues of Concern raised during the Information and Consultation Processes

Category	Issues of concern
Information and consultation processes	1. Operation, composition, resources and role of the committees 2. Access to information on the project 3. Purpose of the consultation process
Methods and means of impact analysis	4. 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) 5. Accuracy of the data used (e.g., margin of error) 6. Ongoing impact analysis 7. Accounting for related projects
Economic development	8. Impacts on the local and regional economy 9. Maximization of local and regional benefits 10. Residential and industrial development 11. Retention of newcomers and population growth
Water	12. Protection of groundwater (eskers, wells, etc.) 13. Potential Contamination of surface water 14. Chemical composition and management of effluent from the impoundments (waste rock piles and tailings storage facilities) 15. Mitigation and compensation measures for impacts on water
Soil and location of components	16. Distance of components from the highway and residences 17. Area of affected land
Fauna, flora and wetlands	18. Impacts on large fauna 19. Compensation for destruction of wetlands
Visual impacts	20. Effect on the landscape 21. Mitigation measures for visual impacts
Climate and air quality	22. Dust emission 23. Dust control and mitigation measures
Human environment	24. Use of the railway 25. Recreational tourism and agroforestry activities 26. Purchase of nearby residences and negotiating process 27. Real estate development 28. Increase in the value of housing and its impact on the ability of residents to pay their taxes 29. Benefits for the community in terms of infrastructure and community investment 30. Social fabric and quality of life
Health and safety	31. Transport of chemicals 32. Health risk to workers and residents related to the presence of chrysotile in dust

Category	Issues of concern
	33. Emergency response plan 34. Site security
Nuisances	35. Noise 36. Nuisances during the exploration and development phases 37. Dust emissions 38. Road congestion 39. Heavy vehicle traffic
Restoration and post-closure	40. Plan for site restoration and future use 41. Financial guarantees for site restoration 42. Economic diversification fund
Project (various)	43. Possibility of gradually filling the pit 44. Exploratory drilling and boreholes 45. Profitability of the project 46. 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)

20.4.1 Future Consultation Activities

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. Consultation activities will be planned share the results of the updated feasibility study.

20.5 Preliminary Environmental & Social Impact Assessment (ESIA)

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 performed in the ESIA describes the residual impact once mitigation measures are applied. On the whole, the majority of the impacts are qualified as being of 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

- 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;
- 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;
- 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. 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 allowed a more precise assessment of the health risks and helped RNC to set up follow-up and preventive measures within the framework of the emergency response plan, in order to ensure adequate protection of workers and the nearby population.

As part of the current study in 2018 and 2019, modifications were made to the project design. An update of the environmental and social impacts evaluation was carried out to consider these modifications. The nature of negative impacts previously identified in the preliminary ESIA remain the same but some of these impacts will be reduced in intensity. However, the negative impact reduction is not significant enough to result in a change in the impact importance evaluation when the impact evaluation methodology is applied.

The environmental components where the project impacts are reduced are air quality and noise. The negative impacts on air quality will be reduced because of the use of a trolley-assist system on ramps in the open pit and a mobile trolley-assist system on the waste dump ramp. The reduction of material transportation (from 2.5 billion t to 2.0 billion t total mined material), combined with the increase of truck payload and reduction of transportation to the TSF will also help attenuate impacts on air quality (WSP 2019a). A comparison between fixed and mobile equipment was completed and the expected related modifications to the ambient noise will be similar to the modelled projected noise environment presented in the EIA (WSP 2019b).

20.5.2 Mitigation Measures

In conjunction with 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 project development on the 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 dike failure.
- implementation of intensive dust control measures to reduce the project impact on air quality for surrounding populations.
- Acquisition of private houses located on the north side of the road 111 that are closer to the mine site and more likely to be impacted by mine activities.
- 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 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 (Ministère de l’environnement et de la Lutte contre les Changements climatiques; MELCC) for acceptance and would be implemented during the construction phase.

A first survey in 2011 and a second in 2014 were completed to characterize and to select potential sites to offset the wetland loss. Map and description of sites that can be used to offset wetlands loss were prepared.

Considering the previously mentioned challenges and exchanges between RNC and the MELCC (formerly known as MDDELCC) as part of the Dumont Project Environmental Assessment, the compensation plan for the loss of wetlands has been divided into three phases covering the entire life cycle of the mining project: 1) conservation and protection of existing wetlands in Launay and Amos, 2) development of a guide on restoration of wetlands at mine sites (including a pilot project) and 3) the restoration or creation of wetlands at the Dumont site post closure (RNC, 2015). A five-year follow-up program will be conducted to assess progress in the implementation of the compensation plan and implement potential updates.

The plan was submitted to government authorities and approved. The report presenting the compensation plan (RNC, 2015) is referenced by the ministerial decree through which the project was authorized (provincial Certificate of Authorization).

20.5.3.2 Fish Habitat

According to the feasibility study site layout, the development of the Dumont project is likely to negatively impact about 35 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. A first survey in 2011 and a second in 2014 were realized to select potential sites to offset the fish habitat loss.

Under Section 27.1 of the *Fisheries Act* RNC is 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 must satisfy both provincial and federal levels of government. 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. The watercourse located in the footprint of the waste rock dump, low grade ore stockpiles, and overburden impoundments have been registered in Schedule 2 of the Metal Mining Effluent Regulations (MMER) under Article 36 (3) of the *Fisheries Act* (see.20.6.5.2 section).

RNC has proposed a mitigation measure to offset the loss of fish habitats and serious damage to fish caused by the Dumont Project, namely the reconstruction of a disrepaired retaining structure located at the outflow of Dasserat Lake to preserve fish habitats in Dasserat Lake and thus allow an increase in fish habitat during low-flow periods and ensure preservation of the habitat threatened by the instability of the existing dam (RNC, 2016). This plan was presented to federal authorities (DFO) and was accepted as part of the request for listing the impacted watercourse in Schedule 2 of the MMER.

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 used to determine 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.

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 current 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 evaluated various alternatives for mining residues storage and clearly demonstrated that the proposed scenario is the most appropriate under environmental, technical, economic and social considerations (section 20.5.3.2). In addition, under Section 27.1, RNC developed and implemented a plan to compensate for damage, destruction and loss of fish habitat (section 20.5.3.2).

20.6.4 Other Applicable Law & Regulations

20.6.4.1 Quebec Mining Act

In order to obtain a mining lease, a 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 based on the approved closure plan. 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.

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.

A first application for exclusion of lands required for mining infrastructures (1,680.46 ha) was submitted to the CPTAQ in February 2013 by the RCM supported by resolutions from the two municipalities directly concerned, Launay and Trécesson. An 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. The exclusion was ordered by the CPTAQ in August 2013.

A second application for the exclusion of remaining agricultural lands between the railway and Highway 111 was submitted to the CPTAQ in November 2014 by the RCM (approximately 201.3 ha). This application was also supported by resolutions from Launay and Trécesson. This exclusion was ordered by the CPTAQ in May 2015.

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 government, 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, now MELCC). 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 (ESIA) 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
Public hearing process kick-off	April 2014 - Completed
BAPE recommendations to provincial authorities	September 2014 - Completed
Regulatory review of ESIA	May 2015 - Completed
C of A delivery (Provincial)	June 2015 - Completed
Environmental assessment decision (Federal)	July 2015 - Completed
Water body listed in Schedule 2 of the Metal Mining Effluent Regulations (MMER)	December 2017 - Completed

Source: RNC, 2019.

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 starts once the developer and the DFO come to an agreement with regards to a funded compensation plan for fish habitat loss. This agreement was concluded in October 2016.

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.

The sections of creek impacted by the two cells of the tailings storage facility were not considered to be fish habitat by DFO and would therefore not trigger the MMER schedule 2 amendment process.

RNC has demonstrated to Environment Canada that waste rock and overburden are not deleterious based on extensive environmental geochemistry characterization. However, an authorization process had to be initiated for the low-grade ore stockpiles as low-grade ore was considered as a deleterious mining waste. This process was completed in December 2017 with the registering of the watercourse located in the footprint of the low grade ore stockpiles in Schedule 2 of the Metal and Diamond Mining Effluent Regulations (MDMER).

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 static leaching test on the composite mineralized rocks showed no concentration in leachates above the criteria. The humidity cell kinetic leaching 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	No. of Samples	Bulk Potential by Rock Type 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. Criteria are those of the Politique de protection des sols et de réhabilitation des terrains contaminés (2013), in effect at the time of the report. Groundwater quality criteria have evolved since then and compliance may differ from stated in this table.

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 (in effect in 2013). 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 leaching 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; criteria in effect in 2013). 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. Criteria are those of the Politique de protection des sols et de réhabilitation des terrains contaminés (2013), in effect at the time of the report. Groundwater quality criteria have evolved since then and compliance may differ from stated in this table.

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. Criteria are those of the Politique de protection des sols et de réhabilitation des terrains contaminés (2013), in effect at the time of the report. Groundwater quality criteria have evolved since then and compliance may differ from stated in this table.

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 (in effect in 2013). 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 Témiscamingue. 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 in effect in 2013. Leachates also generally meet these groundwater criteria with few isolated exceptions for silver, arsenic and manganese (few cycles and marginal exceedances).

Results obtained to the date of the Golder 2013 report corroborate those obtained from the smaller scale standard humidity cell kinetic leaching tests and larger leaching columns; they suggest that leachate water quality contacting tailings and waste rock is likely to be better than those on which are based the leachable classification for these wastes.

20.7.4 Carbon Sequestration and Tailings Cementation

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 can sequestrate carbon by binding atmospheric carbon dioxide (CO2) 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 study, involving researchers from Laval University and Université du Québec en Abitibi-Témiscamingue (UQAT), aimed 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 results show that carbonation mechanisms are influenced by:

- Air CO_2 concentration and its dissolution in water;
- Weather conditions that, in turn, influence input and evaporation of fluids and surface temperature;
- Rock Magnesium Oxides differences in terms of reactivity;
- Material porosity and exposed surface area.

Between 2012 and 2017, researchers from Université Laval and the Université du Québec en Abitibi-Témiscamingue (UQAT) worked with RNC on various projects aimed to better understand carbonation mechanisms.

This work was conducted both at the Dumont project site, at the experimental cell level and in the laboratory in leaching column, mini-cell and diffusion / differential carbonation tests.

The results of these various experiments (A. Entezari-Zarandi, 2017; EHB. Kandji, 2017; A. Gras, 2018) confirmed that both tailings and waste rock of the Dumont project have the capacity to sequester CO2 and that:

- Sequestration mechanisms leads to the formation of hydrated magnesium carbonates including hydromagnesite, nesquehonite or dypingite.
- CO2 from ambient air that is dissolved in pore water is a limiting factor in the carbonation reaction and carbonate precipitation is mainly driven by evaporation.
- Brucite reacts faster than serpentine.
- Low temperatures slow down the carbonation reaction
- Aging of the material (drying / wetting cycles, freezing / thawing) has an effect on the carbonates that are formed.

Experiments revealed that the process of sequestration is also accompanied by a cementing of the material. The 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.

An onsite experiment was also conducted by RNC in 2013 over a 16-week period to characterize short-term weathering of ultramafic waste peridotite, dunite and tailings. The purpose of the onsite experiment was to assess the rate at which the carbonation reaction takes place. Samples were taken after each week of weathering and analysed with SEM equipped with EDS. The tests were conducted both on material exposed outdoor to natural weather conditions and material kept indoors and watered in a way that reproduce external precipitation.

At the microscopic level, signs of the carbonation reaction were observed from the first week of exposure of tailings and fragments of dunite and peridotite. Carbonation and cementation were observed to occur more rapidly in tailings. In the tailings, the carbonation occurred mainly in the exposed upper layer (less than 1 cm) and formed a crust in the first month. An agglomeration between the fragments of dunite and peridotite appeared after 2 to 3 months. Cementation between grains was facilitated when fine particles were present in the tested material. Finally, it was also noted that low temperatures and snow covering on outside tests slowed the carbonation reaction.

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 (1 f/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 have been 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. Measures will be included in the health and safety plan to protect workers during operation activities.

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.

Wind erosion tests were carried out in 2013 on mine tailings using a wind tunnel. These tests were done on tailings samples with different moisture contents (1.44 to 29.8%), with and without the use of abrasive (sand) and for wind speeds ranging from 0 to 17m/s. Particles raised by erosion were measured by a Pm10 particle detector and fibers recovered on filters (NIOSH 7400). The tests found that only negligible amounts of dust were detected regardless of moisture content or wind speed and that no fibers were identified as chrysotile among the fibers collected on the filters.

A toxicological study (Sanexen, 2014) was performed to assess the health risk to neighbouring populations from airborne chrysotile from the Dumont Mine. This study was performed as a follow-up after the main ESIA to address concerns raised by the Environmental Public Hearings. The study concludes that the long-term health risk posed by chrysotile to the neighbouring populations is essentially negligible.

It should be noted 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 update of the capital cost of the project, including the 52.5 kt/d production rate (phase 1), expansion to 105 kt/d (phase 2), and sustaining expenditures over the 30 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, truck shop and wash bay;
- Utilities;
- On-site infrastructure;
- Off-site infrastructure;
- Indirect costs; and
- Contingency.

Wood

- Waste dumps;
- Channel design; and
- Sumps.

David Penswick (independent consultant)

- 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 (CAD) unless otherwise indicated.

21.2 Capital Cost Estimate Summary

The estimate for the FS portion 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 (phase 1), expansion capital (phase 2), and sustaining capital. The costs are expressed in real, Q1 2019 Canadian dollars.

Items originally received in foreign currency were converted in Canadian dollar. For USD currency, the exchange rate (CAD to USD) of 0.75 was used. For others currency, rate as of 2019-04-23 from “Oanda.com” were used.

Indirect costs include first fills of consumable items for the initial and expansion estimates.

Table 21-1: Summary of Capital Costs

Description	Initial Capital (CAD $M)	Expansion Capital (CAD $M)	Sustaining Capital (CAD $M)	Total Capital (CAD $M)
Mine [2,3]	**298**	**0**	**600**	**898**
Process Plant	**461**	**447**	**64**	**972**
Tailings	**48**	**31**	**168**	**247**
Utilities [3]	**180**	**133**	**0**	**312**
Infrastructure [3]	**95**	**24**	**0**	**119**
Indirect Costs [1]	**124**	**87**	**-16**	**196**
Owners Costs [1]	**40**	**7**	**0**	**46**
Contingency	**111**	**71**	**0**	**182**
Total	**1,357**	**801**	**814**	**2,973**

Notes: **1.** Negative indirect costs for sustaining capital reflect the release of first fills.

Table 21-2, Table 21-3 and Table 21-4 show details of the initial, expansion and sustaining capital costs by Area and include the composition by currency.

Table 21-2: Initial Capital Costs by Area (Phase 1)

Area	Currency Composition (CAD $M)	(USD $M)	(AUD $M)	Total Cost (CAD $M)
Area 1 - Mining	298	0.06	0	**298**
Area 2 - Crushing	43	14	0	**61**
Area 3 - Process	262	100	6	**400**
Area 4 - Concentrate Load Out	0.3	0.01	0	**0.3**
Area 5 - Tailings	46	2	0	**48**
Area 6 - Utilities	174	4	0	**180**
Area 7 - Onsite Infrastructure	79	0	0	**79**
Area 8 - Off-site Infrastructure	16	0	0	**16**
Sub-Total Directs	**918**	**119**	**6**	**1,082**
Area 9 - Indirect Costs	117	5	0.1	**124**
Area 10 - Owner’s Costs	40	0	0	**40**
Sub-Total Indirects	**156**	**5**	**0**	**164**
Total Directs + Indirects	**1,075**	**124**	**6**	**1,246**
Area 11 - Escalation	Excluded			
Area 11 - Contingency	95	12	1	**111**
Total Project Costs	**1,169**	**136**	**7**	**1,357**

Ausenco

Table 21-3: Expansion Capital Costs by Area (Phase 2 only)

Area	Currency Composition (CAD $M)	(USD $M)	(AUD $M)	Total Cost (CAD $M)
Area 1 - Mining	0	0	0	**0**
Area 2 - Crushing	42	13	0	**59**
Area 3 - Process	256	99	0	**388**
Area 4 - Concentrate Load Out	0	0	0	**0**
Area 5 - Tailings	27	3	0	**31**
Area 6 - Utilities	127	5	0	**133**
Area 7 - Onsite Infrastructure	24	0	0	**24**
Area 8 - Off-site Infrastructure	1	0	0	**1**
Sub-Total Directs	**475**	**120**	**0**	**635**
Area 9 - Indirect Costs	80	5	0	**87**
Area 10 - Owner's Costs	7	0	0	**7**
Sub-Total Indirects	**88**	**5**	**0**	**95**
Total Directs + Indirects	**563**	**125**	**0**	**730**
Area 11 - Escalation	Excluded			
Area 11 - Contingency	55	12	0	**71**
Total Project Costs	**618**	**137**	**0**	**801**

Table 21-4: Sustaining Capital Costs by Area

Area	Currency Composition (CAD $M)	(USD $M)	(AUD $M)	Total Cost (CAD $M)
Area 1 - Mining [2,3]	600	0	0	**600**
Area 2 - Crushing	0	0	0	**0**
Area 3 - Process	64	0	0	**64**
Area 4 - Concentrate Load Out	0	0	0	**0**
Area 5 - Tailings	168	0	0	**168**
Area 6 - Utilities [3]	0	0	0	**0**
Area 7 - Onsite Infrastructure [3]	0	0	0	**0**
Area 8 - Off-site Infrastructure	0	0	0	**0**
Sub-Total Directs	**831**	**0**	**0**	**831**
Area 9 - Indirect Costs [1]	-15.6	0	0	**-15.6**
Area 10 - Owner's Costs [1]	-0.7	0	0	**-0.7**
Sub-Total In-Directs	**-16.3**	**0**	**0**	**-16.3**
Total Directs + Indirects	**814**	**0**	**0**	**814**
Area 11 - Escalation	Excluded			
Area 11 - Contingency	0	0	0	0
Total Project Costs	**814**	**0**	**0**	**814**

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

The update of 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 Q1 2019 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 the estimate of 2019 feasibility study have been obtained from numerous sources, including:

- Data from the 2013 Dumont Ni and Co Project feasibility study
- feasibility level engineering design;
- mine schedules;
- topographical information obtained from site survey;
- geotechnical investigation;
- revised budgetary equipment quotes from multiple potential OEMs were asked again for 2019 update;
- budgetary unit costs obtained in FS 2013 from local contractors for civil, concrete, steel, electrical and mechanical works were indexed to 2019;
- data from recently completed similar studies and projects; and
- information provided by RNC, David Penswick and Wood.

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 David Penswick. Table 21-5 summarizes elements of the mining capital estimates for the initial, and sustaining phases of expenditure. Note that the strategy of employing accelerated mining rates and large, low grade stockpiles (discussed in Section 16.3.5) effectively decouples the mine plan from that of the mill. There is consequently only minimal investment in mining equipment during the mill expansion and for this reason, all expenditure following the initial period of pre-stripping has been classified as sustaining.

Table 21-5: Summary of Area 1- Mining - Capital Costs ($ M)

WBS Sub-Area	Initial (CAD $M)	Sustaining (CAD $M)	TOTAL (CAD $M)
100: Site Preparation	2	4	6
200: Contractor Stripping	42	0	42
300: Owner Stripping	74	0	74
400: Mining Equipment	130	460	590
500: Ancillary Equipment	17	8	25
550: Technology	18	14	32
600: Infrastructure	14	31	45
700: Trolley Assist	0	84	84
800: First Fills	1	-1	0
Sub-Total	**298**	**600**	**898**

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

Sources of the estimates presented in Table 21-5 are as follows:

Site Preparation – The estimate is based on clearing a total area of 2,700 hectares and an estimated unit rate of approximately $2,100/ha cleared. Thirty percent of the total area would be cleared during the construction period, with the remainder cleared in equal tranches over the following seven years.

Contractor Stripping – The estimate is based on the quantity of mining that would be allocated to the Contractor and unit rates that were calibrated based on estimates 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.

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 and Sandvik. Estimates included not only the cost of machines, but also the associated cost of transport to site and assembly.

Technology – Dumont will make extensive use of technologies that will allow for higher productivity and / or lower unit costs. Included in these technologies are the following:

- A Fleet Management System to automatically dispatch equipment in such a manner as to improve efficiency (i.e., minimal queuing) and effectiveness (e.g., dozers repairing roads where trucks are having to slow down)
- High Precision GPS (HPGPS) guidance and monitoring for drills, to ensure holes are correctly located (without requiring physical staking and measuring by surveyors) and to minimize re-drilling.
- HPGPS guidance and monitoring for excavators and shovels, to minimize dilution and ensure benches and mined to grade.
- Payload monitoring for excavators, shovels and trucks to ensure that trucks are optimally loaded. Note that every 2.5 t increase or decrease in average truck payload has a 1% impact on overall project NPV.
- Shovel and excavator dipper tooth monitoring, to avoid tramp steel being delivered to the primary crusher
- Tire temperature and pressure monitoring for haul trucks, to maximise tire life.

- HPGPS guidance and monitoring for dozers and graders, to ensure roads are maintained on grade.

Infrastructure – The key element of infrastructure will be a maintenance shop for the fleet of mining equipment (the "truck shop"). This 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 formula of 1 workshop bay per five trucks Six bays will be constructed for the initial phase, while the truck shop will ultimately reach 12 bays. The cost of bays is based on the requirement for 290 t class haul trucks. Other items included under infrastructure are:

- The "fuel farm", whose size has been based on diesel consumption as estimated by the zero-based model.
- Dewatering pumps, with additional pumps added as the depth of pit and associated head increases
- The roadstone crusher, which will be installed prior to start-up of the trolley system in Yr 3
- Electrical substations and associated equipment to power the electrical fleet.

The cost of the magazine and facilities for manufacture and storage of explosives will be borne by the explosives supplier and recovered as an operating expense over a period of five years once the operation is generating cash flow.

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 process 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 of the TSF. This provision height is sufficient to store approximately the first two years of tailings production.

The capacity of the TSF would be increased progressively through continual lifting of the dam walls. When feed to the mill switches from the pit to ore stockpiles, tailings will be deposited in the pit.

21.3.4 On-Site Infrastructure

The following buildings will be built:

- main administration building with medical centre and training room;
- mine dry;
- security office;
- security gatehouse; and

- Assay laboratory (Cost excluded of FS 2019 Capex - By 3rd party (SGS) and in OPEX).

In addition, the process plant buildings listed below will be built. The capital cost for these buildings is included in the process plant area of the 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 (site preparation only); 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

For a rail product transport alternative, a 5.5 km spur off the existing CN rail line to reach the storage product area of the processing plant, will be required. Total trackage requirements will be 6.0 kms including interchange tracks and a fuel delivery spur off the truck maintenance shops.

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 by engineering. Rates were obtained from historical local contractors escalated rates. For the updated FS 2019, these costs were indexed to 2019. 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. Initially, a portable plant will be operated by the Mining Contractor. Starting in Year 3, aggregate will be supplied by the Owner’s roadstone crushing plant.

21.4.1.2 Labour rates

Labour rates have been built-up from first principles for different trades (welders, boilermakers, roofers, pipefitters, millwrights, store person, crane operator, etc.). These rates have been based on the Quebec labour collective agreement (Heavy industrial sector – 2018-12-30) which can be found on the website, http://www.ccq.org, and the Guide for Employers 2018 - 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 – Heavy 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 $80.54/hr.

Contractor indirect costs for structural, mechanical, piping, electrical and instrumentation have been developed for the 2019 feasibility study 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 $170.25/h which includes the SMP base crew rate of $80.54/h and the addition of costs associated with the items listed above. This detailed rate calculation results was confirmed by local SMP contractors for the FS 2013. The same approach was used for the updated FS 2019. The electrical and instrumentation (E&I) rate is slightly less, at $153/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 94% 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

No duties were included for the updated FS 2019.

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. For the updated FS 2019, these costs are using the same % ratio as of FS 2013.
- Heavy lift construction cranes. These represent cranes over and above what the construction contractor provides. These are cranes greater than 100 tonne capacity. For the updated FS 2019, these costs are using the same % ratio as of FS 2013.

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, in FS 2013. A detailed assessment of consultants and project general expenses are also included in the EPCM costs of the FS 2013. For the updated FS 2019, these costs were indexed to 2019. 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 percentage 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

Where spares were not priced in the quotation, a percentage of the equipment cost was applied.

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 commissioning construction support 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. For the updated FS 2019, these costs were indexed to 2019.

Commissioning support from vendor was provided with majority of bids. Where not received in the quotation, a percentage of the equipment cost was applied.

21.4.2.7 First Fills

An estimate for first fills for the following reagents has been included in FS 2013: 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 was also included in FS 2013.

An allowance has also been made for oils, lubricants, hydraulics, and greases in FS 2013.

For the updated FS 2019, these costs were indexed to 2019.

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 in FS 2013 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. For the updated FS 2019, these costs were indexed to 2019.

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);
- Capitalized process operating costs (also to the end of Month 1 of mill production);
- 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 in FS 2013. 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. FS 2019 used the same % as the FS 2013.

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 9.5% for the initial capital cost, and 12.7% 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 were in FS 2013:

- 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%

For the updated FS 2019, the same methodology was used, with the modification for TFS from 10% to 8%.

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 $817 M over the mine life, which includes a credit of $16.4 M for the release of first fills and spares 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).
- Expansion of the workshop that will be required as the fleet expands. The initial workshop of 6 bays will be expanded to 12 bays during the expansion.
- Ongoing expansion of the TSF.
- General plant and infrastructure replacements, that are expected to total $63.5 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 Q1 2019 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 key consumables from supplier quotations, regional benchmark costs for other commodities and labour rates for the Abitibi region, as determined from a salary survey.

- Operating costs for the concentrator were based on rates of consumption for reagents and other consumables determined from metallurgical test work and a labour structure that is appropriate for the current flowsheet.
- The operating cost estimate includes those costs associated with operating the TSF.
- G&A costs were based on the level of support required for the operation, including an organizational chart provided by the Owner.
- Costs for realization of nickel were based on the commercial terms discussed in Section 18, and the scheduled production of concentrate.
- Processing operating costs were typically calculated exclusive of variability from design throughputs (e.g., neglects ramp-up period, etc.). One notable exception is reagent consumption which was increased in the first year of operation to account for upsets during start-up and learning-curve period.

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 Yr1-7	105 kt/d Yr8-19	LOM Average
Mine	$/t ore milled	$7.11	$5.46	$3.82
Process	$/t ore	$5.31	$5.20	$5.20
G&A	$/t ore	$0.97	$0.53	$0.54
Site Costs	$/t ore	$13.39	$11.19	$9.56
Site Costs	US$/t ore	US$10.04	US$8.40	US$7.17
Site Costs	US$/lb	US$2.83	$3.14	$3.07
Realization	US$/lb	US$0.15	$0.16	$0.16
C1 Cash Cost[1]	US$/lb	US$2.98	$3.30	$3.22

Note 1. The Base Case design assumes roasting of concentrate, which will not produce payable by-product metals. An alternate case that considers treatment and refining with associated payable production of Co and PGEs is discussed in Section 24.

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 Q1 2019 have been escalated to this date using average Canadian producer price index (PPI)
- US$ denominated prices for goods and services obtained prior to the cost basis date of Q1 2019 have been escalated to this date using average US producer price index (PPI).
- 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 Management 360.
- Based on discussions with Hydro-Quebec, it has been assumed that the project would qualify for the “L Tariff”, including discounts for sustainability. The forecast price varies over the life of project as a function of both the discount and demand, with the weighted average over the life of project expected to average $47.37/MWh.

  The forecast long-term diesel price of $0.89/litre is based on forecast long-term oil prices of US$60/bbl and a C$ F/X rate of US$0.75.

21.5.3 Mining Operating Costs

A summary of mining costs by function and category is provided in Table 21-7 and Table 21-8, respectively. Note that these tables exclude $141m of expenditure on mining activities related to construction of the TSF that have been entirely capitalized. Also excluded is $11m of expenditure on mining activities related to reclamation of the TSF and waste dumps that has been included in the Closure Estimate.

It should be noted that the forecast mining costs for Dumont are low relative to some existing large scale Canadian open-pit hard rock mines, but can be explained by the following factors:

- The mechanical properties of rock at Dumont. These include very low abrasion indices, which will result in lower consumption of ground engaging tools (GET). It will also be possible to blast Dumont rock with a relatively low powder factor that will allow for widely spaced blast holes, leading to low drilling and blasting costs.
- The geometry of mineralization allows for highly productive, bulk mining. This is in contrast to gold mines where irregular mineralization necessitates selective mining, with more units of smaller capacity.
- The use of trolley-assist will reduce the cost of energy and haul truck maintenance, along with improving the productivity of the fleet – leading to fewer drivers.
- Dumont will invest significantly in technologies aimed at maximizing productivity and minimizing cost. These have been outlined previously, in the preceding section.

Also, to be noted is the operating cost estimate assumes that steady-state levels of efficiency will not be achieved from the outset but will only be achieved following a 36 month learning curve. The initial level of efficiency is assumed to be at 50% and increase steadily to the steady-state. For example, the 90t haul trucks are assumed to achieve an average tire life of 3,000 hours initially. During their first full year of operation, during the pre-strip period. During the first year of operation they will average 4,156 hrs, rising to 5,156 in the second year and 5,842 in the third year before the steady-state rate of 6,000 hrs.

Table 21-7: Mining Operating Cost Summary – By Function

Activity	units	Total	Capitalized Pre-Strip	Capitalized TSF & Reclamation	Expensed	% of Total
Contractor	$M	44	42	0	2	0.0%
Owner by Process:						
Production Drilling	$M	147	5	0	142	3.6%
Production Blasting	$M	384	10	0	374	9.5%
Pre-Split Drilling & Blasting	$M	26	0	0	26	0.7%
Loading	$M	379	5	0	375	9.5%
Hauling	$M	1,934	20	0	1,914	48.8%
Low-Grade Ore Rehandle	$M	301	0	0	301	7.7%
TSF Construction & Reclamation	$M	136	0	136	0	0.0%
Roadstone	$M	119	0	0	119	3.0%
Support and Auxiliary Equipment	$M	138	6	0	133	3.4%

Maintenance Labour	$M	355	13	15	326	8.3%
Management, Technical & Admin Total	$M	230	16	0	214	5.5%
Total	$ M	**4,192**	**116**	**152**	**3,925**	**100.0%**
$/t material		2.02	0.06	0.07	1.89	
$/t ore		4.08	0.11	0.15	3.82	

Table 21-8: Mining Operating Cost Summary – By Category

	units	Total	Capitalized Pre-Strip	Capitalized TSF & Reclamation	Expensed	% of Total
Contractor	$M	44	42	0	2	0.0%
Owner by Area:						
Labour cost	$M	1,002	38	64	901	23.0%
Consumables	$M	664	8	12	644	16.4%
Maintenance	$M	1,270	10	39	1,220	31.1%
Diesel	$M	1,056	15	37	1,004	25.6%
Power	$M	127	0	0	127	3.2%
Other	$M	30	3	0	27	0.7%
Total	**$M**	**4,192**	**116**	**152**	**3,925**	**100.0%**
Unit Rate	$/t rock	2.02	0.06	0.07	1.89	
	$/t ore	4.08	0.11	0.15	3.82	

Contractor mining represents 1% of total costs. The majority of the contractor scope of work includes removing all clay overlying the deposit during the initial period of pre-stripping. The contractor will also operate a crusher used to produce aggregate for construction, roadstone and blast hole stemming prior to commissioning of the Owner's roadstone crusher in Year 3. Contractor mining costs were based on a competitive tendering process that 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. The Base Case cost estimates includes use of trolley-assist to reduce energy costs and improve truck productivity. Without the use of trolley-assist, haulage costs would increase by approximately $440m or $0.21/tonne mined.

Table 21-8 indicates that diesel is the largest single element of operating costs, with the 1.15 Mm3 consumed representing 26% of total expenditure. Without the use of trolley-assist, consumption would increase by 450 Mm^3 or 38%.

Key assumptions regarding the cost of equipment maintenance are based on budgetary quotations provided by OEMs.

The workforce averages 318 full-time equivalent positions (FTE) over the life of mine. The maximum and average workforce during pit operations are 602 FTE and 441 FTE, respectively. Following depletion of the Main Pit in Year 19, the workforce averages 110 FTE during the remaining 11 years of Mining Phase 8 and reclaiming stockpiles Note that these totals include personnel allocated to TSF construction and reclamation activities.

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 3.0% of owner mining costs. This low contribution reflects the attractive 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 Management 360;
- 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;
- Costs for the 3rd party operated 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;

- 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 by a 3rd party. Summarized costs provided in Table 21-9 and Table 21-10 include an allowance for the full labour complement to be brought in 3 months prior to commercial start-up. (included in Owner's costs in the capital estimate). Also included is a six-month ramp up to full production for both mill lines and 12-month 'learning curve' of higher reagent consumption. The estimated overall operating cost for the initial processing plant is $5.31/t of ore milled, reducing to $5.18/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**	**122**				
Labour	$ M	66	7	58	0.48	9
Power	$ M	207	0	207	1.69	32
Maintenance Materials	$ M	77	0	77	0.63	12
Reagents and Consumables	$ M	287	0	287	2.35	44
Miscellaneous	$ M	19	0	19	0.15	3
Total	**$ M**	**656**	**7**	**648**	**5.31**	**100**

Table 21-10: Process Plant Cost Summary– Expanded Phase at 105 kt/d

Area	units	Total	Capitalized	Expensed	$/tonne	M$/annum
Ore Milled	**Mt**	**906**				
Labour	$ M	311	0	311	0.34	13
Power	$ M	1,726	0	1,726	1.91	73
Maintenance Materials	$ M	486	0	486	0.54	20
Reagents and Consumables	$ M	2,063	0	2,063	2.28	87
Miscellaneous	$ M	112	0	112	0.12	5
Total	**$ M**	**4,697**	**0**	**4,697**	**5.18**	**198**

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 and Table 21-12 are almost entirely fixed in nature, with the result that unit costs at 105 kt/d fall to half the rate for the initial 52.5 kt/d scope of project.

Table 21-11: G&A Cost Summary– by Element

	52.5 ktpd		105 ktpd	
Element	**$M pa**	**$/t**	**$M pa**	**$/t**
Labour	4.1	0.22	4.1	0.11
Consumables	0.6	0.03	0.7	0.02
Maintenance	0.1	0.00	0.1	0.00
Power	0.2	0.01	0.4	0.01
Diesel	0.1	0.01	0.2	0.00
Other	13.1	0.70	13.1	0.34
Total	**18.3**	**0.97**	**18.5**	**0.48**

Table 21-12: G&A Cost Summary– by Area

	52.5 ktpd		105 ktpd	
Area	**$M pa**	**$/t**	**$M pa**	**$/t**
Labour	4.1	0.22	4.1	0.11
General Management	4.8	0.26	4.7	0.12
Human Resources	1.5	0.08	1.4	0.04
Admin & IT	3.9	0.21	4.0	0.11
Environmental	1.3	0.07	1.3	0.03
Loss Control and HSEC	1.9	0.10	1.9	0.05
Shipping / Purchasing	0.5	0.03	0.5	0.01
Mobile Equipment	0.3	0.02	0.4	0.01
Total	**18.3**	**0.97**	**18.5**	**0.48**

21.5.6 Contingency

Contingency is not included in the operating cost estimate.

22 ECONOMIC ANALYSIS

22.1 Summary

This economic analysis of the Dumont Feasibility Study focuses on the base case, which includes use of trolley-assisted truck haulage in the mine but does not include use of autonomous equipment. The base case also assumes nominal process plant throughput of 52.5 ktpd initially. A project to double capacity would start in Year 6 and process the first incremental ore 18 months later. It has been assumed that all concentrate produced would be sold to third parties for roasting at a facility located outside of the province of Quebec. With roasting, no revenues would be realized from by-product cobalt or platinum group elements (PGE). The base case also does not include the potential benefits from magnetite as a by-product.

Salient metrics for this base case are presented in Table 22-1.

Table 22-1: Feasibility Study Summary Metrics

	Unit	C$ Basis	US$ Basis
Ore Mined	Mt	1,028	1,028
Payable Ni	Mlbs	2,402	2,402
Gross Revenue	$/t ore	$25.60	$19.20
Realization [1]	$/t ore	$1.94	$1.45
Net Smelter Return	$/t ore	$23.66	$17.75
Site Operating Costs	$/t ore	$9.56	$7.17
C1 Costs [2]	$/lb Ni	$4.30	$3.22
EBITDA	$/t ore	$13.23	$9.92
Peak Funding Requirement [3]	$M	$1,386	$1,039
Total Investment [4]	$M	$3,047	$2,285
AISC [5]	$/lb Ni	$5.07	$3.80
Total Costs [6]	$/lb Ni	$5.94	$4.46
Pre-Tax $NPV_{8\%}$	$M	$6,725	$5,043
Post-Tax $NPV_{8\%}$	**$M**	**$1,226**	**$920**
Post-Tax IRR		**15.4%**	**15.4%**

Notes: **1.** Realization includes the cost of concentrate transport and implied costs of metal deductions, **2.** C1 Costs include Realization and Site Operating Expenditures, **3.** Peak Funding represents the cumulative unlevered investment prior to generation of positive cash flow, **4.** Total Investment includes all Capital and Closure expenses, **5.** All In Sustaining Costs include C1 Costs, Royalties, IBA, Sustaining Capital and Closure expenses, **6.** Total Costs include AISC, Initial Capital and Expansion Capital

22.2 Assumptions

Key price assumptions included in the base case analysis are as follows:

- The forecast long term price for Nickel of US$7.75/lb is based on a market studies performed by the consulting groups CRU Strategies and Red Door Consulting.
- The forecast long term US$ exchange rate of US$0.75 is based on consensus projections of North American equity analysts.

- The forecast long term price for oil of US$60/bbl has been taken from consensus projections of North American equity analysts. Based on the current relationship between the prices of oil and diesel in the Abitibi, this oil price translates to a delivered cost of diesel at site of $0.89/litre.
- The weighted average LOM electricity prices is forecast to be $47/MWh, which is based on the current L-rate tariff for Quebec and Dumont's expected demand profile. This weighted average price also accounts for the rebate of up to 20% for the period to December 2027, for which Dumont would qualify.
- The forecast long term price for acid is $114/t, based on a market study performed by the consulting group CRU Strategies.

Key assumptions related to production 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 43.2%, which takes into account a ramp-up of 6 months to achieve nameplate performance.
- Roaster deductions would be 8.5% of nickel contained in concentrate, for payability of 91.5%. This deduction would cover the cost of roasting, with no additional treatment or refining charge.

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.

The calculated royalty payments include the assumption that the non-overlapping Coyle-Roby royalty of 2% and Marbaw royalty of 3% NSR royalties will be bought down to 1% and 1.5%, respectively, as is provided for in the contracts. The payment calculation also assumes that the 0.8% NSR royalty owned by Ressources Québec will be bought out while the 1.75% NSR royalty owned by Red Kite will be bought down to 1.375%. The LOM weighted average royalty rate, post buy-downs and buy-back, will be 2.77% of NSR.

The evaluation also includes the Impacts Benefit Agreement (IBA) that has been negotiated with the local First Nation.

Results were calculated on a pre-tax and post-tax basis, based on the current fiscal regime.

22.3 Base Case Results

The total life of project can be subdivided into the following periods:

- Construction for a period of 24 months
- Phase 1 production at a concentrator throughput rate of 52.5 kt/d for 78 months (6.5 years)
- Phase 2 production at a concentrator throughput of 105 kt/d and the open pit being operational, for 201 months (16.75 years)

- Phase 2 production at a concentrator throughput of 105 kt/d following the completion of open pit mining, for an additional 81 months (6.75 years)

Summary metrics for each of these periods are presented in Table 22-2. It can be seen that the cumulative NPV to the end of pit life is US$806 M or 88% of the project total. The remaining 12% of project NPV ($112 M) is realized during the period that the only source of ore is the low-grade stockpile, with the benefits of lower costs offsetting lower grade and recovery.

Table 22-2: Summary of Economic Metrics by Period

Item	Construct	Phase 1 Yr1-7	Phase 2 Yr8-19	Phase 2 Yr20-30	Total
Ore Mined (Mt)	13	252	732	31	1,028
Total Mined (Mt)	42	614	1,361	63	2,080
Stripping Ratio (waste: ore)	2.33	1.43	0.86	1.05	1.02
Ore Milled (Mt)	0	122	477	429	1,028
Grade (% Ni)	0.00	0.33	0.28	0.23	0.27
Concentrator Recovery (% of Ni)	0.0	52.6	47.1	34.1	43.2
Payable Ni (Mlbs)	0	474	1,392	759	2,625
C1 Cash Costs (US$/lb Ni)	$0.00	$2.98	$3.30	$3.25	$3.22
Initial Capital (US$m)	$1,018	$0	$0	$0	$1,018
Expansion Capital (US$m)	$0	$601	$0	$0	$601
Total Investment (US$m)[1]	$1,063	$941	$251	$2	$2,256
Post-Tax NPV $_{8\%}$ (M)	($922)	$449	$1,101	$291	$920
Post-Tax IRR					**15.4%**

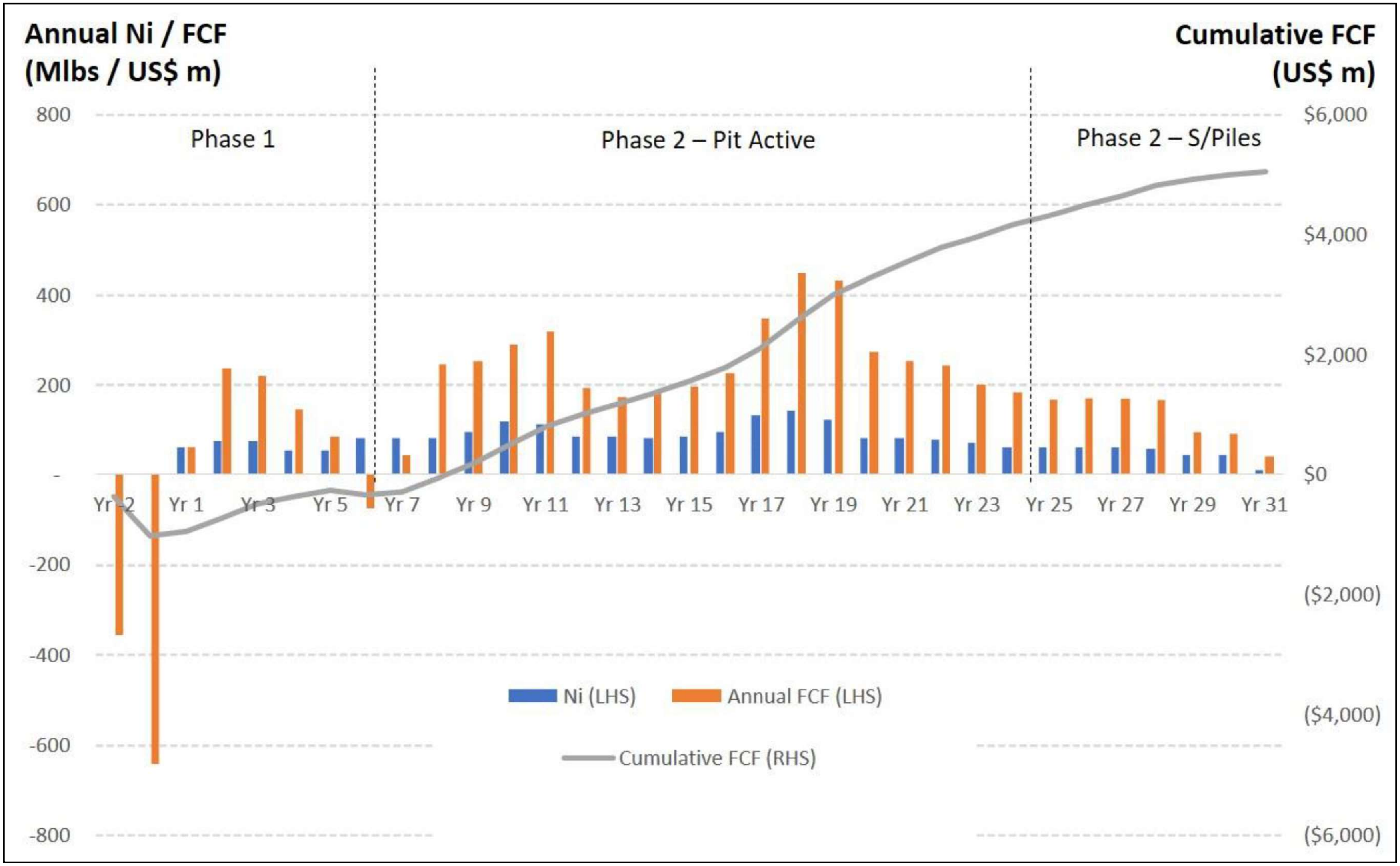
Notes: 1. Total investment includes expenditures of US$26m for Closure activities

Figure 22-1 provides a life of project graph of cash flow. The following information is highlighted:

- The peak funding requirement of US$1,039 M is reached three months after the start-up of commercial operations. Note that the operation is forecast to be break-even on an operating cash flow basis during the first quarter of operation and free cash flow positive from the second quarter of operation. During the five years prior to the commencement of capital expenditure for the initial expansion phase of operation at 52.5 ktpd, annual post-tax free cash flow averages US$ 149m.
- 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 an assumed 5 year maturity for the initial debt package, during which time cumulative free cash flow equates to 75% of the total capital requirement. Approximately 66% of the total investment required for the expansion period (including non-expansion sustaining capital) would be generated from internal free cash flows, with additional funding of approximately US$202 M required. Following expansion to 105 kt/d, annual post-tax free cash flow during the period that the Main Pit is operational averages approximately US$274 M.
- Payback of all invested capital (including the expansion) is achieved approximately eight years after initial start-up.
- For the final 11 years of the project life when mill feed is either primarily or entirely sourced from low grade stockpiles, annual free cash flow averages US$ 180M.

From the start-up of mill operations, free cash flow averages $201m per annum. Table 22-4 provides detailed metrics for the life of mine cash flow, with time periods presented as years after start-up.

Figure 22-1: Life of Project Cash Flow



Source: RNC.

Table 22-3: Detailed Economic Metrics

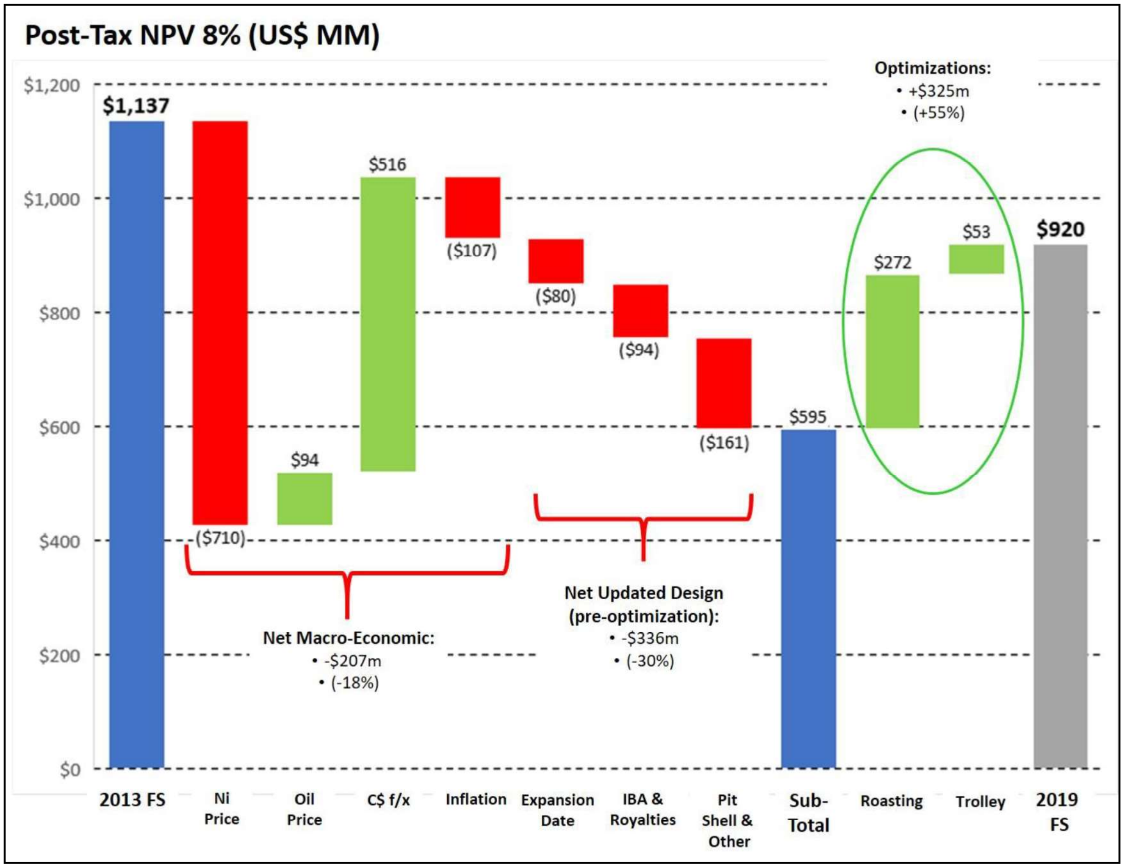
Item	Units	Total	Pre-Prod'n	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11 - 20	Yr21-31
Ore Processed	Mt	1,028	0	17	19	19	19	19	19	26	38	38	38	383	391
Ni Contained in Concentrate	Mlb	2,625	0	64	80	81	58	60	88	90	90	102	128	1,111	672
Payable Ni[1]	Mlb	2,402	0	59	73	74	53	55	81	82	82	93	117	1,017	615
Gross Revenue	US$ M	$18,617	$0	$456	$567	$576	$413	$427	$626	$635	$636	$724	$908	$7,881	$4,767
Realization[1]	US$ M	$375	$0	$9	$10	$12	$8	$9	$12	$12	$14	$15	$18	$157	$98
Net Smelter Return	**US$ M**	**$18,243**	**$0**	**$448**	**$557**	**$564**	**$406**	**$417**	**$614**	**$623**	**$623**	**$709**	**$890**	**$7,724**	**$4,669**
Mining	US$ M	$2,943	$0	$62	$102	$82	$96	$127	$122	$123	$145	$159	$168	$1,445	$312
Processing	US$ M	$4,009	$0	$70	$74	$74	$75	$77	$78	$107	$149	$149	$149	$1,493	$1,514
G&A	US$ M	$419	$0	$13	$14	$13	$14	$14	$14	$14	$15	$16	$16	$148	$128
Total Operating Costs	**US$ M**	**$7,371**	**$0**	**$145**	**$189**	**$169**	**$185**	**$218**	**$214**	**$245**	**$309**	**$324**	**$333**	**$3,086**	**$1,954**
C1 Cash Costs	**US$ / lb**	**$3.22**	**$0.00**	**$2.61**	**$2.73**	**$2.44**	**$3.61**	**$4.13**	**$2.80**	**$3.14**	**$3.93**	**$3.63**	**$2.99**	**$3.19**	**$3.34**
Initial Capital[2]	US$ M	$968	$968	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Expansion Capital[2]	US$ M	$601	$0	$0	$0	$0	$0	$0	$416	$185	$0	$0	$0	$0	$0
Sustaining Capital	US$ M	$716	$33	$149	$60	$86	$34	$77	-$15	$84	$2	$26	$72	$154	-$44
Working Capital and Closure	US$ M	$26	$33	$71	-$15	-$4	-$17	$1	-$55	$69	-$6	$12	$12	-$31	-$45
Total Investment [2]	**US$ M**	**$2,285**	**$1,000**	**$149**	**$60**	**$86**	**$34**	**$77**	**$401**	**$268**	**$2**	**$26**	**$72**	**$154**	**-$44**
Royalties[3] and IBA	US$ M	$736	$0	$81	$16	$17	$12	$13	$20	$18	$23	$27	$34	$295	$179
Cash Federal Income Taxes	US$ M	$1,053	$0	$0	$22	$31	$10	$8	$23	$19	$17	$30	$53	$499	$342
Cash Provincial Income Taxes	US$ M	$836	$0	$0	$17	$24	$8	$7	$18	$15	$13	$24	$42	$396	$271
Cash Provincial Mining Tax	US$ M	$919	$0	$13	$17	$18	$11	$11	$11	$13	$15	$32	$64	$509	$207
Pre-Tax Free Cash Flow	$ M	$7,851	-$1,000	$72	$291	$291	$175	$109	-$21	$91	$289	$332	$451	$4,189	$2,581
Post-Tax Free Cash Flow [4]	**$ M**	**$5,043**	**-$1,000**	**$59**	**$235**	**$219**	**$146**	**$84**	**-$73**	**$44**	**$244**	**$247**	**$292**	**$2,785**	**$1,761**

Notes: 1. Roaster deductions for payability have been included under payable Ni and are excluded from realization. Assuming a Roaster recovery of 98% leads to an implied cost for treatment of 6.5% or US$0.50/lb Ni at $7.75/lb Ni price. 2. The timing and quantum of capital expenditures include an estimated US$29M associated with leasing of mine fleet. 3. Includes buy-down and buy-back of Royalties totalling US$63m 4. Leasing of fleet has a ($18M) impact on NPV 0% and a $8m impact of NPV 8%. **Source:** RNC.

22.4 Reconciliation to Revised Pre-Feasibility Study

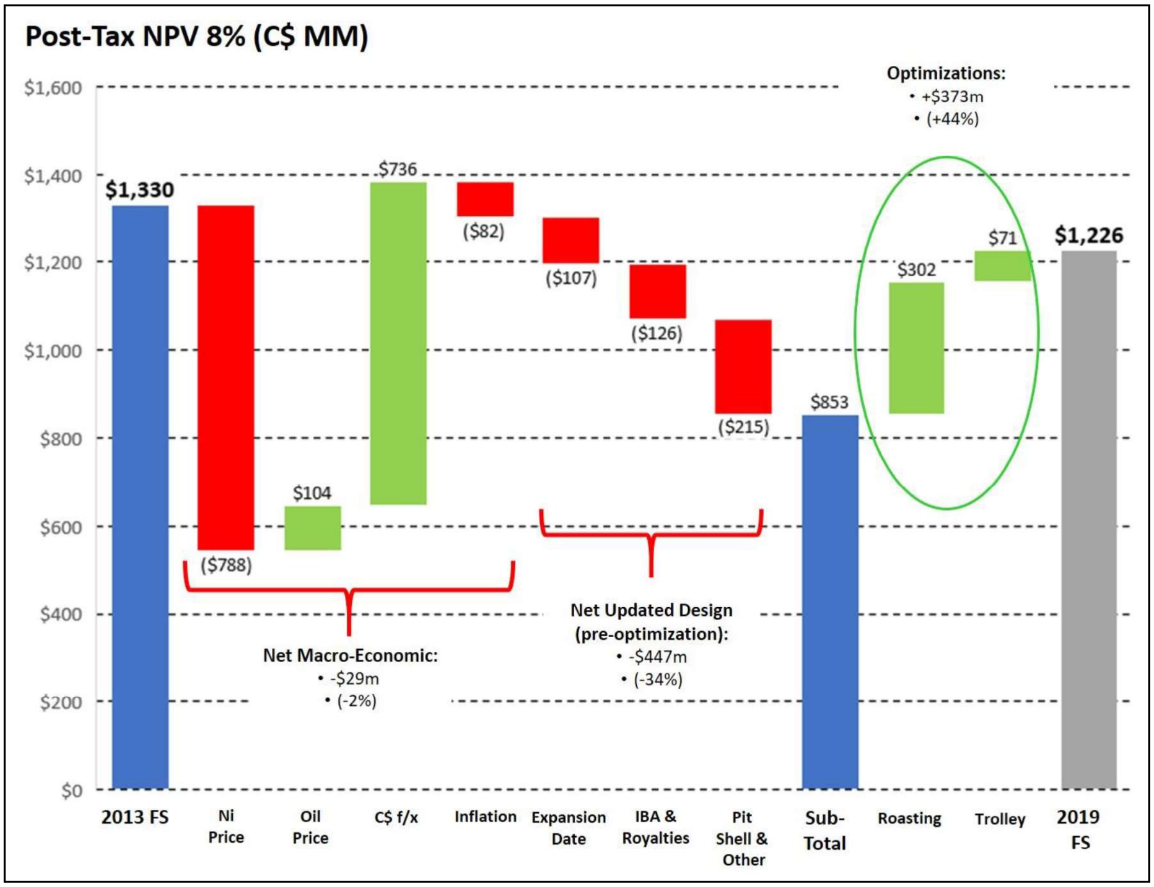
Figure 22-2 and Figure 22-3 provide waterfall graphs that illustrate changes to project NPV, since the revised Prefeasibility Study (PFS) in US$ and C$ terms, respectively.

Figure 22-2: Changes to Project NPV (US$ 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:

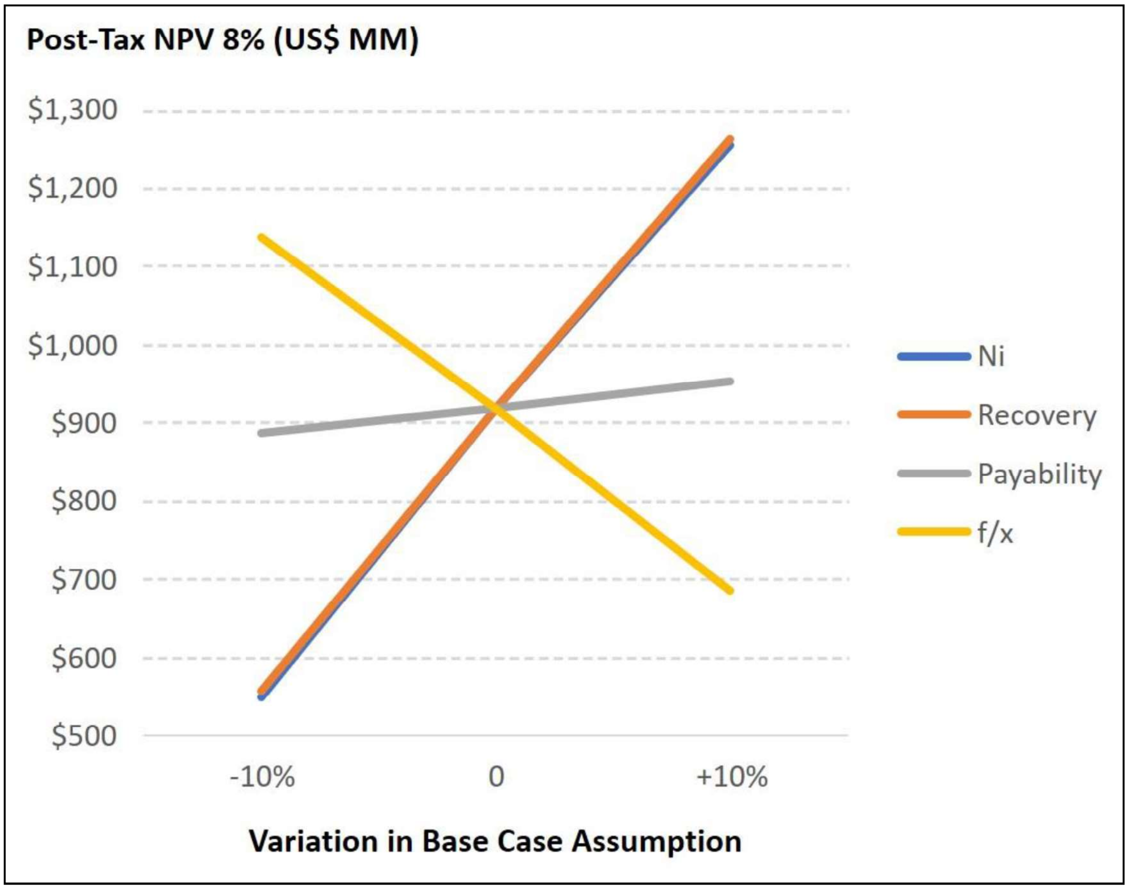
- The reduction in forecast Ni price from $9.00/lb in 2013 to $7.75 currently has the largest single impact on overall project economics, at 62% and 59% of the 2013 NPV in US$ and C$ terms, respectively.
- The impact of lower Ni prices is exacerbated by inflation over the intervening time, though partially offset by more favourable prices for oil and the Canadian dollar exchange rate. The net impact of changing macro-economic parameters is a reduction of the 2013 NPV by 18% and 2%, in US$ and C$ terms respectively.
- Subsequent to the 2013 FS, an IBA was negotiated with the local First Nation and an additional 0.75% NSR royalty sold to Red Kite. The current plan also defers the date of expansion to 105 ktpd by 2 years, until midway through the 7th year of production. The current pit shell contains approximately 17% less total material, and is mined at lower production rates, which contributes to higher unit costs. The net impact of all these changes is to reduce the US$ NPV by 30% (34% for the C$ NPV).
- The current design has been enhanced by the decision to treat concentrate by roasting rather than conventional smelting. The impact of reduced treatment charges, net of the loss of by-product revenue, is an increase to the 2019 unoptimized design by 46% in US$ terms or 35% in C$ terms.

- Another key change is the incorporation of trolley-assisted truck haulage to improve trucking productivity and energy costs. This feature increases the US$ NPV by 9%, or 8% for the C$ NPV.

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 asymmetric, with the percentage increase in NPV for higher revenue approximately 5 – 10% lower than 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 also produces asymmetric outcomes, with the upside from a 10% decrease in the exchange rate (a 25% improvement in NPV) is 7% less than the reduction in NPV resulting from a 10% strengthening in exchange rate. Payables represents a ±10% change to the roaster deduction (base case assumption is 8.5%), with a 10% change resulting in a symmetric variation in NPV of 4%.

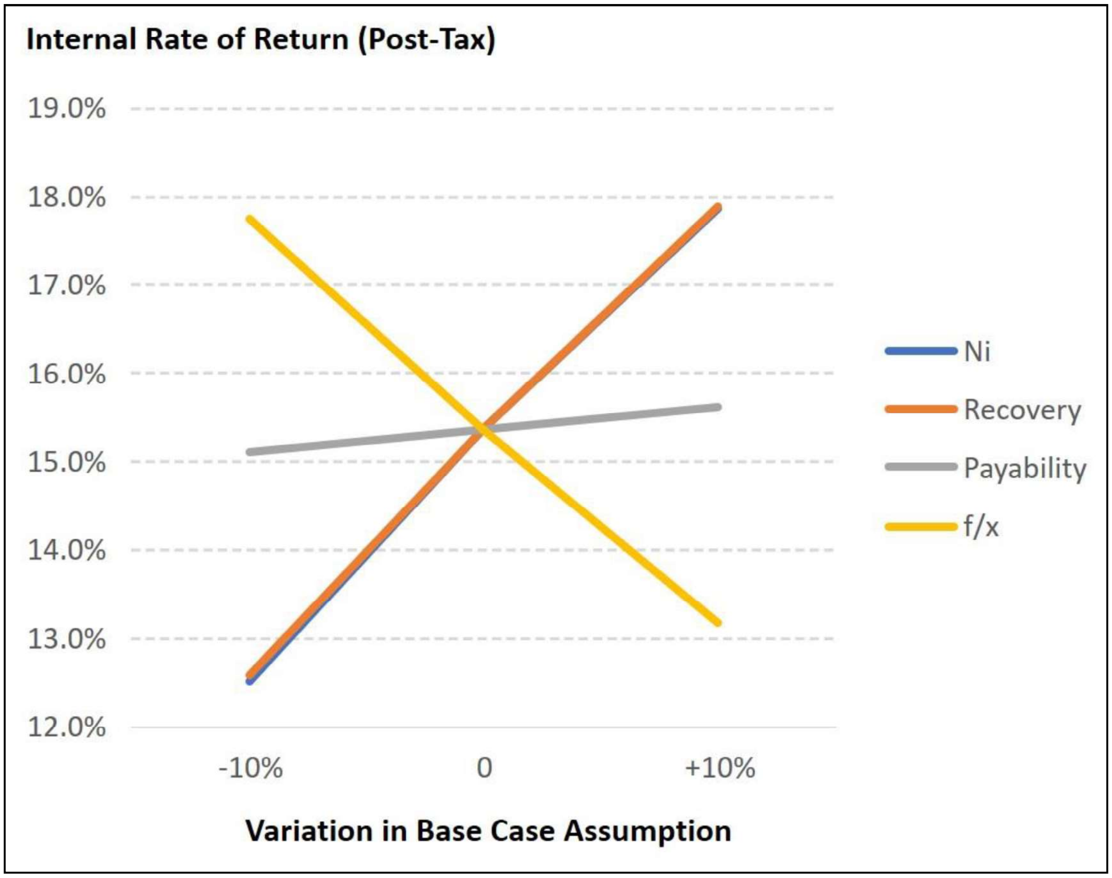
Figure 22-4: Sensitivity of Project NPV to Variation in key Assumptions



Source: RNC.

Figure 22-5 illustrates a similar relationship for the sensitivity of IRR to changes in the key parameters.

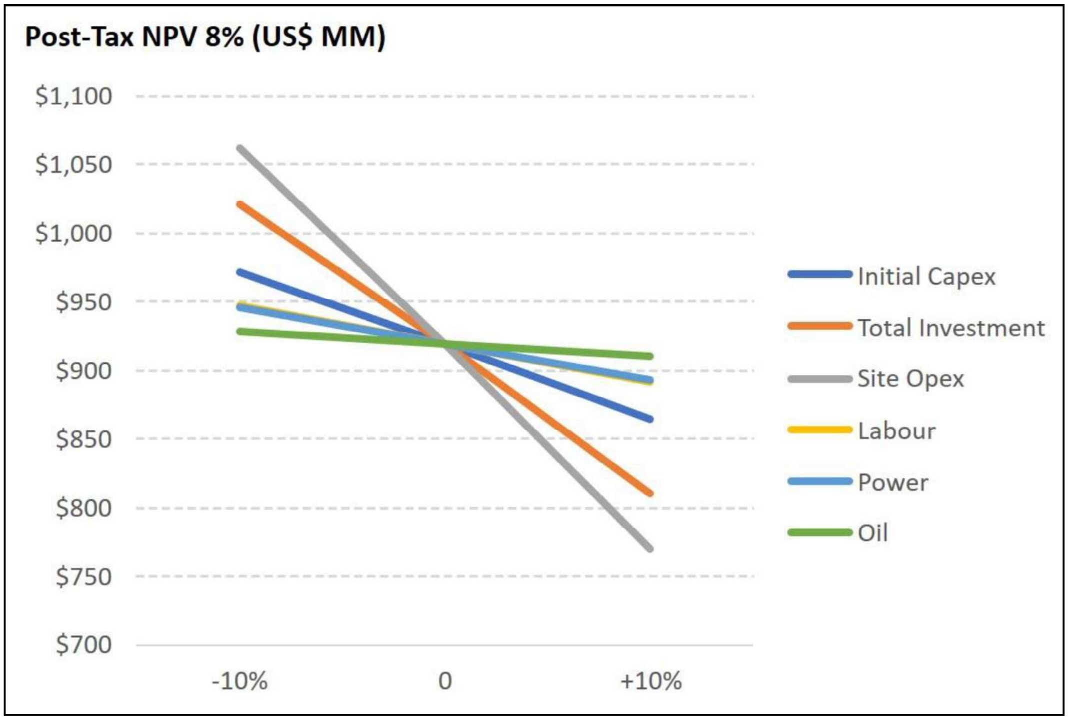
Figure 22-5: Sensitivity of Project IRR to Variation in Key Assumptions



Source: RNC.

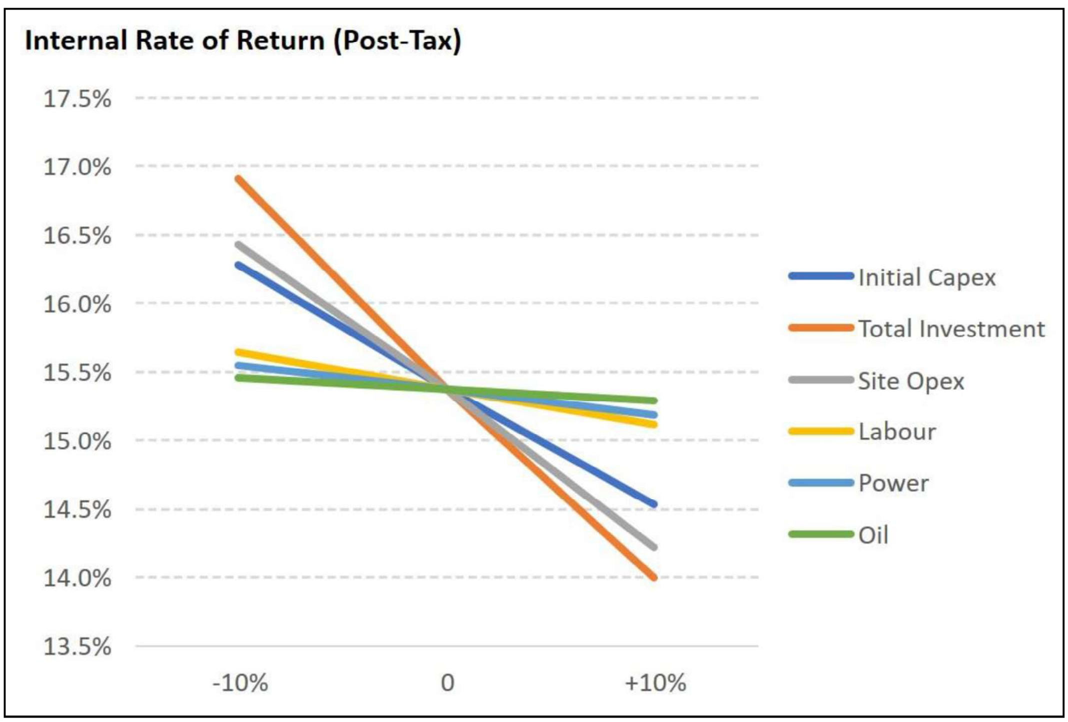
The project returns are less sensitive to the variation of other parameters – with a 10% variation in site operating costs having a 16% impact on project NPV. With the staged development plan, returns are less sensitive to capital costs and a 10% change in total capital cost has a lower impact, at only 12% of NPV, while the impact of a similar variation in initial capital is half as much at 6%. The three largest single elements of operating costs are Electricity (21% of total operating expenditures), Labour (16% of total operating expenditures) and Diesel (11% of total operating expenditures). Returns are marginally more sensitive to the cost of labour than that of electricity, which reflects the respective profiles in complement and power consumption. Returns are relatively insensitive to variation in the diesel price – in part due to the use of trolley assist to minimize diesel consumption.

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.

Table 22-4 to Table 22-7 tabulate the sensitivity of the project NPV, IRR, Cash Flow and Costs to the same parameters. Note that in all tables, Ni payables are expressed as the variance in roaster deductions (±10% = 0.85 percentage points from 91.5% to 92.35%).

The post-tax break-even Ni prices are as follows:

- $NPV_{0\%}$ = US$ 4.38/lb
- $NPV_{8\%}$ = incentive Ni price (NPV = $0) is US$5.86/lb.

Table 22-4: Sensitivity of Project NPV 8%

		Post Tax NPV			Pre-Tax NPV		
Discount Rate = 8%	**Units**	**-10%**	**0%**	**10%**	**-10%**	**0%**	**10%**
Ni Price	US$ M	551	920	1,255	1,125	1,713	2,275
Recovery	US$ M	559	920	1,264	1,136	1,713	2,289
Payability	US$ M	886	920	953	1,658	1,713	1,768
C$ F/X	US$ M	1,138	920	685	2,069	1,713	1,357
Initial Capital	US$ M	972	920	864	1,793	1,713	1,632
Total Investment	US$ M	1,022	920	810	1,870	1,713	1,556
Site Operating Costs	US$ M	1,063	920	769	1,953	1,713	1,473
Power	US$ M	946	920	893	1,757	1,713	1,669
Oil	US$ M	929	920	910	1,728	1,713	1,698
Labour	US$ M	947	920	892	1,757	1,713	1,669

Table 22-5: Sensitivity of Project IRR

		Post-Tax IRR (%)			Pre-Tax IRR (%)		
IRR		**-10%**	**0%**	**10%**	**-10%**	**0%**	**10%**
Ni Price	US$ M	12.5	15.4	17.9	16.0	19.9	23.6
Recovery	US$ M	12.6	15.4	17.9	16.1	19.9	23.7
Payability	US$ M	15.1	15.4	15.6	19.6	19.9	20.3
C$ F/X	US$ M	17.7	15.4	13.2	23.5	19.9	16.9
Initial Capital	US$ M	16.3	15.4	14.5	21.4	19.9	18.7
Total Investment	US$ M	16.9	15.4	14.0	22.3	19.9	18.0
Site Operating Costs	US$ M	16.4	15.4	14.2	21.5	19.9	18.4
Power	US$ M	15.6	15.4	15.2	20.2	19.9	19.7
Oil	US$ M	15.5	15.4	15.3	20.1	19.9	19.8
Labour	US$ M	15.6	15.4	15.1	20.3	19.9	19.6

Table 22-6: Sensitivity of Project Cash Flow & EBITDA

		Avg. Operating Cash Flow per Annum			EBITDA Ratio (EBITDA : NSR)		
Cash Flow/EBITDA	Units	-10%	0%	10%	-10%	0%	10%
Ni Price	US$ M	191	224	254	51.4	55.9	59.1
Recovery	US$ M	192	224	256	51.5	55.9	59.5
Payability	US$ M	221	224	227	55.5	55.9	56.3
C$ F/X	US$ M	233	224	214	59.6	55.9	52.2
Initial Capital	US$ M	222	224	226	55.9	55.9	55.9
Total Investment	US$ M	220	224	227	55.9	55.9	55.9
Site Operating Costs	US$ M	237	224	211	59.9	55.9	51.9
Power	US$ M	227	224	221	56.7	55.9	55.1
Oil	US$ M	225	224	223	56.1	55.9	55.7
Labour	US$ M	226	224	222	56.5	55.9	55.4

Table 22-7: Sensitivity of Project Cash Costs

		C1 Cash Costs			AISC		
Costs	Units	-10%	0%	10%	-10%	0%	10%
Ni Price	US$/lb Ni	3.22	3.22	3.22	3.77	3.80	3.88
Recovery	US$/lb Ni	3.57	3.22	2.95	4.17	3.80	3.50
Payability	US$/lb Ni	3.25	3.22	3.19	3.83	3.80	3.77
C$ F/X	US$/lb Ni	2.93	3.22	3.51	3.51	3.80	4.10
Initial Capital	US$/lb Ni	3.22	3.22	3.22	3.81	3.80	3.80
Total Investment	US$/lb Ni	3.22	3.22	3.22	3.78	3.80	3.83
Site Operating Costs	US$/lb Ni	2.92	3.22	3.53	3.50	3.80	4.10
Power	US$/lb Ni	3.16	3.22	3.29	3.74	3.80	3.87
Oil	US$/lb Ni	3.21	3.22	3.24	3.79	3.80	3.82
Labour	US$/lb Ni	3.18	3.22	3.27	3.76	3.80	3.85

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 Project Implementation

24.1.1 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. The distribution will be tailored 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;
- flotation;
- 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 2021 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 aim of optimizing and de-risking the project's development and construction.

24.1.2 Project Schedule

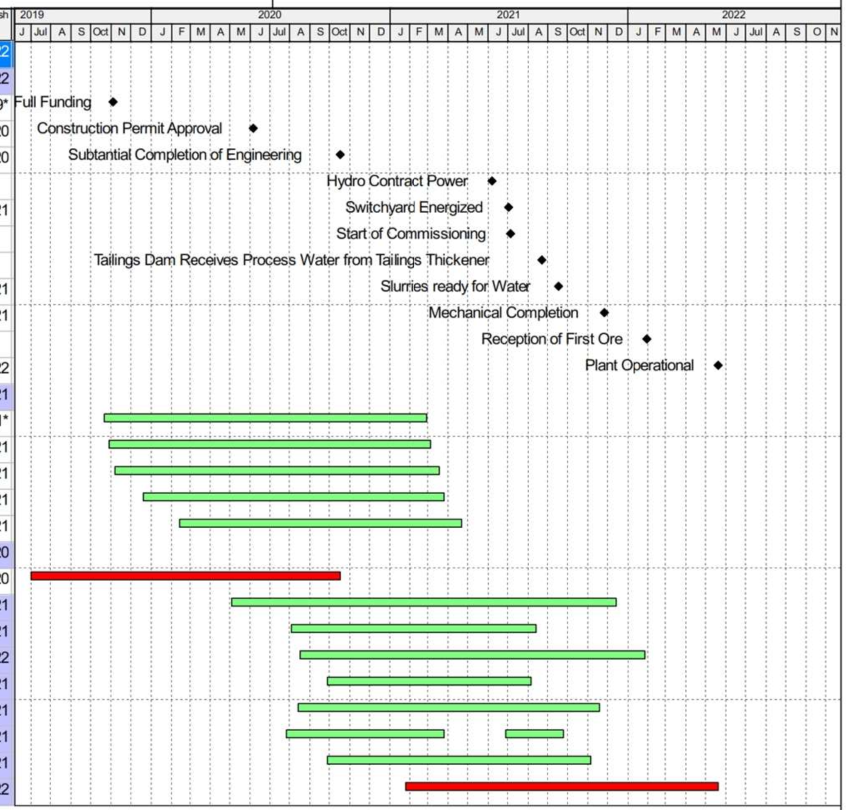
The summarized project schedule is shown in Figure 24-1. The current schedule shows:

- The overall schedule duration from the start of basic engineering in order to procure long-lead equipment to the end of ore commissioning is 33 months. Key milestone dates are described in Table 24-1.
- 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.
- Approval of a Site Construction Permit is scheduled for Q2 2020.
- Geotechnical drilling for detailed engineering will commence in Q3 2019 and be completed by Q4 2019.
- Basic engineering will commence in Q3 2019, with a commitment to purchase major mechanical capital items like the mills, mill motors, primary crusher, and flotation cells in Q4 2019.
- Award of the EPCM contract will be in Q4 2019, with full engineering effort commencing in Q1 2020.

Figure 24-1: Summarized Project Schedule

Royal Nickel - Dumont | Standard Activity Status | 10-05-2019 16:30

Activity ID	Activity Name	Start	Finish
Royal Nickel - Dumont		01-07-2019	20-05-2022
PROJECT MILESTONES		04-11-2019	20-05-2022
A1000	Full Funding		04-11-2019*
A1010	Construction Permit Approval		05-06-2020
A1020	Subtantial Completion of Engineering		16-10-2020
A1030	Hydro Contract Power	07-06-2021	
A1040	Switchyard Energized		02-07-2021
A1050	Start of Commissioning	05-07-2021	
A1060	Tailings Dam Receives Process Water from Tailings Thickener	23-08-2021	
A1070	Slurries ready for Water		17-09-2021
A1080	Mechanical Completion		26-11-2021
A1090	Reception of First Ore	31-01-2022	
A1100	Plant Operational		20-05-2022
PROCUREMENT		21-10-2019	22-04-2021
A1110	SAG Mill	21-10-2019	26-02-2021*
A1120	Ball Mill #1	28-10-2019	05-03-2021
A1130	Ball Mill #2	07-11-2019	18-03-2021
A1140	Regrind Mill	19-12-2019	25-03-2021
A1150	Gyratory Crusher (Primary)	13-02-2020	22-04-2021
ENGINEERING		01-07-2019	16-10-2020
A1160	Engineering - All	01-07-2019*	16-10-2020
100 - MINING		04-05-2020	14-12-2021
200 - CRUSHING		03-08-2020	13-08-2021
300 - PROCESS		17-08-2020	27-01-2022
400 - CONCENTRATE LOADOUT		28-09-2020	06-08-2021
500 - TAILINGS		13-08-2020	18-11-2021
600 - UTILITIES		27-07-2020	24-09-2021
700 - ONSITE INFRASTRUCTURE		28-09-2020	05-11-2021
900 - INDIRECTS		25-01-2021	20-05-2022



Remaining Level of Effort | Actual Work | Critical Remaining Work
Actual Level of Effort | Remaining Work | ◆ Milestone

Page 1 of 1 | TASK filter: All Activities | © Oracle Corporation

Table 24-1: Dumont Nickel Project Schedule – Key Milestone Dates

Criteria	Date*
Commence Detailed Engineering for Long Lead Equipment	-11Q
Commence Full EPCM	-10Q
Order Long Lead Equipment	-10Q
Individual construction permit approval	-8Q
Substantial Completion of Engineering	-7Q
Hydro Contract Power	-4Q
Start of Commissioning	-3Q
Mechanical Completion	-2Q
Reception of First Ore	-1Q
Plant Operational	0

*Q = Quarter of a year (3 months)

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 2019 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.

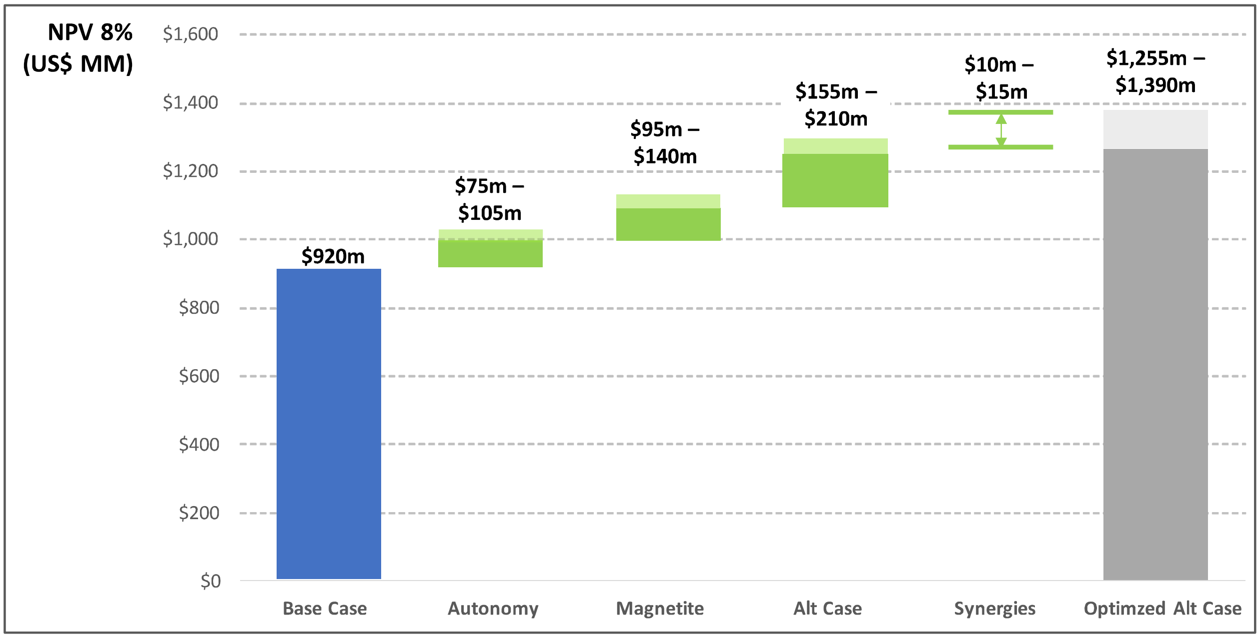
Ideally, the number of site contractors should be minimized, although this may be dictated by market and commercial considerations at the time.

24.2 Opportunities Summary

There are a number of opportunities to improve economic returns for Dumont beyond the Base Case results presented in Section 22. Those opportunities which have been investigated to a PFS or scoping study level are summarized below, with their potential economic impact illustrated in Figure 24-2. The increase in NPV for each opportunity has been presented as a range, to reflect the lower confidence of estimate compared to the Base Case.

- **Autonomous Equipment:** As autonomous equipment have been employed in open pits for over a decade and the global fleet currently approximates 400 units, automation is rapidly becoming proven technology. Dumont is considered an ideal candidate for use of autonomous equipment for factors that include the bulk nature of mining, planned use of large equipment and proximity to skilled labour. Accordingly, an industry expert Peck Tech Consulting Ltd. (Peck Tech) were engaged to assess the suitability of Dumont for automation. Based on Peck Tech's pre-feasibility level assessment, the implementation of an Autonomous Haulage System (AHS) could reduce the peak truck fleet by 20% and reduce site-wide AISC by over 3%. Further potential could be achieved with an Autonomous Drilling System (ADS).
- **Magnetite:** Dumont ore contains an average of 4.37% Fe in magnetite and is classified as Indicated Resources. Test work completed for the 2013 Study indicated that recovery of 46% to a concentrate grading 63% Fe could be achieved. Life of Project production could total 33 Mt, or approximately 1.1 Mt annually. The sale of magnetite concentrate would have the added benefit of reducing the tonnage impounded in the TSF by in excess of 19 Mt.
- **Alternate Case:** In 2017, a trade-off study identified the potential benefit of expanding the scope of operation at start-up. The concept has now been advanced to PFS level, with a modified grinding circuit allowing for initial production of 75 ktpd followed by an expansion in Year 6 to 100 ktpd. While the initial capital required for the 75 ktpd Alternative is approximately 20% higher than that of the Base Case, the modified circuit leads to greater capital efficiency over the life of project, reducing total capital by approximately 5%.
- **Synergies:** The application of automation and/or the magnetite circuit to the Alternate Case would yield incremental benefits to those achieved with the Base Case.

Figure 24-2: Potential Impact of Opportunities



Opportunities at an earlier stage of investigation, for which the potential benefit has yet to be quantified, include:

- **Staged Flotation Reactors:** The Staged Flotation Reactor is a relatively new development that offers potential savings in both capital costs (through reduced footprint) and operating costs (primarily through lower power and maintenance costs). Testing of ore properties and validation of unit capability is required prior to completing any further engineering on the concept.
- **Reblocking Measured Resources:** It would be possible to reduce the Smallest Mining Unit (SMU) for Measured Resources planned to be mining by hydraulic excavators. This could reduce dilution in the initial years of production, leading to higher grades and recovery - which will ultimately improve cash generation and reduce payback.

24.3 Autonomous Mining Equipment

Autonomous mining equipment use a combination of sensors and computers to replace the actions of a human operator. To date, OEMs have focused on units with the greatest amount of routine operation, being drills (ADS) and haulage trucks (AHS). The first commercial autonomous units began operating in 2008. At present, both Epiroc and Caperpillar offer ADS as a factory installed option on various drills. Similarly, Caterpillar and Komatsu both offer AHS on several different trucks within their fleets while Hitachi and Liebherr have successfully prototyped units and have plans for commercialization in the near future. Currently, the global fleet of ADS number 75 while there are 320 AHS units. Automation is rapidly becoming considered proven technology.

Dumont is considered to be a suitable candidate for the implementation of automation for reasons that include:

- The bulk nature of mining. As the ore zone at Dumont is massive (several hundred metres thick), continuous (no interstitial zones of waste) and homogenous, mining will be bulk in nature. Delays for shovel moves and blasting will be minimized, and the primary focus of the mining operation will be on efficiency.
- The size of mining equipment planned for use, which is aligned with the focus of automation by OEMs to date.

- Dumont's proximity to skilled labour, for supervising and maintaining autonomous equipment

Accordingly, the industry experts Peck Tech were engaged to provide inputs that would allow the potential impact of autonomous equipment to be estimated. Based on Peck Tech's recommendations, autonomy has been considered for the following units:

- Rotary blast hole drills (automation reduces the peak fleet from 5 units to 4)
- 90 t haul trucks (automation reduces peak fleet from 20 units to 16)
- 290 t trolley assist haul trucks (automation reduces peak fleet from 46 units to 37)

Areas where operating cost savings are to be expected can be summarized as follows:

- Operations Labour; with the individual units being unmanned. Replacing the operators are the supervisory staff of a 'run team', who are responsible for continuously monitoring the equipment in the field and intervening as necessary. Typically, each individual on the run team is responsible for 3 - 4 units and is at a higher pay grade than operators, resulting in a net savings for operating labour of 60 – 70%
- Haul truck tires; as operator error reduces the life of tires. The experience at operations currently using AHS is that tire life can improve 10 – 25%. The Dumont evaluation conservatively assumed a 10% improvement.
- Equipment Maintenance; The experience at operations currently using AHS and ADS is that more consistent operation of machines leads to an improvement in mechanical availability of 1 – 2% and reduction in maintenance costs of up to 10%.
- Fuel consumption; Studies have shown fuel consumption of equipment is highly dependent upon operator behaviour, with consumption by the same truck on the same profile varying by up to 40% for different operators (counter-intuitively, one study showed the driver using less fuel also achieved a faster over-all cycle time). Autonomous equipment can be programmed to operate at the set point that minimizes overall costs given local inputs, including fuel prices and labour rates.

These savings will be partially offset by the assumed increase in maintenance labour that will be required to maintain the more complex autonomous vehicles.

The capital costs associated with autonomy reflect the increased cost of individual units and a reduction in the number of units purchased over the life of mine (given the increased efficiency of individual units). For the drill fleet, where autonomy reduces the drill fleet by a single unit over the life of mine, there will be a net increase in capital costs equal to approximately 6% of the operating cost savings. For the 90 t haul trucks, there will be a greater reduction in total units purchased but the cost of automating each truck will be a significant percentage of the base machine cost and the net increase in capital expenditure will equate to 11% of the operating cost savings. For the more expensive 290 t haul trucks, the cost of automating each truck is a much lower percentage of the base cost and overall capital costs would be lower.

Table 24-2: Estimated Savings Achieved with Autonomous Equipment (C$ millions)

Operating Costs	ADS	90t Truck	290t Truck
labour [1]	($18)	($43)	($129)
consumables	$0	($3)	($26)
maintenance	($2)	($6)	($104)
diesel	($1)	($10)	($81)
power	$0	$0	$0
Capital Costs			
Fleet Capex	$1	$6	($25)
TOTAL	**($20)**	**($55)**	**($365)**

Notes

1. Labour costs include operating and maintenance personnel

Additional benefits with automation for which an economic impact has not been quantified include:

- Reduced maintenance costs for the trolley assist system due to more consistent operation and elimination of operator error (maintenance costs for the system have been based on actual costs for operations using manually operated trucks)
- The improvement in utilization and associated reduction in number of fleet units will translate to less 'bunching' of trucks on the line, for example at shift change (autonomous trucks would continue to operate through the change in crews). This will lower peak demand, resulting in a higher utilization of the trolley system and a lower cost of electricity.

24.4 Magnetite

The concept for producing a magnetite concentrate at Dumont has not changed since the 2013 Feasibility Study and is summarized below.

Dumont ore grades 4.37% Fe, resulting in a total of 44.9 Mt contained iron or 62.0 Mt magnetite. For the Base Case design, the majority of magnetite in feed reports to the magnetic concentrate and is then rejected to tailings after sulphide and awaruite recovery.

Test work performed for the 2013 Feasibility Study determined that it would be possible to recover approximately 46% of magnetite to a concentrate achieving a saleable Fe grade of 63% (see Table 24-3):

Table 24-3: 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**

A magnetite circuit at Dumont would include a four-stage cleaner separation; with the 1st stage non-magnetics report to the tailings thickener and the non-magnetics of the following stages recirculated to the regrind mill. Additionally, the awaruite 1st cleaner scavenger concentrate would report to the regrind mill.

The estimated capital cost for a magnetite circuit, escalated to current terms, is $49m per each 52.5 ktpd line of the mill.

The site costs for operating the circuit has been estimated at $2.1m pa at 52.5 ktpd, increasing to $4.0m at 105 ktpd. These costs equate to a LOM average of $3.27/t magnetite produced. The largest single element of operating costs would be plant maintenance at approximately 48% of the total. Labour costs would represent a further 19% of the total, power 12%, various other consumables 11%. Miscellaneous items add the remaining 10%.

The cost of product logistics would be significantly higher, at an estimated C$49/t for shipping overseas to Europe or Asia. In the event that product were sold to one of the domestic iron ore operations with upgrading facilities located in Quebec, logistics costs could be reduced significantly.

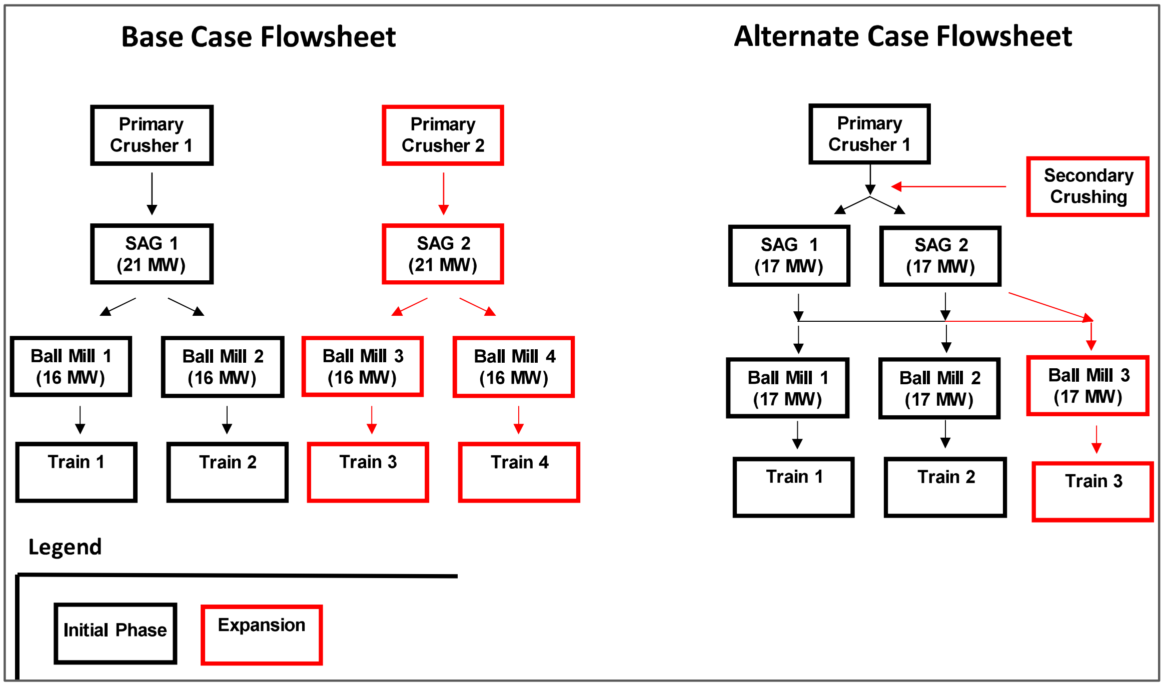
The summary value for magnetite presented in Figure 24-2 assumes production of a 62% Fe magnetite concentrate that would sell for US$ 60/t, which is near the trough in prices over the past 10 years. A $10 increase in the price received would increase the net present value attributable to the magnetite circuit by over US $50m, or 48%.

24.5 Alternate Case Production Schedule

A key consideration in selecting mill throughput of 52.5 ktpd as the Base Case is the associated capital cost. Work performed subsequent to the 2013 FS demonstrated that greater overall capital efficiency could be achieved by modifying the grinding circuit. A trade-off study completed in 2017 identified the optimal circuit, from the perspective of capital efficiency, would achieve initial throughput of 75 ktpd. This would be achieved by a single 60” x 109” gyratory crusher feeding two 36 ft SAG mills (compared to the single 38 ft SAG selected for the 52.5 ktpd Base Case) feeding twin 26.5 ft Ball Mills (compared to the two 36 ft units planned for the Base Case).

For the expansion to 100 ktpd, as the primary crusher has been increased from the 60” x 89” selected for the Base Case, no additional primary crushing capacity would be required. With the addition of a secondary crushing stage, the F_{80} to the SAG mills would be reduced from 90 mm to 50 mm and no further SAG capacity would be required. A third ball mill, identical to the two selected for the first Phase would complete the circuit expansion (See Figure 24-3).

Figure 24-3: Comparison of Base Case and Alternate Case Process Flowsheets



The PFS study performed focused on the modified comminution circuit. The rest of the flowsheet will be similar to the flowsheet considered for the Base Case. To accommodate the initial higher throughput (75 ktpd vs 100 ktpd), the flotation lines will be lengthened, or, in some cases, larger cells will be installed. The magnetic separation circuits will also be lengthened. For the expansion to 100 ktpd, instead of installing a parallel line identical to the first phase plant as was considered in the Base Case, the different flotation and magnetic separation circuits will be lengthened, and in some cases, additional lines will be added to the circuit.

In the event the decision was made to proceed with the Alternate Case, the flotation and magnetic separation circuits equipment selection and layout will require further engineering to bring the Alternate Case plant estimate and design to the same level as the Base Case.

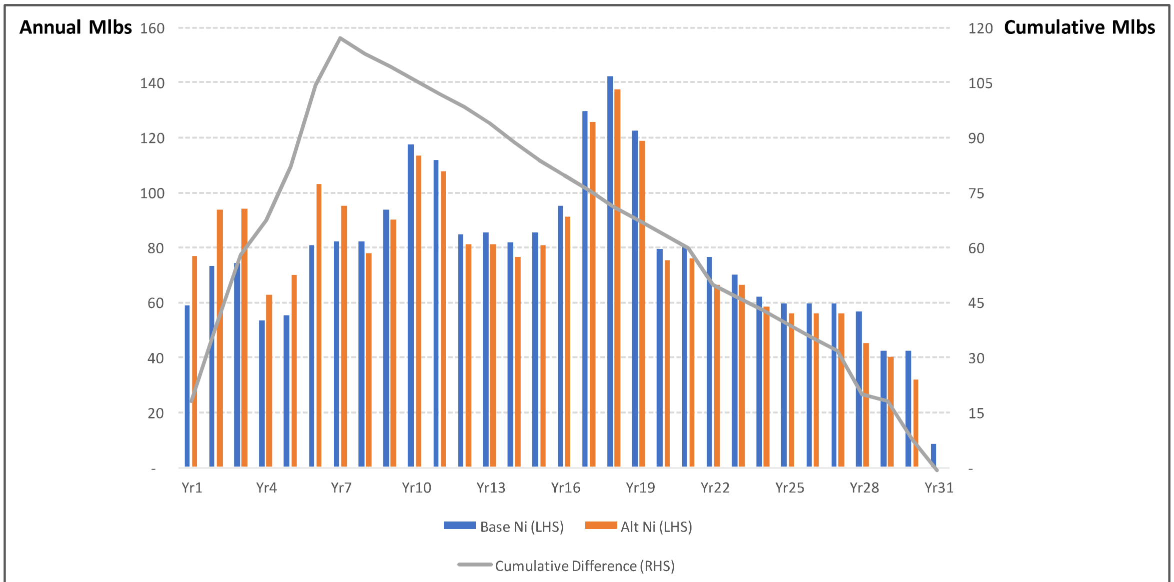
Table 24-4 compares the capital estimate for the Alternate Case to that for the Base Case. The 19% increase in initial capital costs is more than offset by the 54% decrease in expansion expenditures and total capital costs are 6% lower. Note that sustaining capital expenditures for the Alternate Case are marginally higher, due to a 6% increase in the tonnage of tails impounded within the TSF. This increase reflects the use of the same mine production schedule for both the Base and Alternate Cases. This plan results in average production of 211 ktpd over the 8 years prior to the expansion (this duration includes 2 years of pre-stripping), compared to steady-state production rates of 300 – 350 ktpd post expansion. In the event the decision was made to proceed with the Alternate Case, a revised mine plan that produced higher tonnages during the pre-expansion period would be expected to result in further economic benefit.

Table 24-4: Comparison of Base Case and Alternate Case Capital Estimates

	Base Case				Alternate Case			
WBS Area	**Ph I**	**Ph II**	**Sustain**	**Total**	**Ph I**	**Ph II**	**Sustain**	**Total**
Area 1000 - Mining	$298	$0	$600	$898	$298	$0	$599	$896
Area 2000 - Crushing	$64	$62	$0	$126	$66	$30	$0	$96
Area 3000 - Process	$397	$385	$64	$846	$517	$128	$64	$708
Area 4000 - Concentrate Load-Out	$0	$0	$0	$0	$0	$0	$0	$0
Area 5000 - Tailings	$48	$31	$167	$247	$59	$21	$174	$254
Area 6000 - Utilities	$180	$133	$0	$312	$234	$78	$0	$312
Area 7000 - On-Site Infrastructure	$79	$24	$0	$103	$94	$31	$0	$125
Area 8000 - Off-Site Infrastructure	$16	$1	$0	$17	$12	$4	$0	$17
Area 9000 - Indirects	$124	$87	$0	$212	$155	$40	$0	$196
Area 10000 - Owner's Costs	$40	$7	$0	$47	$43	$6	$0	$49
Area 11000 - Contingency	$111	$71	$0	$182	$130	$31	$0	$162
Total	**$1,357**	**$801**	**$831**	**$2,990**	**$1,609**	**$370**	**$836**	**$2,815**
Variance					**19%**	**(54%)**	**1%**	**(6%)**

The difference in operating costs for the two cases is marginal, with LOM costs for the Alternate Case being US$0.05/t (0.7%) less than those of the Base Case. A much more significant impact is the timing of Ni output. Figure 24-4 illustrates that by Year 7, the Alternate Case has produced 117 million lbs more Ni than the Base Case. Post expansion, the Base Case recovers this difference through the 5 ktpd difference in Phase II milling rates (105 ktpd for Base vs 100 ktpd for Alternate). As a result of this accelerated profile, the $NPV_{8\%}$ for the Alternate Case NSR is 7% higher than that for the Base Case.

Figure 24-4: Comparison of Base Case and Alternate Case Payable Ni Production



24.6 Other Opportunities

24.6.1 Staged Flotation Reactor

As part of the 2019 feasibility study update, a conceptual study was completed to evaluate the use of Woodgrove Staged Flotation Reactor (SFR) in lieu of conventional tank flotation cells. The “first generation” SFR have now been superseded by the next generation Direct Flotation Reactor (DFR).

The key benefits of the SFR/DFR arrangement, when compared to conventional cells (as promoted by Woodgrove), are improved collection efficiency, better sulfide and non-sulfide gangue rejection (from dramatically lower air consumption and froth washing), and higher froth and stage recoveries. In practical terms, this equates to the potential for fewer stages, reduced overall cell volume and significantly smaller footprint and installed capital and operating cost..

24.6.1.1 Capital Costs Estimate

The capital costs were prepared to a ±30% level of accuracy with a base date of first quarter 2019 (‘Q1CY2019’) and in Canadian dollars (‘C$’).

Two options have been estimated as follows:

Option 1 – Installation of conventional flotation tank cells (the “Base Case”)

Option 2 – Installation of direct flotation reactor cells (the “DFR case”).

Table 24-5 shows the direct capital costs for each option considered. The direct capital cost estimate covers the design and construction of the process plant and utilities for the flotation circuits for the 52.5 kt/d Dumont Nickel Project (the first phase of the project that will be duplicated for the second phase).

Table 24-5: Direct capital cost breakdown of both options (±30%, Q1CY2019 C$ millions)

Item	Total Cost Estimate (C$ million)		
	Option 1 - Base Case	Option 2 - DFR Case	Difference
03 PROCESS			
03-100 Process General	**88.8**	**62.2**	**(26.6)**
03-200 Grinding Circuit	**157.5**	**157.5**	**---**
03-300 Slimes Flotation	**41.4**	**35.3**	**(6.1)**
03-400 Nickel Flotation	**49.2**	**40.5**	**(8.7)**
03-500 Magnetic Separation	**27.0**	**27.8**	**0.8**
06 UTILITIES			
06-100 Air Systems	**8.4**	**6.7**	**(1.7)**
Total Direct Costs, C$ million	**372.3**	**330.3**	**(42.3)**
Reduction in Capital Cost			**11.4%**

24.6.1.2 Operating Cost Estimate

The operating cost estimate is presented in Canadian dollars (C$) with a base date of first quarter of 2019 (‘Q1CY2019’). The estimate is considered to have an accuracy of ±30%. No allowance has been included in the estimate for escalation from this date.

The estimate incorporates common cost areas (i.e. utilities and maintenance spares) for both the conventional flotation tank cell and DFR circuits. The estimate of total costs is summarized by area in Table 24-6.

Table 24-6 – Operating cost breakdown of both options (±30%, Q1CY2019 C$ millions)

Cost Area	Option 1 - Base Case		Option 2 - DFR	
	Annual Cost (C$,000)	Unit Cost (C$/ton)	Annual Cost (C$,000)	Unit Cost (C$/ton)
Utilities	10,750	0.20	7,340	0.14
Materials and Supplies	3,154	0.06	2,111	0.04
Total	**13,904**	**0.26**	**9,451**	**0.20**

24.6.1.3 Recommendations for Future Work

There are some areas which require further works that may reduce the risk profile of the DFR option. It is recommended that the following additional works be undertaken, including:

- Further piloting test work is required to generate, investigate or confirm parameters and design criteria developed and assumptions made for the inclusion of the Woodgrove DFR cells in the Dumont flow sheet. Test work should be conducted on ore types with variable serpentine and brucite content.
- A review of the data from previous Woodgrove pilot tests and plant trials and operating data from the various plants with SFRs/DFRs installed would also be very beneficial (depending on any confidentiality considerations).
- Continue to develop the DFR circuit to provide additional confidence in relation to the layout and equipment costs. Footprint estimates are considered preliminary and equipment costs are budget prices only. A formal enquiry with duty specifications should be issued to Woodgrove to provide greater certainty around these items.
- Investigate alternative layout options to reduce circuit complexity and costs. Layouts options include: relocating the regrind ball mill from the grinding building to within the DFR flotation building; and assuming the current project flotation buildings sizes which can install the entire DFR flotation circuit for the expanded throughput of 105 kt/d.

The DFR piloting test work can be completed within the schedule timeline within delays to the overall project schedule. An improved project schedule is achievable due to the shortened equipment leads times, less bulk materials, and resulting reduction in the construction and installation of the DFR cells.

24.6.2 Reblocking Measured Resource

The resource block model uses a Smallest Mining Unit (SMU) of 20m x 20m x 15m in X-Y-Z, which is appropriate for both the rope shovels that will be used for the bulk of mining and the density of drilling for Indicated Resources, which comprise approximately 67% of the total ore. An SMU of this size results in some unavoidable smoothing of resource grades as discrete zones of lower and higher grades are combined in a single block. The 33% of total ore that is classified as Measured Resources may support a smaller block size of 10m x 10m x 7.5m, and this SMU would be appropriate for the smaller hydraulic excavators that will be responsible for the bulk of loading in the early years of operation (to the end of Yr5 of mill operations, 61% of ore is planned to be loaded with excavators). Reblocking the Measured Resources that will be loaded by excavators could result in higher grades and recovery for the initial years of operation, which will ultimately improve cash generation and reduce payback.

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 $1,357 M, an expansion capital investment of $801 M and sustaining capital of $815 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 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 wall 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 test work 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 kWh/t, RWi 15 kWh/t, CWi 14 kWh/t, and Ai 0.009g.
- 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 test work indicates that nickel recovery is relatively insensitive to grind sizes (P_{80}) up to about 150 mm. Further test work and flowsheet development has lead to the selection of a grind size of 150 mm (P_{80}) for the plant design.
- 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% Ni.
- Cobalt recovery to concentrate was estimated at 33%.
- 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 that the rail option was the most economical and practical alternative to transport the nickel concentrate production to markets.
- To effectively settle the slimes portion of the tailings, a small portion of coarse material must be added
- 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, noise, wetlands, fish habitat, water resources, and the social environment. No specific inordinate environmental risk to project development was identified. Although, they are some sensitive element in the footprint surrounding, the work of optimization made in 2018-2019 on the mining plan and design eliminate or reduce the effect of the project on these components.
- 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. The negative impacts previously identified in the preliminary ESIA remain the same, after the optimization of the mine design in 2018-2019, but impacts on air quality and noise will be reduced. However, the negative impact reduction is not significant to result in a change in the impact importance evaluation when the impact evaluation methodology is applied.
- 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.

- Continue environmental baseline studies as required;
- Complete detailed design that considers the following points:
    - Using a smaller SMU size to reblock Measured Resources planned to be mined with smaller excavators. This could result in delivery of higher grade and/or recovery material delivered to the plan in initial years of operation.
    - Begin detailed engineering upon additional financing and procure long lead equipment in order to maintain the target 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;
    - 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, 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 test work program that will include:
    - Slimes cyclone pilot scale testing for detailed engineering design
    - Awaruite recovery circuit optimization
    - Recovery opportunities from scavenger non-magnetic stream
    - Complete test work to quantify grindability characteristics of regrind mill feed
    - Additional thickening test work on the slimes and coarse portion of the tailings by various ore type domains
- Specific high voltage power studies as recommended for confirmation of high voltage supply by Hydro Quebec.
- Continue mining lease process.
- Continue surface lease process.
- 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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