GEOMETALLURGICAL MODELING OF THE DUMONT DEPOSIT

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ABSTRACT

The Dumont deposit is a large (1,179 Mt proven and probable reserve) low grade (0.27% nickel), nickel sulphide project located in the Abitibi region 25km west of Amos, Quebec. Royal Nickel Corporation acquired the project in 2007 and has since taken the project from a scoping study in 2010 through to completing the feasibility study in July 2013.

The Dumont deposit in a nickel deposit with the recoverable nickel contained in three minerals; pentlandite, haezlewoodite and awaruite. The minerals are recovered by a combination of flotation and magnetic separation. Over the course of the studies many flowsheet design decisions and changes were made. This paper outlines an overview of the project, flowsheet design including the comminution circuit, desliming and awaruite recovery. The overall resulting flowsheet that was used for the feasibility study is then presented along with the confirmatory locked cycle testing that showed the feasibility design basis was able to produce the predicted concentrate grade and recovery.

KEYWORDS

Flowsheet design, nickel, ultramafic, desliming

INTRODUCTION

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

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.

The Feasibility Study for the Dumont project was completed in July 2013. Since the feasibility study was completed the focus of the work on Dumont was to support the environment approval process and financing. The Certificate of Authorization from the Quebec Ministry of Sustainable Development, Environment and the Fight Against Climate Change was received in June 2015. This authorization is the most significant permit for mining projects in Quebec and positions Dumont to proceed to construction upon completion of financing. In July 2015, RNC received a positive Environment Assessment Decision for Dumont from the Federal Minister of the Environment.

MINEROLOGY OVERVIEW

The Dumont sill lies within the Abitibi sub-province 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.

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.

PROJECT OVERVIEW

Dumont Project

The Dumont Nickel Project has a very large, low grade proven an probably reserve of 1,179 M tonnes at 0.27% Ni. The open pit mine has been designed to provide ore to the plant in a manner that optimises net present value. The initial plant throughput is 52.5 kt/d, with expansion in Year 5 to 105 kt/d. A key component of the mine plan is the accelerated release of ore from the pit, with higher value ore being fed directly to the mill and lower value material being temporarily stockpiled. During the life of pit, a total of 606 Mt will be loaded to the low-grade stockpiles. Of this, 103 Mt of the highest value stockpile material will be reclaimed during the initial 20 years that the pit is still active. The remaining 503 Mt will be reclaimed after pit closure, extending the life of project for a total of 33 years.

Stockpiling lower-value material maximises the value of material treated during the initial years. Annual output averages 68 Mlbs payable Ni during the first 4.5 years of production, when the concentrator throughput is 52.5 kt/d. Annual output increases to 104 Mlbs for the next 15 years when the pit is active. After the pit is depleted and processing of stockpile material only commences, annual output drops to an average of 65 Mlbs. The strategy of accelerated mining has the additional advantage of creating a void, which would accommodate approximately 43% of the tailings produced, thus reducing the surface footprint of operations.

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

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

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

The 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.
- Water – Water for start-up will be provided by surface water storage at the Southeast Reservoir and, possibly, local groundwater wells. During operations, water demand will largely be met by recycling water from the tailings facility. Make-up water and freshwater requirements will be provided by the Southeast Reservoir. A water treatment plant will be constructed to treat excess water from the TSF prior to its discharge to the Villemontel River.

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

The TSF will be situated approximately 400 m west of the process plant and consists of two cells. Cell 1 will be constructed initially, followed by Cell 2 during Year 6 of operations. The TSF is designed to store approximately 680 Mt of tailings produced over a period of approximately 20 years. Once mining has ceased at the open pit, stockpiled ore will be processed for approximately 13 years and those tailings, approximately 498 Mt, will report to the open pit.

Mill Flowsheet Development Goals

The mill flowsheet was designed with several goals. The metallurgical test program was completed to achieve these goals. In many cases, the goals were conflicting (i.e. sometimes reducing operating costs means increasing capital costs), this was handled by various trade-off studies during the pre-feasibility and feasibility study.

- Ability to handle ore variability
- Equipment selection based on existing technology, currently operating at similar scale
- Simplify flowsheet to increase operability
- Minimize operating cost
- Minimize capital cost

This paper reviews the work that was carried out to achieve these goals and produce the overall feasibility study mill flowsheet.

GRINDING CIRCUIT

Overview of Variability Samples

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

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

Overall the hardness seen in the 102 samples shows a very small range of variability compared with other deposits. This is explained by the overall composition of the dunite material is >90% serpentine without the presence of large dykes or other lithologies within the pit shell. An attempt was made to domain the results but with the lack of variability from hardest to softest the regressions weren't strong with either the assays or mineralogy.

There was a slight trend seen in samples that were higher in olivine. They were slightly harder than those with higher amounts of Fe-Serpentine, which in turn were marginally harder than samples containing mostly Mg-Serpentine and very little Fe-Serpentine. This information was entered in the block model and the Bwi and Axb are estimated in mill feed in the feasibility model.

Grind Size Selection

Based on the initial minerolgy, it was thought that due to the fine dissemination of the recoverable nickel minerals and their lack of apparent liberation, there would be a need for a relatively fine grind. This is problematic with ultramafic ores because of their tendency to become highly viscous as the grind gets finer. Initially tests were targeting a 44 micron final grind, however the high viscosity slurry that resulted was unable to be floated. Subsequent testing showed that despite initial mineralogical evidence, decent liberation and recovery was seen at much coarser grinds.

The standard flotation test that was used to evaluate the variability samples, used an initial grind size of 150 microns. Test work was then completed to confirm the optimum size to maximize rougher nickel recovery. Tests were performed on two samples, the sulphide composite (pentlandite dominated) and 222BDE, a low recovery Haezlewoodite sample.

The results are shown below in Figure 1. This figure plots the flotation rougher recovery vs. the P_{80} of the sample. Both samples show an approximate peak in flotation recovery between 130-160 µm.

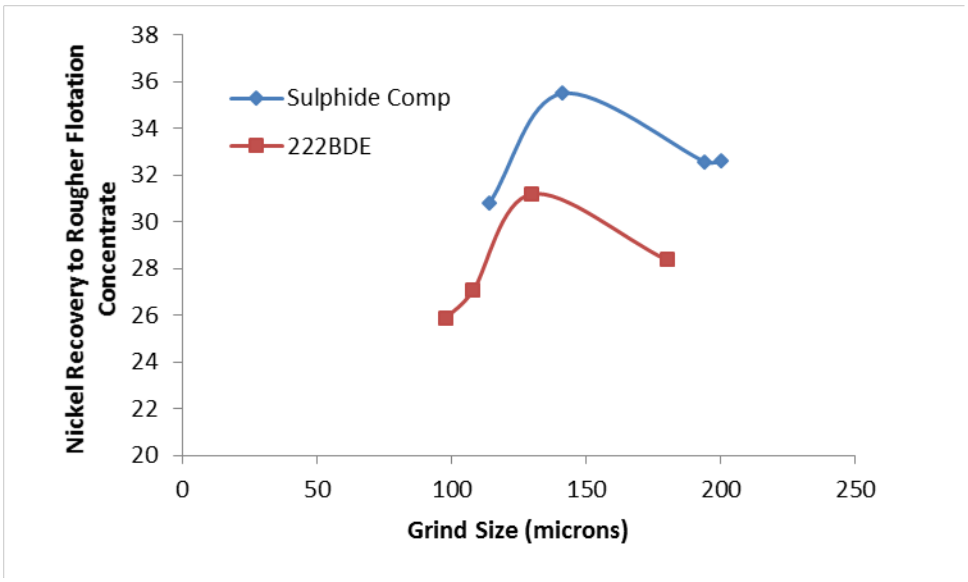

Figure 1 – Effect of Grind Size on Rougher Flotation Recovery

Total rougher recovery is comprised of both the flotation and magnetic recovery. Figure 2 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.

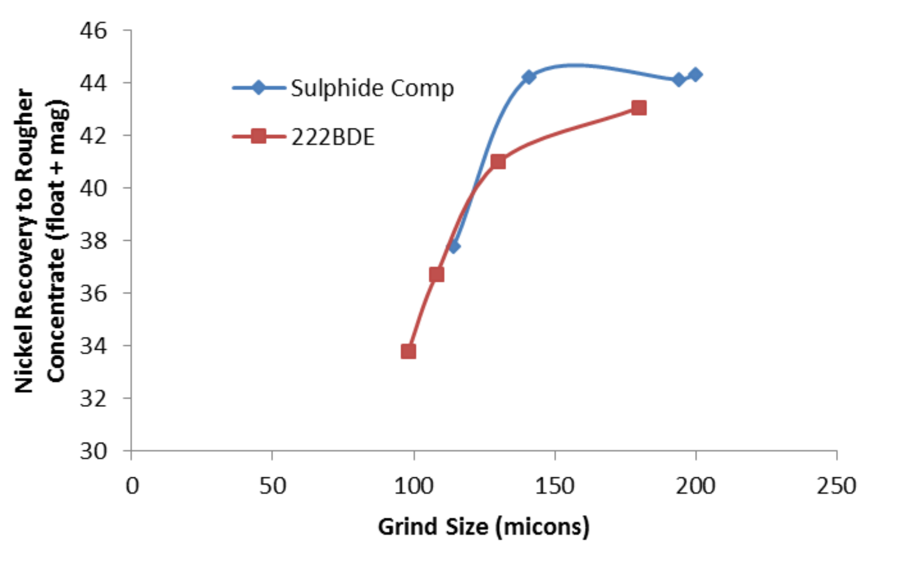

Figure 2 – Effect of Grind Size on Total Rougher Recovery (including Magnetic Concentrate)

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 3).

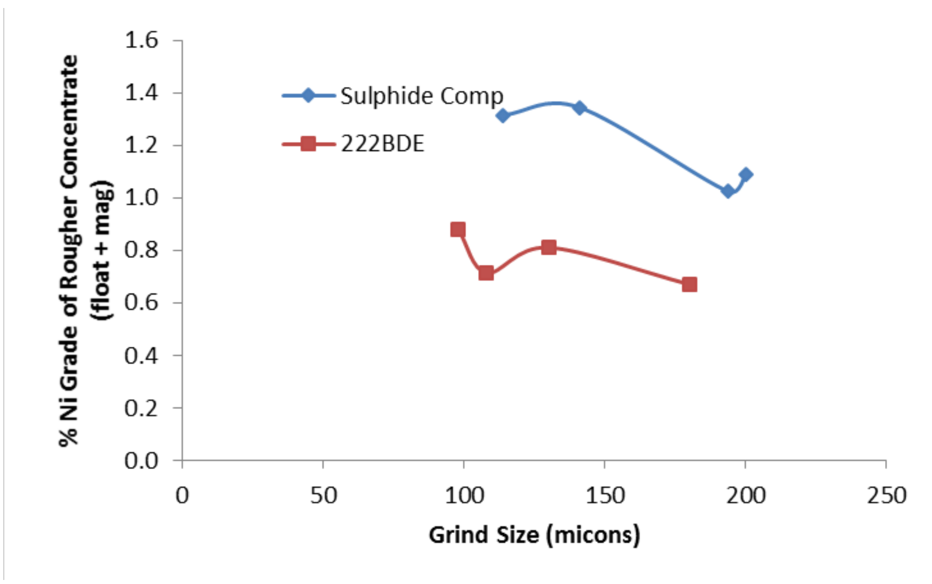

Figure 3 – Effect of Grind Size on Overall Rougher Concentrate Grade

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

ROUGHER CIRCUIT

Reagents used consisted of xanthate, methyl isobutyl carbonyl (MIBC) and a dispersant. The most crucial was the dispersant. Serpentine has an opposite surface charge from the Ni sulphide minerals and has a tendency to cover the Ni mineral surface. For the test work, the dispersant used was hexametaphosphate (Calgon) in the rougher and carboxymethyl cellulose (CMC) in the cleaners. Sulfuric acid was also successful as an alternative to Calgon in the rougher. CMC can be used in the rougher but can create complications with the froth in the cleaner circuit because of the amount required. In addition CMC is the most expensive per tonne, and therefore to minimize operating cost optimization focused on the use of Calgon and sulphuric acid with minimal CMC added during cleaning.

Ni recovery in the rougher circuit highly depended on the grind size and slimes removal. The rougher circuit design used to study this, shown in Figure 4, was selected for the standard test procedure (STP).

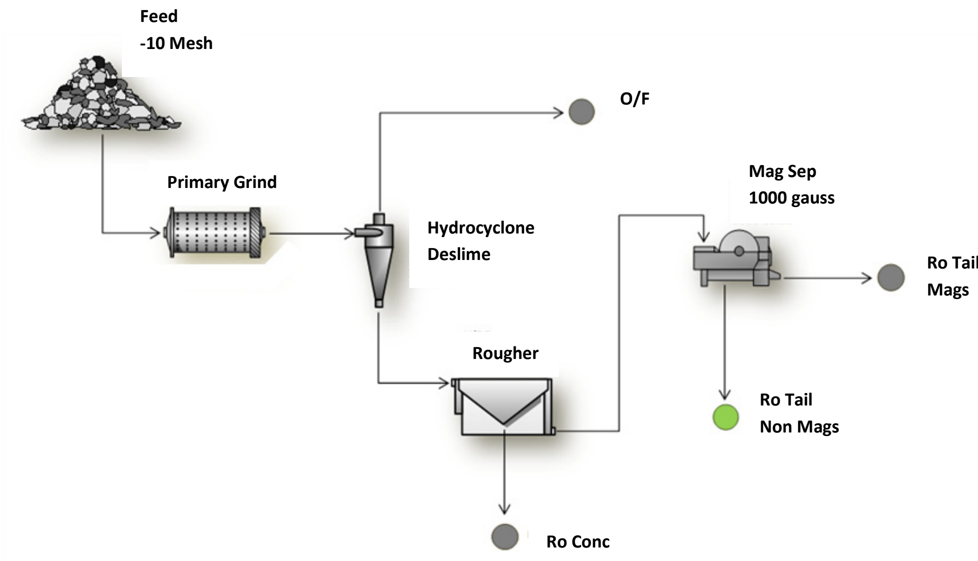

Figure 4 – Rougher Circuit

Mineralogy suggested a fine grind size ranging from 50 to 70 µm was required to liberate the Ni minerals for flotation but because the Ni minerals were associated with magnetite which is more brittle compared serpentine it was possible to liberate the Ni using a coarser grind. The non-liberated Ni sulphides still associated with magnetite and awaruite was recoverable by magnetic separation of the flotation tails.

It was also important to understand how to interpret the rougher recovery resulting from the STP circuit. Overall rougher results can be misinterpreted if the concentrate grades are not taken into consideration. This ore contains non-recoverable nickel in the serpentine gangue which can be interpreted as recoverable Ni if a higher mass pull was obtained in the rougher. Understanding the overall rougher recovery numbers can be difficult especially when the initial feed grade is very low.

To overcome this problem an equation was derived using the cleaning behavior of this ore from 36 locked cycle tests for the overall circuit.

Equation 1 calculates overall Ni sulphide and awaruite cleaner recoveries for a chosen Ni cleaner grade using rougher grades and recoveries assuming the circuit is in steady state.

$$Cl_{cr} = 0.9459\left[\frac{Cl_{cg}\,Ro_{cr}}{\left(Cl_{cg} - 2Ro_{tg}\right)\left(1+\frac{2Ro_{tg}}{Ro_{cg} - 2Ro_{tg}}\right)} + 0.7\left(\frac{Mg_{cr} - \frac{60}{100}Ro_{tg}}{Mg_{cr}}\right)\right] + 4.0073 \qquad (1)$$

Where:

- Cl_{cr} – Cleaner Recovery, % Ni
- Cl_{cg} – Targeted Cleaner Grade, % Ni
- Ro_{cg} – Ro Conc Grade, % Ni
- Ro_{cr} – Ro Conc Recovery, % Ni
- Ro_{tg} – Ro Tail Non Mag Grade, % Ni
- Mg_{cr} – Ro Tail Mag Recovery, % Ni

Verification of the equation was done by plotting a curve comparing 18 locked cycle test (LCT) cleaner recoveries to rougher flotation calculated cleaner recoveries of various composites. Figure 5 shows the correlation between the 2 cleaner recovery products.

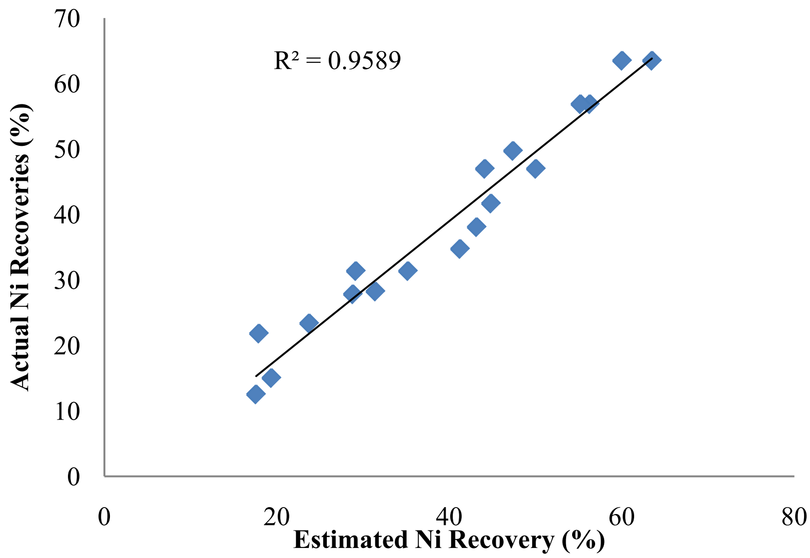

Figure 5 – LCT Measured Cleaner Recoveries vs Rougher Calculated Cleaner Recoveries

Using this method with estimate cleaner recovery was used to evaluate the rougher flotation results of various test work. However, the estimated cleaner recovery models were only used to evaluate optimization test work. Overall recoveries used for the feasibility study economics were based on actual test work.

HYDROCYCLONE DESLIME CIRCUIT

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. Desliming is performed after the primary grind using hydro cyclones. The overflow (O/F) is sent to the slimes rougher circuit and the underflow (U/F) is sent to the nickel sulphide rougher circuit.

The standard flotation test procedure had an average of 7% mass report to the slimes fraction. With 7% mass sent to the O/F the rougher of the U/F could be floated but at an effective pulp density of 16 % solids and with a long residence time. Using this method produced a rougher concentrate high in fiber content and very difficult to clean in a cleaner circuit. Test work using a higher mass pull (approximately 20% by weight) to the O/F showed that it was possible to improve the U/F rougher float kinetics while reducing the reagent consumption and increasing the pulp density but Ni loses to the O/F was a concern.

Table 1 shows comparative rougher test work conducted on various samples using a high mass pull to the O/F versus a low mass pull to the O/F. Equation 1 was used to estimate the Ni cleaner recoveries for each rougher test.

Table 1 – Actual Ni Rougher Recoveries and Estimated Cleaner Recoveries vs. P80

Sample ID		Batched Test Products, % Ni					Ni Recovery, %		O/F wt
		Ni Rocg	Ni Rocr	Ni Rotg	Mg1cg	Mg1cr	Comb. Ro	Calc. Clnr	% dist.
Comp 1	High O/F	3.68	9.73	0.23	0.70	28.5	38.2	24.9	19.7
	Low O/F	0.76	12.9	0.23	0.59	17.5	30.4	16.0	7.75
Comp 2	High O/F	9.15	28.1	0.21	0.94	10.2	38.3	35.0	17.6
	Low O/F	1.91	32.9	0.22	1.44	8.21	41.2	33.0	7.17
Comp 3	High O/F	10.8	24.2	0.19	0.36	22.0	46.1	33.2	22.6
	Low O/F	1.31	25.4	0.22	0.39	20.7	46.0	26.1	7.57
Comp 4	High O/F	9.77	36.9	0.18	0.49	17.2	54.1	45.2	20.4
	Low O/F	1.57	42.6	0.18	0.44	17.2	59.8	42.2	8.03
Comp 5	High O/F	6.61	10.4	0.23	0.32	26.6	37.0	18.3	18.6
	Low O/F	1.06	15.6	0.23	0.34	24.7	40.3	17.8	7.36
Comp 6	High O/F	7.71	28.3	0.20	0.45	21.1	49.4	37.5	17.3
	Low O/F	1.62	32.4	0.21	0.39	18.5	50.9	32.7	9.26
Comp 7	High O/F	5.08	19.6	0.19	0.36	23.2	42.8	28.6	22.4
	Low O/F	0.81	25.2	0.19	0.33	21.2	46.3	22.7	7.29

Results show that in most cases the overall rougher recoveries were higher when using the lower mass pull to the O/F but with a much lower concentrate grade. When calculating the Ni cleaner recoveries it was clear that using the higher mass pull for desliming was more effective. This led to the decision to use 20% weight split to the overflow for feasibility flowsheet design.

MAGNETIC CONCENTRATE PROCESSING CIRCUIT

After the nickel sulphide rougher flotation the rougher tails are fed to a magnetite separator. The magnetic separator is used to recover any unrecovered sulphides that are locked with magnetite and awaruite, which is highly magnetic. The rougher magnetic concentrate and first sulphide cleaner tails magnetics report to a regrind mill to liberate any locked particles associated with magnetite prior to sulphide scavenging and awaruite (Aw) flotation.

Nickel sulphide scavenger recovery required a regrind size of ~75 µm but liberation of the Aw particle required a grind size ranging from 45 to 60 µm. Locked cycle tests showed good Ni recovery when the regrind size was targeted at 50 µm.

Ni Sulphide Scavenger and Awaruite Flotation

After regrinding, the Ni sulphide can be recovered by flotation leaving the Aw and magnetite in the scavenger tails. A process needed to be developed to separate the Aw from the magnetite, both flotation and gravity separation were considered. Aw has a specific gravity of ~8, and magnetite has a specific gravity ~6. Although close, there was enough of a difference between the two minerals to try gravity separation.

Samples for this test work were generated from magnetic concentrate that had been regound to 50 microns and then floated to remove the nickel sulphide material.

Gravity separation had some success in recovering the Aw using a shaking table. Due to the very fine liberation needed to upgrade the Ni, shaking table operation was very difficult. Further testing conducted with a Kelsey Jig obtained similar results. The gravity recovery results are shown in Table 2.

Table 2 – Aw Recovery of the Ni Sulphide Float Tails Using Gravity

Comp 4 Products	Weight %	Grade % Ni	Ni Recovery %
Gravity Conc	8.48	2.54	45.7
Gravity Tail	91.5	0.28	54.3
Scav Tail (Feed)	100	0.47	100

A test program was designed to evaluate Awaruite flotation under various conditions. After several attempts using different flotation conditions, Aw recovery was achieved from the re-ground magnetic concentrate. The flotation process was demonstrated to be the most effective method in recovering and upgrading the Aw. The results from a selected sample are show in Table 3.

Table 3 – Aw Recovery of the Ni Sulphide Float Tails Using Flotation

Comp 4 Products	Weight % Dist.	Grade % Ni	Ni Recovery % Ni
Ro Conc	1.59	18.3	63.7
Ro Tail	98.4	0.17	36.3
Scav Tail (Feed)	100	0.46	100

Figure 6 shows the scavenger circuit developed for Ni sulphide and Aw recovery where the scavenger concentrate is circulated back to the 1st nickel sulphide cleaner.

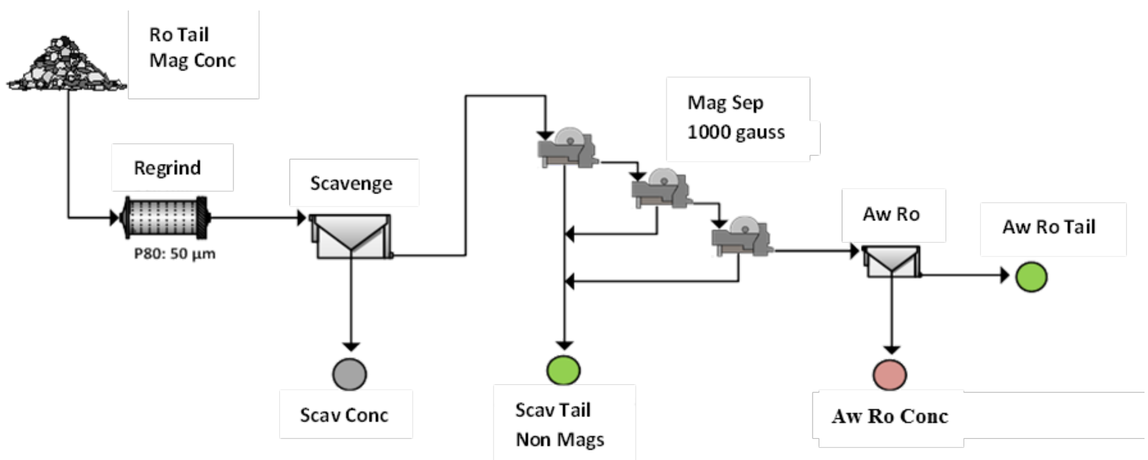

Figure 6 – Scavenger Circuit (Ni Sulphide and Aw Flotation)

Over time, additional magnetic cleaning stages on the scavenger tails were added as the test work was optimized. Three magnetic separation stages were used to produce a clean Ni iron product. This allowed for reduced reagents and a very selective Aw float leaving a high grade magnetite in the Aw rougher tails. Weight removal, after the magnetic separation, range from 50 to 70 % of the initial magnetic concentrate. Ni grade to the Aw rougher feed after regrind and the three magnetic separation stages can range from 0.05 up to 2 % Ni depending on the Aw content in the sample.

Test work on various composite samples were conducted using the rougher tail magnetic concentrates as feed for the scavenger circuit. Results are shown on table 4.

Table 4 – Ni Recovery of the Magnetic Concentrate of various composite samples

Sample ID		Weight %	Grade % Ni	Ni Recovery %
Comp 1	Scav Conc	1.01	4.58	13.1
	Aw Ro Con	0.75	16.5	35.3
Comp 3	Scav Conc	1.23	7.84	23.6
	Aw Ro Con	1.11	12.8	34.4
Comp 4	Scav Conc	1.14	8.44	20.2
	Aw Ro Con	1.57	15.2	50.1
Comp 5	Scav Conc	1.84	1.89	9.63
	Aw Ro Con	0.66	22.6	41.4
Comp 6	Scav Conc	1.18	15.9	40.5
	Aw Ro Con	0.94	9.25	18.8
Comp 7	Scav Conc	1.01	10.8	28.1
	Aw Ro Con	0.55	16.7	23.4

Ni recovery from the magnetic concentrate ranged from 48.4 to 70.3 % using the scavenger circuit with the regrind size for each composite were within the range of 40-50 µm.

After Aw roughing, the rougher concentrate was fed to a single stage Aw cleaner, which upgrades the Aw rougher concentrate to final concentrate grade, approximately 30-50% Ni depending on the sample.

FEASIBILITY FLOWSHEET

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

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

The process plant will be built in two phases. Initially, the plant will be designed to process 52.5 kt/d with allowances for a duplicate process expansion to increase plant capacity to 105 kt/d. Common facilities will include concentrate thickening and handling and sulphuric acid off-loading and containment. The mill flowsheet is shown in Figure 7.

The overall feasibility flowsheet is the culmination of 7 years of design work completed on various samples. To complete the feasibility study, 6 locked cycle tests were completed on samples from the main identified ore domains to confirm the overall flowsheet delivered nickel grade and recovery with the grind, residence times, reagent suite etc.

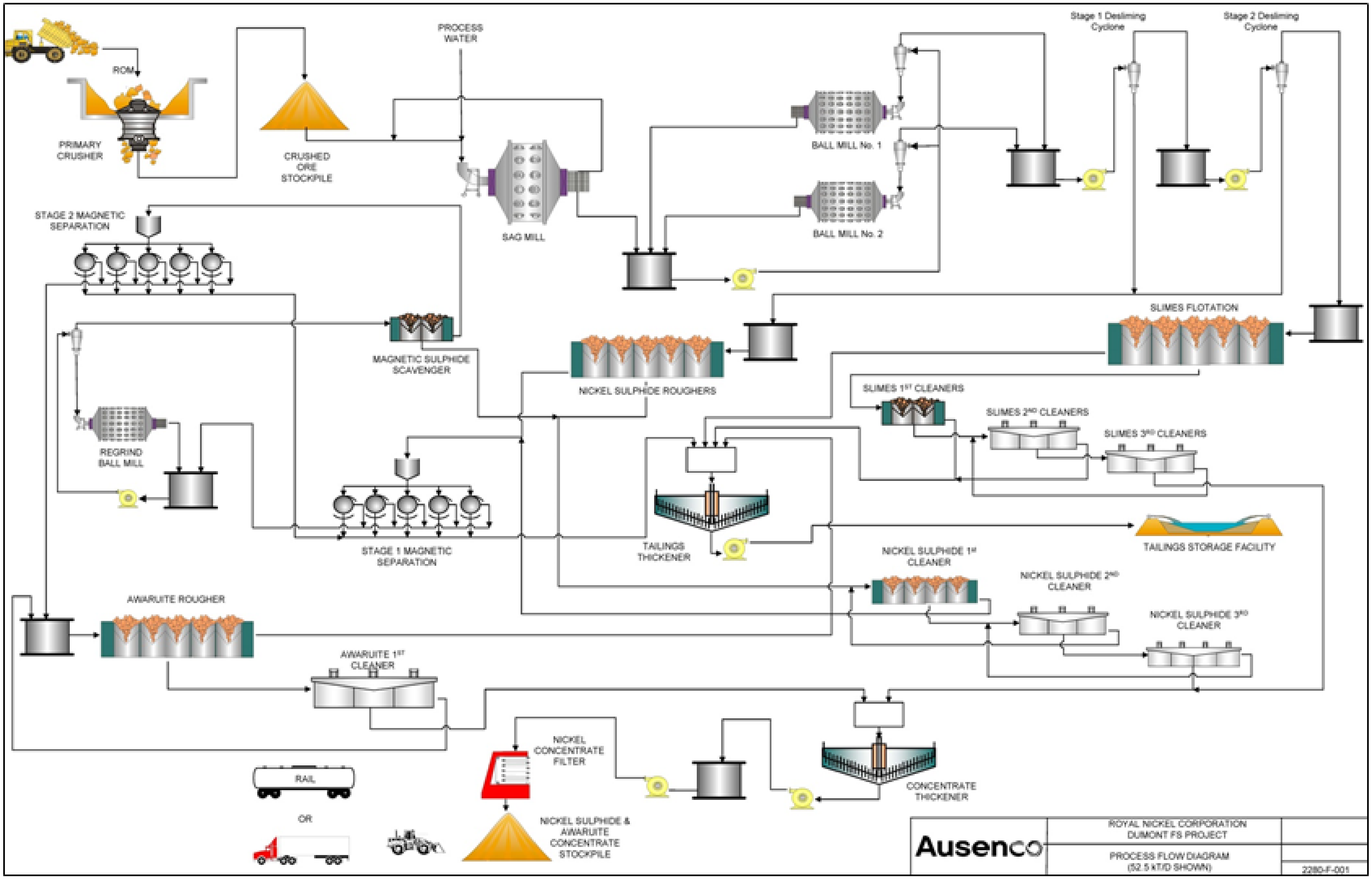

Figure 7 – Dumont Feasibility Study Flowsheet

The reagent suite used for the feasibility costing was compared with the actual average used in the 2013 LCT testing (Table 5). 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 (cost shown from 2013 feasibility study).

Table 5 - Reagent Consumption for the 2013 Locked Cycle Tests

	PAX (g/t)	MIBC (g/t)	Cytec 65 (g/t)	Calgon (g/t)	CMC (g/t)	H_2SO_4 (g/t)	Cost ($/t)*
Feasibility Opex Basis	80	89	2	254	6	3888	1.41
Average 2013 LCT usage	89	77	0	135	16	5100	1.25

The locked cycle recoveries for the 2013 samples are shown in Figure 8. Overall the results, that while there was variation around the model, the average nickel recovery in the locked cycle tests was similar to the predicted feasibility recoveries, as obtained with the feasibility design basis.

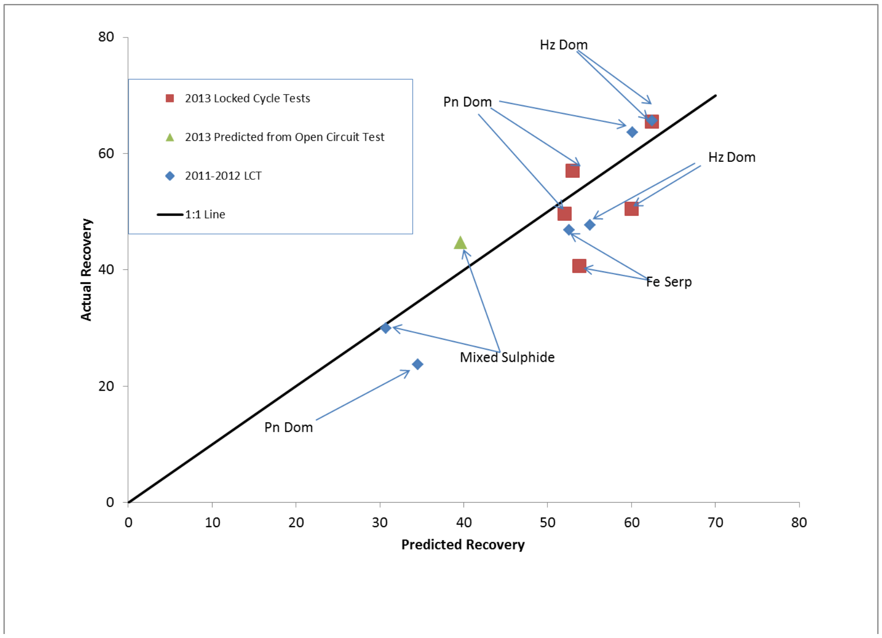

Figure 8 – Locked Cycle Test Performance vs. Feasibility Recovery Model

CONCLUSIONS

Flowsheet development for the Dumont deposit started in 2008 with initial metallurgical and mineralogical testing, culminating in 2013 with the Feasibility Study. Over the time period the flowsheet changed significantly to reduce both capital and operating cost, simplify unit operations as much as possible and produce a robust capable of handling the variability seen in the ore samples. Each circuit was optimized for grade and recovery, while minimizing operating and capital costs. The overall flowsheet, was demostrated towards the end of the feasibility study to confirm the design basis achieved the predicted concentrate grade and recovery.

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