WISCO Century Sunny Lake Iron Mines Limited

Technical Report on the Preliminary Economic Assessment for the Full Moon Project

3767 | M03483A

April 14, 2015

M03483A CIMA+

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1.1 Introduction

CIMA+ was retained by Century Iron Mines Corporation (TSX: FER) ("Century"), through WISCO Century Sunny Lake Iron Mines Limited ("WCSLIM"), a joint venture with WISCO International Resources Development & Investment Limited ("WISCO") to prepare a Technical Report on the Preliminary Economic Assessment ("PEA") for the Full Moon Project (the "Project"), located in Québec. SRK was assigned to prepare the mineral resource estimate and Met-Chem was to develop the mine plan and the in-pit resource estimate. Soutex was to provide their expertise for the metallurgical testing. The environmental considerations and permitting was carried out by WSP Canada Inc. ("WSP").

The financial analysis for the Project was developed by Michel Bilodeau and the product-selling price was developed using market studies provided by WISCO Century Sunny Lake Iron Mines Limited.

Site visits by CIMA+ and Soutex were carried out May 16 and 17, 2012. Met-Chem visited the site September 19, 2012

1.2 Property Description and Ownership

The Sunny Lake project is subdivided into the Rainy Lake and Lac Le Fer properties that are located 80 kilometers and 65 kilometers northwest of the town of Schefferville, Québec, respectively. The Sunny Lake project consists of 864 claims covering an area of 422.40 square kilometers (42,240 hectares) within two non-contiguous claim blocks. The mineral rights exclude surface rights and were acquired by staking. All claims are located on Crown lands. The Rainy Lake property is located entirely within the Province of Québec, including the mineral resource reported herein. As of the date of this report, Century has 81.4% interest and WISCO has 18.6% interest in the Sunny Lake project.

1.3 Geology and Mineralization

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The Rainy Lake property is located on the extreme western margin of the Labrador Trough adjacent to Archean basement gneisses. The Labrador Trough is a sequence of Proterozoic sedimentary rocks, which includes the Sokoman Formation within the Knob Lake Group. The Sokoman Formation is an iron formation consisting of a continuous stratigraphic unit that thickens and thins throughout the Labrador Trough.

The thickness of the Sokoman Formation varies between 120 and 240 meters and is a typical Lake Superior type iron-formation (taconite) consisting of banded sedimentary rock composed principally of layers of iron oxide, magnetite and hematite. Iron-rich bands are intercalated with cherty bands composed of variable amounts of silicate, carbonate, sulphide, ferruginous slaty iron formation, and carbonaceous shale. The Sokoman Formation is subdivided into eight stratigraphic subunits: Lean Chert ("LC"), Jasper Upper Iron Formation ("JUIF"), Green Chert ("GC"), Upper Red Chert ("URC"), Pink Grey Chert ("PGC"), Lower Red Chert ("LRC"), Lower Red Green Cherty ("LRGC"), and Lower Iron Formation ("LIF").

On the Rainy Lake property the Sokoman Formation is thickened by shallow east dipping northwest-southeast thrust faults and is gently folded resulting in unusual thickness of iron mineralization reaching 400 meters locally. The area investigated by drilling was named the Full Moon iron deposit.

1.4 Exploration and Drilling

Exploration activities on the Rainy Lake property between 2009 and 2012 included an airborne magnetic geophysical survey, geological mapping, composite chip sampling of outcrops, a mineralogical study, ground gravity surveys, a LiDAR survey and core drilling. Between 2011 and 2012, WCSLIM drilled 147 core boreholes (30,932 meters) in an area approximately 10.5 by 3.5 kilometers.

In the opinion of SRK, the sampling procedures used by WCSLIM conform to industry best practice and the resultant drilling pattern is sufficiently dense to interpret the geometry and the boundaries of the iron mineralization with confidence. All drilling sampling was conducted by appropriately qualified personnel under the direct supervision of appropriately qualified geologists.

1.5 Mineral Processing and Metallurgical Testing

1.5.1 PEA Study Metallurgical Testwork

In 2012-2013 COREM performed metallurgical testwork on drill core samples from seven (7) lithology samples from the Rainy Lake Property: Jasper Upper Iron Formation – strongly magnetic ("JUIF-High"), Jasper Upper Iron Formation – weakly magnetic ("JUIF-Low"), LRC, PGC, URC, LRGC and GC. Based on the grindability testwork results, all tested lithologies were classified as hard or very hard.

Characterization testwork showed that all the lithology units exhibit a concentration of magnetite between 18.0 % and 28.9 % except the GC lithology (3.3 % and the JUIF-Low lithology (8.9 %).

Mineralization Liberation Analysis ("MLA") showed that the main gangue mineral was quartz and that the content in iron oxides (valuable iron) varied between 30 and 50 %, except for GC (<7 %). The iron distribution showed that all the samples contained at least 85 % of the iron as valuable iron; except the GC sample (less than 30 %).

Based on the Dense Media Separation ("DMS") results, it was concluded that gravity separation was not an appropriate concentration technique for the samples.

Liberation Davis Tube tests show that a target grind size of around 35-45 µm would be necessary to obtain a final concentrate with the required 4.5 % silica grade. At this grind size, a magnetite recovery of 96-98 % was obtained, except for the GC lithology unit for which magnetite recovery is in the 80-90 % range.

1.5.1.1 Magnetite Plant Benchscale Beneficiation Testwork

Cobber tests on samples ground at 100 % passing -4.0,-2.8 and 2.0 mm showed that all lithology units have a mass rejection of 15-20 % for a magnetite recovery of 98-99 % (except JUIF-Low which has a mass rejection of around 48 % for a magnetite recovery of 93-95 %). Based on these results, benchmarking with existing operations and after discussions with High Pressure Grinding Rolls ("HPGR") vendors, the target particle size selected for dry cobbing was 100 % passing 3 mm.

To produce material for the next testwork steps (flotation and pelletizing), a semi-continuous mini pilot with cobber, regrinding, rougher and finisher Low Intensity Magnetic Separators ("LIMS") was used. Weight recoveries could not be confirmed but the production showed that it was possible to reach the 4.5 % SiO_2 grade.

Preliminary reverse flotation tests on the magnetic 4.5 % SiO_2 concentrate permitted concentrates at 1.5 % SiO_2 to be produced. Results showed that the optimization and regrinding of the rougher flotation froth is required to increase recoveries.

1.5.1.2 Hematite Plant Benchscale Beneficiation Testwork

Beneficiation testwork was conducted on the non-magnetic products from the semi-pilot to evaluate the potential iron recovery of a hematite scavenging plant.

Concerning the Wet High Intensity Magnetic Separator ("WHIMS") tests, iron recoveries of 76-89 % were obtained with a mass rejection of 43-62 %, showing that WHIMS could be used as a rougher to treat the non-magnetic tails.

Selective flocculation and reverse flotation tests were too preliminary to permit a final concentrate to be produced.

1.5.1.3 Pelletizing Testwork

Pelletizing tests (balling tire test and basket test) were conducted at COREM on the composite Wet LIMS concentrate produced by the semi-pilot to investigate the suitability of the ore for producing commercial grade pellets. Three (3) blast furnace pellet chemistries were tested: two (2) acid pellets and one (1) fluxed pellet. After basket firing, all three (3) pellet samples showed good physical and metallurgical properties.

1.5.1.4 Process Flowsheet Development

The results from the above-mentioned testwork, as well as historical test data and adjacent properties' process information, were used to develop a preliminary process flowsheet for the Full Moon deposit. The selected flowsheet has the following features:

- Two (2) stages of crushing followed by a grinding stage via HPGRs are required in order for the ROM to reach the optimum grain size for processing;
- The magnetite beneficiation process consists of a three (3) stage magnetic separation circuit with regrinding after the cobber stage;
- To produce a Low Silica Concentrate ("LSC") from the magnetite concentrate, a two (2) stage flotation circuit with regrinding of the rougher flotation froth is required;
- A scavenging hematite plant recovers the cobber and rougher LIMS tailings. The circuit includes the following steps: regrinding, wet high intensity magnetic separation, and flotation;
- A two (2) stage flotation circuit with regrinding of the rougher flotation froth is required on the hematite concentrate to produce a LSC.

This flowsheet with a magnetite plant and a scavenging hematite plant has the advantage of maximizing the iron recovery from the Full Moon deposit.

1.5.2 Weight Recovery Model

A weight recovery model was developed for the above proposed flowsheet using the geological Davis Tube results database and the metallurgical testwork results. Table 1.1 presents the total weight recovery correlation obtained for each lithology.

Table 1.1 – Total Weight Recovery Models per Lithology

Samples	Correlation	R^2
JUIF	Total WR = 1.0411 x Feed Fe_Tot + 3.3655	R^2 = 0.4710
LC	Total WR = 1.5992 x Feed Fe_Tot - 12.729	R^2 = 0.9007
LRC	Total WR = 1.4233 x Feed Fe_Tot - 0.9894	R^2 = 0.9504
LRGC	Total WR = 1.7700 x Feed Fe_Tot - 23.990	R^2 = 0.6457
PGC	Total WR = 1.3293 x Feed Fe_Tot + 0.8395	R^2 = 0.9351
URC	Total WR = 0.9113 x Feed Fe_Tot + 8.4630	R^2 = 0.5568
GC	Total WR = 1.3285 x Feed Fe_Tot - 16.060	R^2 = 0.4885

1.5.3 Process Plant Feed Design Criteria

Since the block model does not provide the magnetite or the hematite content for each block but only the total iron feed grade, the geological Davis Tube ("DT") results database was processed to select the plant magnetite feed characteristics. A filtration using a cut-off Davis Tube Weight Recovery ("DTWR") of 18 % and a cut-off concentrate SiO_2 of 8 % was conducted in order to obtain an average concentrate SiO_2 of 4.5 %. This gave a feed grade of 31.3 % total Fe and a 27 % magnetite grade. This composition corresponds to an average DTWR of 27.1 % and a hematite plant weight recovery of 10.2% for a total weigh recovery of 37.3 %.

1.6 Mineral Resource Estimate

The mineral resource model presented herein represents the first resource evaluation prepared for the Full Moon iron deposit. The mineral resource model considers 121 core boreholes drilled by WCSLIM during the period of 2011 to 2012. The resource evaluation work was completed by Filipe Schmitz Beretta under the supervision of Mr. Mark Campodonic, MAusIMM (CP#225925) and Dr. Jean-Francois Couture, P.Geo. (OGQ#1106, APGO#0197). The effective date of the Mineral Resource Statement is October 22, 2012.

The mineral resource estimation process was a collaborative effort between SRK and WCSLIM staff. WCSLIM provided to SRK an exploration database and a geological interpretation comprising a series of vertical cross sections through the areas investigated by core drilling. The geology model, geostatistical analysis, variography, selection of resource estimation parameters, construction of the block model, and the conceptual pit optimization work were completed SRK. The current drilling information is sufficiently reliable to interpret with confidence the boundaries of the Sokoman Formation stratigraphy and the assaying data is sufficiently reliable to support mineral resource estimation.

A three dimensional geological model honouring drilling data was constructed for eight members of the Sokoman Formation (LC, JUIF, GC, URC, PGC, LRC, LRGC and LIF). Each lithological unit was considered as separate domains for resource modelling and grade estimation.

The mineral resources were modelled using a geostatistical block modelling approach constrained by the subunits of the Sokoman Formation. A block model rotated 150 degrees around the vertical axis was constructed. The parent block size was set at 100 meters by 100 meters by 10 meters (X, Y, and Z, respectively). The subcell function of CAE Studio 3 was applied. Only parent blocks were estimated.

Sample data were composited to 5-meter composites and extracted for geostatistical analysis and variography. The JUIF, URC, PGC, LRC and LRGC domains are those considered as mineralized and were estimated. The LC and GC units are considered as waste. The block model was populated with common major oxides (Fe, SiO_2, Al_2O_3, P_2O_5, MnO and loss on ignition) and specific gravity using ordinary kriging. Variables were estimated in each subunit separately with estimation parameters derived from variography informed from a combined JUIF, URC, PGC, LRC and LRGC dataset. Subunit boundaries were considered hard boundaries. Three estimation runs were used considering increasing search neighbourhoods and less restrictive search criteria. The first search was based on the iron variogram full ranges. The second search considered search neighbourhoods set at twice the first. For the third search the neighbourhood was inflated to 100 times the first search to ensure that all the blocks were estimated. All domains were estimated using dynamic anisotropy, in CAE Studio 3, to assist the interpolation in areas of folding.

Block model quantities and grade estimates were classified according to the CIM Definition Standards on Mineral Resources and Mineral Reserves (November 2010). SRK is satisfied that the geological model for the Full Moon iron deposit honours the current geological information and knowledge. The location of the samples and the assaying data are sufficiently reliable to support resource evaluation and do not present a risk that should be taken into consideration for resource classification. Blocks classification considered three main criteria: geological continuity, grade continuity, and block estimation quality.

No blocks were classified as Measured. An Indicated classification was assigned to contiguous volumes of mineralisation informed by boreholes spaced at 400 by 500 meters or less and blocks estimated during the first estimation run with a slope of regression greater than or equal to 0.6. An Inferred classification was assigned to blocks estimated using composites from at least 2 boreholes by any of the three estimation runs and are located not farther than 500 meters from the last boreholes in all directions and to a depth not exceeding 400 meters. All other model blocks were not categorized.

SRK considers that the iron mineralization delineated by core drilling is amenable to open pit extraction. To assist with determining which portions of the modelled iron mineralization show “reasonable prospect for economic extraction” from an open pit, and to assist with selecting reasonable reporting assumptions, SRK used a pit optimizer to develop conceptual open pit shells using reasonable assumptions derived from similar projects. In absence of specific metallurgical data for each resource domain, SRK used average recovery information sourced from nearby similar taconite projects targeting the Sokoman Formation. After review, SRK considers that the iron mineralization located within a resulting conceptual open pit shell above a cut-off grade of 20 percent total iron satisfies the definition of a mineral resource and thus can be reported as a mineral resource.

The Mineral Resource Statement presented in Table 1.2 was prepared by Filipe Schmitz Beretta under the supervision of Mark Campodonic (CP#225925) and Dr. Jean-Francois Couture, P.Geo. (OGQ#1106, APGO#0197). Mr. Campodonic and Mr. Couture are independent Qualified Persons as this term is defined by National Instrument 43-101. The effective date of the Mineral Resource Statement is October 22, 2012 and it was published on SEDAR on December 6, 2012

Table 1.2 – Mineral Resource Statement*, Full Moon Iron Deposit, Rainy Lake Property, Sunny Lake Project, Québec, SRK Consulting (Canada) Inc. (October 22, 2012)

Domain	Volume (Mm^3)	Quantity (Mt)	SG	Fe (%)	SiO_2 (%)	Al_2O_3 (%)	P_2O_5 (%)	P** (%)	MnO (%)	Mn** (%)	LOI (%)
Indicated Mineral Resources											
JUIF	1,109.4	3,562.8	3.21	29.45	45.06	0.50	0.03	0.02	0.90	0.70	5.86
URC	235.4	777.1	3.30	33.51	40.31	0.12	0.02	0.01	0.96	0.75	5.37
PGC	399.6	1,314.8	3.29	31.3	43.31	0.12	0.02	0.01	0.61	0.47	5.01
LRC	309.2	997.0	3.22	30.58	45.71	0.14	0.02	0.01	0.52	0.40	4.01
LRGC	194.7	607.9	3.12	27.4	47.13	0.17	0.02	0.01	0.67	0.52	6.52
Total Indicate	**2,248.2**	**7,259.6**	**3.23**	**30.18**	**44.52**	**0.31**	**0.03**	**0.01**	**0.78**	**0.61**	**5.46**
Inferred Mineral Resources											
JUIF	683.0	2,185.2	3.20	29.17	45.14	0.48	0.03	0.02	0.97	0.75	5.99
URC	235.1	787.1	3.35	33.35	40.69	0.18	0.02	0.01	0.93	0.72	5.12
PGC	547.3	1,773.2	3.24	31.14	43.90	0.14	0.02	0.01	0.58	0.45	4.70
LRC	690.1	2,239.4	3.25	30.43	45.71	0.14	0.02	0.01	0.52	0.40	3.98
LRGC	543.5	1,708.6	3.14	27.22	47.38	0.21	0.02	0.01	0.65	0.51	6.44
Total Inferred	**2,699.0**	**8,693.5**	**3.22**	**29.86**	**45.10**	**0.24**	**0.02**	**0.01**	**0.71**	**0.55**	**5.23**

* Reported at a cut-off grade of 20 percent total iron inside a conceptual pit envelope optimized considering reasonable open pit mining, processing and selling technical parameters and costs benchmark against similar taconite iron projects and a selling price of US$110 per dry metric tonne of iron concentrate. All figures are rounded to reflect the relative accuracy of the estimates. Mineral resources are not mineral reserves and do not have a demonstrated economic viability.

** Converted from estimated oxide

1.7 Mineral Reserve Estimate

Since this report is a Preliminary Economic Assessment report, no Mineral Reserves are estimated. The Mineral Resources have been classified as In-pit Mineral Resources.

1.8 Mining Methods

The mining method selected for the Project is a conventional open pit, drill and blast, truck and shovel operation with 10 meter high benches. Topsoil and overburden will be stripped and stockpiled for future reclamation use. The mineralization and waste rock will then be drilled, blasted and loaded into rigid frame haul trucks with hydraulic shovels. The mineralized material will be hauled to the primary crushers and the waste rock will be hauled to the waste rock pile. The mine will operate 365 days per year, 24 hours per day.

Since mining all of the Mineral Resources would result in a 290 year mine life at the planned production rate of 20 Mt of concentrate per year, it was decided that the PEA would be limited to a 30 year mine life. Pit optimization techniques were used to determine the area for the pit design that would provide for a

30 year mine life. This ensured that the pit design would include a considerable amount of high grade material, have a low stripping ratio and have relatively short haul distances to the crushers and dumps. The area selected for the pit design also accounted for minimizing the environmental disturbance.

The open pit design was done with an inter-ramp angle of 52^{o} for the configuration of the final pit wall and a haul road width of 31 m. The 30 year open pit includes 1,283 Mt of Indicated Mineral Resources at a Total Fe grade of 30.8 % (Weight Recovery of 36.9 %) and 327 Mt of Inferred Mineral Resources at a Total Fe grade of 30.7 % (Weight Recovery of 37.7 %). In order to access these Mineral Resources, 90 Mt of overburden, 9 Mt of Menihek Shale and 54 Mt of low grade mineralization must be mined. This total waste quantity of 153 Mt results in a stripping ratio of 0.1 to 1.

A 30 year production schedule (mine plan) was developed for the Project, which targets the production of 20 Mt of iron concentrate per year. The mine plan was used to estimate the fleet of mining equipment which resulted in 20 haul trucks (227 tonne), 3 hydraulic shovels (26.5 m^3 bucket), 2 wheel loaders (1,100 kW), 3 production drills as well as a fleet of support and service equipment. The peak workforce for the mine reaches 276 employees.

1.9 Recovery Methods

The process design for the Full Moon Project concentration plant is based on laboratory testwork and benchmarks from nearby developing projects.

1.9.1 Concentrator

The process plant is designed to produce 20.0 Mtpy of high silica content (4.5 %) concentrate over a 30-year mine life. The Run of Mine ("ROM") is calculated based on a magnetite plant weight recovery of 27 % and a hematite plant weight recovery of 9.2 %. A design factor of 20 % is applied on nominal requirements to ensure that the process equipment has enough capacity to take care of the expected feed variation.

The production of LSC (<1.5 %) concentrate leads to a weight recovery loss of 3 % and a production of 18.3 Mtpy of concentrate.

Two (2) stages of crushing followed by a grinding stage via HPGR are required in order for the ROM to reach the optimum grain size for processing. The magnetite beneficiation process then consists of a magnetic separation circuit, followed by a flotation circuit to produce a LSC (1.5 %).

The magnetic separation circuit is a three (3) stage process whose purpose is to separate the magnetite from the non-magnetic material. Grinding is added after the cobber magnetic separation stage in order to increase particle liberation. The regrind product is fed to a rougher LIMS whose role is to immediately reject the non-magnetic particles that have been liberated through grinding, before re-circulating them into the mill. This reduces the grinding energy requirements. The regrind product is then further processed in a finishing magnetic step followed by a final classification to achieve the targeted iron and High Silica Concentrate ("HSC") (4.5 %) targets. To produce a magnetic LSC (1.5 %), the HSC undergoes a reverse flotation concentrating step and the rougher froth is further reground. The reground product is fed to a magnetic separator to remove the liberated silica and the target LSC is then achieved via a final flotation step.

The hematite plant is a scavenging plant that treats the magnetite plant cobber and the rougher LIMS tailings. The material is first reground in order to increase particle liberation. Hematite is then recovered in a high intensity magnetic separation step and sent to a desliming thickener for dewatering and for slime particles removal. The target HSC is then achieved via a final flotation step. Similar to the magnetite plant, the HSC has to undergo a flotation step and further regrinding to produce a low silica grade concentrate. The reground product is sent to a final flotation step to produce the LSC.

Finally, the magnetite and hematite concentrates are combined, thickened, filtered and dried for transport and pellet production.

1.9.2 Pellet Plant

The pellet plant is designed to produce 17.0 Mtpy of fired pellets in two (2) completely identical and independent processing lines. The production rate is based on induration machines designed to process magnetic concentrates. When fed by a blend of magnetite and hematite concentrates, the pellet plant production rate is expected to be lower or coke breeze addition may be required to maintain the production rate; this will have to be confirmed by further testwork.

The pellet plant processes the iron concentrate as received from the concentrator without any beneficiation plant to reduce impurity levels. There is no tailings stream at the pellet plant and no process water effluent is expected.

The pellet plant is designed to offer sufficient flexibility to produce many types of pellets from the low and high silica concentrates produced at the concentrator. The design pellet mix is:

- Direct Reduction Iron pellet ("DR") with low silica and additives content;
- High Silica Flux pellet ("HSF") with high silica and additives content.

In each processing line, concentrate is reground in HPGRs to control the concentrate Blaine in the appropriate range for balling. The pellet plant also includes dry grinding of the additives (dolomite and limestone), bentonite and coke breeze. Concentrate, bentonite, additives and coke breeze are then mixed in the proportions required by the pellet type produced. The mixed material is conveyed to the balling area. A conventional arrangement for the balling discs, the single roller deck screens and the fine and coarse green balls return conveyor is proposed. One-size green balls feed the indurating straight travelling grate. Pellets are conveyed outside the pellet plant onto product piles where the reclaiming system allows their retrieval for expedition

1.10 Project Infrastructures

The Full Moon Property is located 88 km (by the projected road) northwest of the town of Schefferville, Québec. The waste and overburden dumps, the crushing plant as well as the buildings, such as concentrator, offices and workshops, are located west of the planned open pit. Drainage ditches will be constructed around the open pit and dumps to direct water runoff to settling ponds to avoid contamination. The mineralized material will be hauled by the mine haul trucks to the two gyratory crushers about 2 km from the concentrator. A haulage road will be constructed between the mine and the crushers. All crushed material will be sent, via two conveyors (1.69 km and 1.24 km) to the two cone crushing and screening plants, stockpiled, and, subsequently reclaimed and transported to the concentrator via a short conveying system.

The annually produced 20 Mt of iron concentrate (10 Mt per line of the concentrator) will be conveyed to two 60,000 tonne storage silos or to a combined emergency stockpile. The stored iron concentrate will be loaded in train cars and transported by rail via the newly constructed railway loop. This railway loop will tie-in to the new WCSLIM railway and the concentrate will be hauled and ultimately tie-in to an existing railway system near Schefferville.. An accommodation camp will be built about 1.5 km from the concentrator. A 450 km long, new 315 kV power line will be built starting at the LG4/Tilley substation.

Four options were analyzed, namely:

- Option 1: High Silica Concentrate without pelletizing plant;
- Option 2: Low Silica Concentrate without pelletizing plant;
- Option 3: High Silica Concentrate with a pelletizing plant; and
- Option 4: Low Silica Concentrate with a pelletizing plant.

The concentrate will be transported via the new, 91 km long railway line, first to Schefferville and subsequently, via the existing TSH and QNSL railroads from Schefferville to Sept-Iles, where the ore cars (gondolas) will be transferred to a new multi-user terminal. From the multi-user terminal, the iron concentrate could be sent, via a conveying system, to pellet plants or to the port facilities to be loaded directly into vessels.

1.11 Market Studies and Pricing

The estimation of the selling prices was based on a long term price forecast at US$95 DMT (Fe 62% Fines Tianjin Port CRF Spot). From that price, various premiums were applied to reflect the type of product and content of each product. Depending of the option retained, there are four potential products. Table 1.3 shows the estimated price of products.

Table 1.3 – Products Selling prices

Product	CFR Price $US/DMT	Shipping Cost $US/DMT	FOB Price $US/DMT	FOB Price $CAD/DMT
Low Silica Product				
DR Pellet	140.00	15.00	125.00	156.25
Low Silica Concentrate	118.00	15.00	103.00	128.75
High Silica Product				
HSF Pellet	135.00	15.00	120.00	150.00
High Silica Concentrate	112.00	15.00	97.00	121.25

1.12 Environment Studies, Permitting and Social or Community Impact

The Project will be subject to Environmental Assessment ("EA") in accordance with provincial and federal requirements. Following release from the provincial and federal EA processes, the project will require a number of approvals, permits and authorizations prior to initiation and throughout all stages in the life of the project. In addition, WISCO Century Sunny Lake Iron Mines Ltd will be required to comply with any other terms and conditions associated with the EA release issued by the provincial and federal regulators. Additional details are provided in Section 20.

1.13 Capital and Operating Costs

The capital and operating costs are shown for four (4) different options as described below:

- Option 1: High Silica Concentrate without pelletization;
- Option 2: Low Silica Concentrate without pelletization;
- Option 3: High Silica Concentrate with pelletized;
- Option 4: Low Silica Concentrate with pelletized.

The capital cost of the project is the cost for the initial development of the project.

Table 1.4 shows the summary of the estimated capital cost.

Table 1.4 – Summary of Capital Cost Estimate

WBS No	Description	Option 1 ($'000) (Preferred)	Option 2 ($'000)	Option 3 ($'000)	Option 4 ($'000)
Direct Cost					
00000	Project General	655,681	671,999	655,681	671,999
11000	Mine-Equipment	187,527	187,527	187,527	187,527
14000	Full Moon Mine	54,813	54,813	54,813	54,813
34000	Concentrator	2,513,852	2,626,031	2,513,852	2,626,031
44000	Tailings	450,379	450,379	450,379	450,379
54000	Railroad & Rail Yard	441,101	441,101	441,101	441,101
66000	Sept-Îles Pellet Plant	0	0	1,678,807	1,678,807
74000	Infrastructures	859,802	859,802	859,802	859,802
	Total Direct Cost	**5,163,154**	**5,291,651**	**6,841,962**	**6,970,458**
Indirect Costs					
91000	EPCM Management	286,644	293,778	286,644	293,778
92000	Construction Services	137,548	140,971	137,548	140,971
93000	Construction Indirect	151,013	154,771	151,013	154,771
C0000	Contingency	603,051	618,059	603,051	618,059
E0000	Escalation	398,973	408,902	398,973	408,902
R0000	Risk	418,921	429,347	418,921	429,347
	Total Indirect Cost	**1,996,149**	**2,045,827**	**1,996,149**	**2,045,827**
Other Costs					
	Mine – Pre-Production	48,013	48,013	48,013	48,013
	Total Project Cost	**7,207,316**	**7,385,492**	**8,886,124**	**9,064,299**

The summary of the annual costs and unit costs per tonne of concentrate and per tonne of pellet of an average year of operations, are shown in Table 1.5, Table 1.6, Table 1.7 and Table 1.8 for each of the four options.

Table 1.5 – Summary of an Average Year of Operations per Area – Option 1 (Preferred)

Aera	Annual Cost ($'000)	Unit Cost ($/t conc.)
Mining	111,975	5.60
Concentrating	259,544	12.98
Tailings	14,608	0.73
General and Administration	53,236	2.66
Rail Transportation & Port	557,625	27.88
Pellet Plant	0	0
Total	**996,988**	**49.85**

Table 1.6 – Summary of an Average Year of Operations per Area – Option 2

Aera	Annual Cost ($'000)	Unit Cost ($/t conc.)
Mining	111,975	6.12
Concentrating	329,084	17.98
Tailings	14,608	0.80
General and Administration	53,233	2.91
Rail Transportation & Port	510,363	27.89
Pellet Plant	0	0
Total	**1,019,263**	**55.70**

Table 1.7 – Summary of an Average Year of Operations per Area – Option 3

Aera	Annual Cost ($'000)	Unit Cost ($/t conc.)	Unit Cost ($/t Pellets)
Mining	111,975	5.60	5.20
Concentrating	259,544	12.98	12.06
Tailings	14,608	0.73	0.68
General and Administration	53,236	2.66	2.48
Rail Transportation & Port	557,625	27.88	25.91
Pellet Plant	190,198		11.19
Total	**1,187,186**	**49.85**	**57.52**

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Table 1.8 – Summary of an Average Year of Operations per Area – Option 4

Aera	Annual Cost ($'000)	Unit Cost ($/t conc.)	Unit Cost ($/t Pellets)
Mining	111,975	6.12	5.87
Concentrating	329,084	17.98	17.24
Tailings	14,608	0.80	0.77
General and Administration	53,233	2.91	2.79
Rail Transportation & Port	510,363	27.89	26.74
Pellet Plant	182,427	0	10.73
Total	**1,201,690**	**55.70**	**64.14**

The capital expenditures during the life of the mine ("the Sustaining Capital") are required to maintain or upgrade the existing asset and to continue the operation at the same level of production. They are charged as an operating cost and are shown in Tables 21.10 to 21.13.

Mine closure costs for the Project are estimated at approximately M$178.21 spread over three years and must be secured in a trust fund at the beginning of mining operations. It is assumed that trust fund payments are made in the last pre-production year and in the first two years of operation in the proportions of 50/25/25 %, respectively.

1.14 Economic Analysis

A preliminary economic analysis has been carried out for the Full Moon Project using a cash flow model. The model is constructed using annual cash flows in constant first-quarter 2015 Canadian dollars and is based on a combined iron concentrate/pellet production of some 20 million tonnes per year over a mine life limited to 30 years. Four production options are considered: HSC only, HSC & HSF pellets, LSC only and LSC & DR pellets.

The selling prices of the mine products are based on a 62% iron concentrate price forecast of US$95 per tonne (CFR China). An exchange rate of US$0.80 per CAD is assumed to convert the revenue estimates into Canadian dollars.

The financial assessment is carried out on a "100% equity" basis, i.e. the debt and equity sources of capital funds are ignored. No provision is made for the effects of inflation. Results are given before and after taxation. Current Canadian tax regulations are applied to assess the corporate tax liabilities while the recently proposed regulations in Quebec (Bill 55, December 2013) are applied to assess the mining tax liabilities.

The summary of the economic analysis is shown in Table 1.9.

Table 1.9 – Summary of Financial Results

Description	Units	Option 1 (Preferred)	Option 2	Option 3	Option 4
Total Revenue FOB Sept-Îles (LOM)	M$	72,384.3	70,328.5	91,316.2	86,973.1
Total Operating Costs (LOM)	M$	29,759.3	30,424.2	35,436.6	35,869.5
Total Pre-production Capital Costs	M$	7,207.3	7,385.5	8,886.1	9,064.3
Total Sustaining Capital Costs (LOM)	M$	658.0	658.0	658.0	658.0
Initial Working Capital	M$	369.9	378.6	439.5	445.4
Mine Closure Costs	M$	178.2	178.2	178.2	178.2
Salvage Value	M$	358.0	366.9	441.9	450.8
BEFORE TAX					
Total Cash Flow	M$	34,939.5	32,049.5	46,599.2	41,654.0
Payback Period	Years	5.7	6.3	5.4	6.0
NPV @ 8%	M$	5,771.0	4,806.7	8,196.0	6,626.3
NPV @ 6%	M$	9,233.6	8,026.4	12,772.2	10,779.7
NPV @ 10%	M$	3,395.2	2,604.2	5,048.3	3,779.1
IRR	%	**15.2**	**13.9**	**16.2**	**14.6**
AFTER TAX					
Total Tax Payments (LOM)	M$	12,360.0	11,170.1	16,321.7	14,323.0
Total Cash Flow	M$	22,579.5	20,879.4	30,277.5	27,330.9
Payback Period	years	6.3	6.8	5.9	6.5
NPV @ 8%	M$	2,965.3	2,335.8	4,418.9	3,409.1
NPV @ 6%	M$	5,326.2	4,560.4	7,539.7	6,285.5
NPV @ 10%	M$	1,334.1	802.8	2,258.5	1,423.5
IRR	%	**12.4**	**11.4**	**13.2**	**12.0**

Both the project's net present value and internal rate of return are more sensitive to changes in operating costs than to changes in capital costs. As expected however, the project's financial performance is most sensitive to changes in selling price. See Section 22.2 for a description of the key economic, operating and technical assumptions used in preparing the economic analysis.

The economic analysis contained in this report is preliminary in nature. It incorporates inferred mineral resources that are considered too geologically speculative to have the economic, considerations applied to them that would enable them to be categorized as mineral reserves. It should not be considered a prefeasibility or feasibility study. There can be no certainty that the estimates contained in this report will be realized. In addition mineral resources that are not mineral reserves do not have demonstrated economic viability.

1.15 Recommendations

1.15.1 Geology

The block model constructed by SRK is sufficiently reliable to support mine planning and allow evaluation of the economic viability of a mining project. On this basis, the work program recommended by SRK includes:

- Infill drilling along the more widely spaced drilling areas to reduce spacing to 200 by 250 meters spacing with 70 to 90 core boreholes;
- Preliminary rock geotechnical investigations (10 to 20 boreholes); and
- Geology and mineral resource modelling after reception of all Davis Tube testing results.

1.15.2 Mining

For the next phase of the project Met-Chem recommends that:

- The Mineral Resource Estimate be updated to consider the results of the Davis Tube and Satmagan tests that were completed on the 2012 drillhole assays;
- Geotechnical pit slope analysis be done to determine the appropriate pit wall configuration;
- A geotechnical analysis be prepared to confirm the stability of the dump and stockpile designs;
- Geochemical testwork be carried out on the overburden and waste rock to evaluate if there is a potential for this material to be a generator of acid rock drainage; and
- A hydrogeological study be carried out to estimate the amount of groundwater that is expected to be encountered during the mining operation.

1.15.3 Metallurgy

The benchscale testwork performed during this study led to the definition of the Magnetite Plant flowsheet producing a concentrate at 4.5 % SiO2. To bring the project to the Pre-Feasibility Study level, complementary testwork is required to firm up the Hematite Plant Scavenging flowsheet and the flowsheet sections producing a LSC from the HSC:

- Benchscale testwork including MLA and flotation tests will confirm the LSC circuit flowsheet;
- Benchscale testwork including MLA to confirm regrind size, WHIMS tests, flotation and selective flocculation will be necessary to better define the hematite recovery circuit flowsheet;

- Pelletizing tests will be realised to qualify the feasibility to produce pellets using magnetite hematite HSC and LSC.
- Samples should be collected for the Feasibility testwork:
  - Samples to evaluate the process variability (grindability and magnetite & hematite plant beneficiation confirmation testwork);
  - A large bulk sample representative of the ore body for pilot plant testwork.

1.15.4 Environment and Social Aspects

With respect to environmental considerations, WSP recommends to:

- Carry out the Environmental Assessment as well as any related environmental baseline studies;
- Engage discussions with local community and include additional stakeholders to identify key areas and subjects to be addressed during the advancement of the exploration project and through the future EA phase of the Project; and
- Conduct a geochemical testing to determine Acid Generating/Non-Acid Generating Potential of mineralized rock waste rock and tailings as well at the respective potential for metal leaching/non leaching.

1.15.5 Infrastructures

- Initiate discussions with electric power company (Hydro-Québec) to confirm the power supply options;
- Initiate discussions with multi-user terminal at Sept-Îles; and
- Initiate discussions with rail operators from Schefferville to Sept-Îles.

2 Introduction

2.1 Scope of Study

This Technical Report presents the results of the Preliminary Economic Assessment for the development of the Full Moon Project. The Project is entirely located in Québec, approximately 80 km northwest of Schefferville, Québec. In October 2014, WISCO Century Sunny Lake Iron Mines Limited mandated CIMA+ to prepare the PEA Study. The services of SRK were retained to produce the mineral resource estimate. Met-Chem provided the mine plan and the in-pit resource estimate. Soutex was to provide their expertise for the metallurgical testing and the elaboration of the process. AMEC developed a conceptual design for the tailings pond. The environmental considerations and permitting was carried out by WSP. The preliminary economic analysis was prepared by Mr. Michel L. Bilodeau.

This Report titled “Technical Report on the Preliminary Economic Assessment for the Full Moon Project” was prepared by CIMA+ with contributions by SRK, Met-Chem, Soutex, and WSP. The report follows the guidelines of the “Canadian Securities Administrators” National Instrument 43-101 (effective June 30, 2011), and is in conformity with the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum (“CIM”) Standard on Mineral Resources and Reserves. Table 2.1 shows the responsibilities for each section of the report.

Table 2.1 – Responsibilities of Report Sections

Report Section		Responsible	Comment
Section 1 -	Summary	JST	And other
Section 2 -	Introduction	JST	
Section 3 -	Reliance on Other Experts	JST	
Section 4 -	Property Description and Location	JFC	
Section 5 -	Accessibility, Climate, Local Resources, Infrastructure and Physiography	JST	
Section 6 -	History	JFC	
Section 7 -	Geological Setting and Mineralization	JFC	
Section 8 -	Deposit Type	JFC	
Section 9 -	Exploration	JFC	
Section 10 -	Drilling	JFC	
Section 11 -	Sample Preparation, Analyses and Security	JFC	
Section 12 -	Data Verification	JFC	
Section 13 -	Mineral Processing and Metallurgical Testing	SF	
Section 14 -	Mineral Resource Estimate	JFC	
Section 15 -	Mineral Reserve Estimate	JC	
Section 16 -	Mining Methods	JC	
Section 17 -	Recovery Methods	SF	
Section 18 -	Project Infrastructure	JST	
Section 19 -	Market Study and Contracts	JST	
Section 20 -	Environmental and Social Impact	JSH	
Section 21 -	Capital and Operating Cost	JST	And other
Section 22 -	Economic Analysis	MB	
Section 23 -	Adjacent Properties	JST	
Section 24 -	Other Relevant Data and Information	JST	
Section 25 -	Interpretation and Conclusions	JST	And other
Section 26 -	Recommendations	JST	And other
Section 27 -	References	JST	

MB - Michel Bilodeau, Eng. Independent consultant

JC - Jeffrey Cassoff, Eng. Met-Chem

JFC – Jean-François Couture, P. Geo. SRK

SF - Simon Fortier, Eng. Soutex

JSH – Jean-Sébastien Houle, Eng. WSP

JST - Jean-Sébastien Tremblay, Eng. Cima+

2.2 Sources of Information

Information contained in this report is based on:

- "NI 43-101 Mineral Resource Evaluation, Full Moon Taconite Iron Deposit, Rainy Lake Property, Schefferville, Québec of 0849873 BC Ltd." Prepared by SRK Consulting (Canada) Inc. dated December 6, 2012;
- Information provided by personnel of WISCO Century Sunny Lake Iron Mines Limited.

2.3 Abbreviations in the report

Table 2.2 lists the abbreviations used in the report.

Table 2.2 – Report Abbreviations

Abbreviations	Description
%	Percent Sign
°	Degree
°C	Degree Celsius
$	Canadian dollar
$/h	Canadian dollar per hour
µm	Micrometer
CFR	Cost and Freight (and port of destination)
cm	Centimeter
DMT	Dry metric tonne
EA	Environmental assessment
FOB	Free on board (and port of destination)
ft	Feet
g/cm^3	Gram per cubic centimeter
ha	Hectare
IRR	Internal Rate of Return
kg/t	Kilogram per metric tonne
km	Kilometer
km^2	Square kilometer
km/h	Kilometer per hour
PEB	Pre engineered building
kW	Kilowatt
LiDAR	Laser Illuminated Detection And Ranging
LOM	Life of Mine
m	Meter
m^3	Cubic meter
mm	Millimeter
Mt	Million metric tonnes
Mtpy	Million tonnes per year
M$	Million Canadian dollars
M$/y	Million Canadian dollars per year
NPV	Net Present Value
ROM	Run of mine
t	Metric tonne
t/m^3	Metric tonne per cubic meter

2.4 Site Visit

The following qualified persons for this report personally inspected the Full Moon Property; the dates of the visits are shown in Table 2.3.

Table 2.3 – Site Visits of the Qualified Persons

Qualified persone	Company	Date
Jean-Francois Couture, P.Geo.	SRK	May 15 to 17, 2012
Jeffrey Cassoff, Eng.	Met-Chem	September 19, 2012
Jean-Sébastien Tremblay, Eng.	CIMA+	May 16 to 17, 2012

Each qualified person considers the site visit current, per Section 6.2 of NI 43-101CP, on the basis that the material work completed on the Full Moon Property was reviewed during the site visit, all practices and procedures documented were adhered to and no further work were carried out on the property since 2012.

3 Reliance on Other Experts

CIMA+ prepared this report using documents as noted in Section 27 "References". Any statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of this report.

SRK was responsible for the resource estimate, Met-Chem for the mine design, Soutex for the process definition, WSP for the environmental study and M. L. Bilodeau for the economic analysis.

This report includes technical information, which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

For the purpose of this technical report, CIMA+ has relied mainly on information provided by WISCO Century Sunny Lake Iron Mines Limited for the section entitled Market Studies and Contracts.

The Authors of this Report are not qualified to comment on issues related to legal agreements. The Authors have relied upon the representations and documentations supplied by the Company management. The Authors have reviewed the mining titles, their status, the legal agreement and technical data supplied by WISCO Century Sunny Lake Iron Mines Limited and any public sources of relevant technical information.

The qualified persons are specialists in their respective fields and CIMA+ has no reason to doubt their conclusions and recommendations. The responsibility for the various components of the Summary, Interpretation and Conclusions and Recommendations remains with each qualified persons for their specific area of the scope.

The QPs who prepared this report relied on information provided by experts who are not QPs. The QPs believes that it is reasonable to rely on these experts, based on the assumption that the experts have the necessary education, professional designations, and relevant experience on matters relevant to the technical report.

QP Jean-Sébastien Houle, Eng. has relied upon and disclaims information provided by Mr. Martin Larose, B.Sc., senior biologist and assistant vice-president for environmental studies at WSP. Mr. Larose cumulates 20 years in the environmental area. As project director or project manager, Martin has taken part in numerous impact studies and environmental assessment studies. He has also conducted several fish habitat development and restoration projects as well as many environmental surveys and monitoring studies. Mr. Larose is very knowledgeable with provincial and federal environmental assessment procedures. He was Project Director for the environmental and social impact study on many mining projects in the North-Shore region for ArcelorMittal Mine Canada, Cliffs Natural Resources, Century Iron Ore, Consolidated Thompson (now Cliffs Natural Resources). Martin Larose has provided the environmental baseline data, the permitting process required for the Project based on the information provided by WCSLIM and other available data. The information he provided has been used in Section 20 of this report.

QP M. L. Bilodeau Eng. as relied upon Marc Robert, CPA for the fiscal aspects of the economic analysis.

4 Property Description and Location

The Full Moon iron deposit on the Rainy Lake property, part of the Sunny Lake project, is located in northeastern Québec approximately 80 kilometers northwest of the town of Schefferville, 220 kilometers north of Labrador City, Newfoundland and Labrador, and 600 kilometers north of Sept-Îles, Québec (Figure 4.1).

The centre of the Rainy Lake property is located at approximately latitude 55.40 degrees north and longitude 67.60 degrees west.

4.1 Location

The Sunny Lake project comprises two separate exploration properties named the Rainy Lake and Lac Le Fer properties that are located 80 kilometers and 65 kilometers northwest of the town of Schefferville, Québec, respectively. The Sunny Lake project consists of 864 claims covering an area of 422.40 square kilometers (42,240 hectares) in two non-contiguous claims blocks.

The Rainy Lake property, including the mineral resource of the Full Moon iron deposit reported herein, is located entirely within the Province of Québec.

The Rainy Lake property consists of 378 contiguous map designated claims (18,500 hectares) recorded under the names of WISCO Century Sunny Lake Iron Mines Limited. The Lac Le Fer project consists of 486 contiguous map designated claims (23,739 hectares) registered to WISCO Century Sunny Lake Iron Mines Limited (Figure 4.2).

SRK understands that WISCO Century Sunny Lake Iron Mines Limited is a joint venture company between Century Iron Mines, and WISCO wherein WISCO can earn up to 40 percent of the property.

The claims have not been legally surveyed. Map designated cells are defined on the basis of Universal Transverse Mercator coordinates for the corner points. The location of each corner point of each cell is predefined by the claim staking system maintained by the Ministère des Ressources Naturelles et de la Faune du Québec ("MRNF").

The list of claims, renewal dates, work requirements, and renewal fees as established by the MRNF are summarized in Table 4.1

The tenure information was extracted from the Government of Québec's GESTIM website (as of the date of this technical report).

Figure 4.1 – Project Location

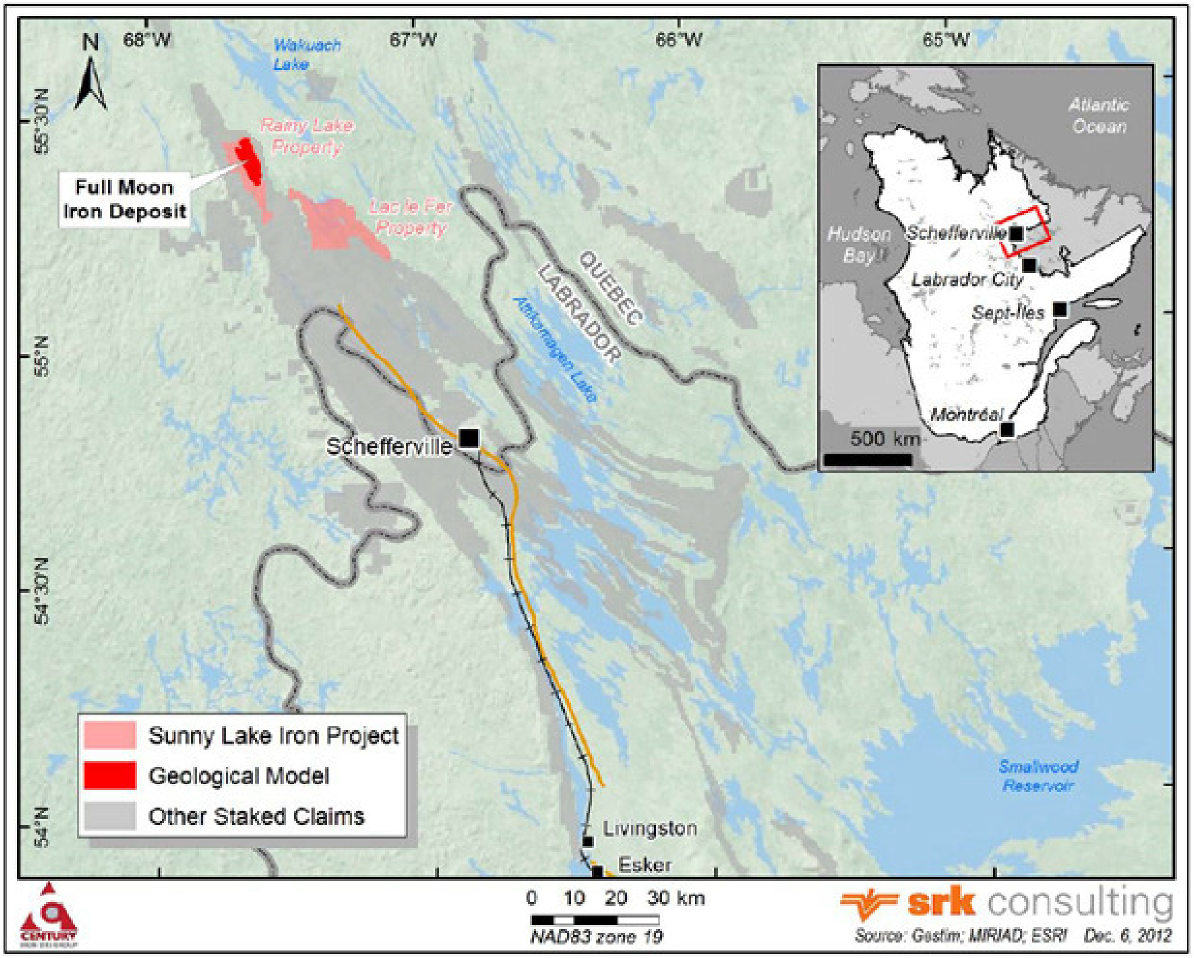
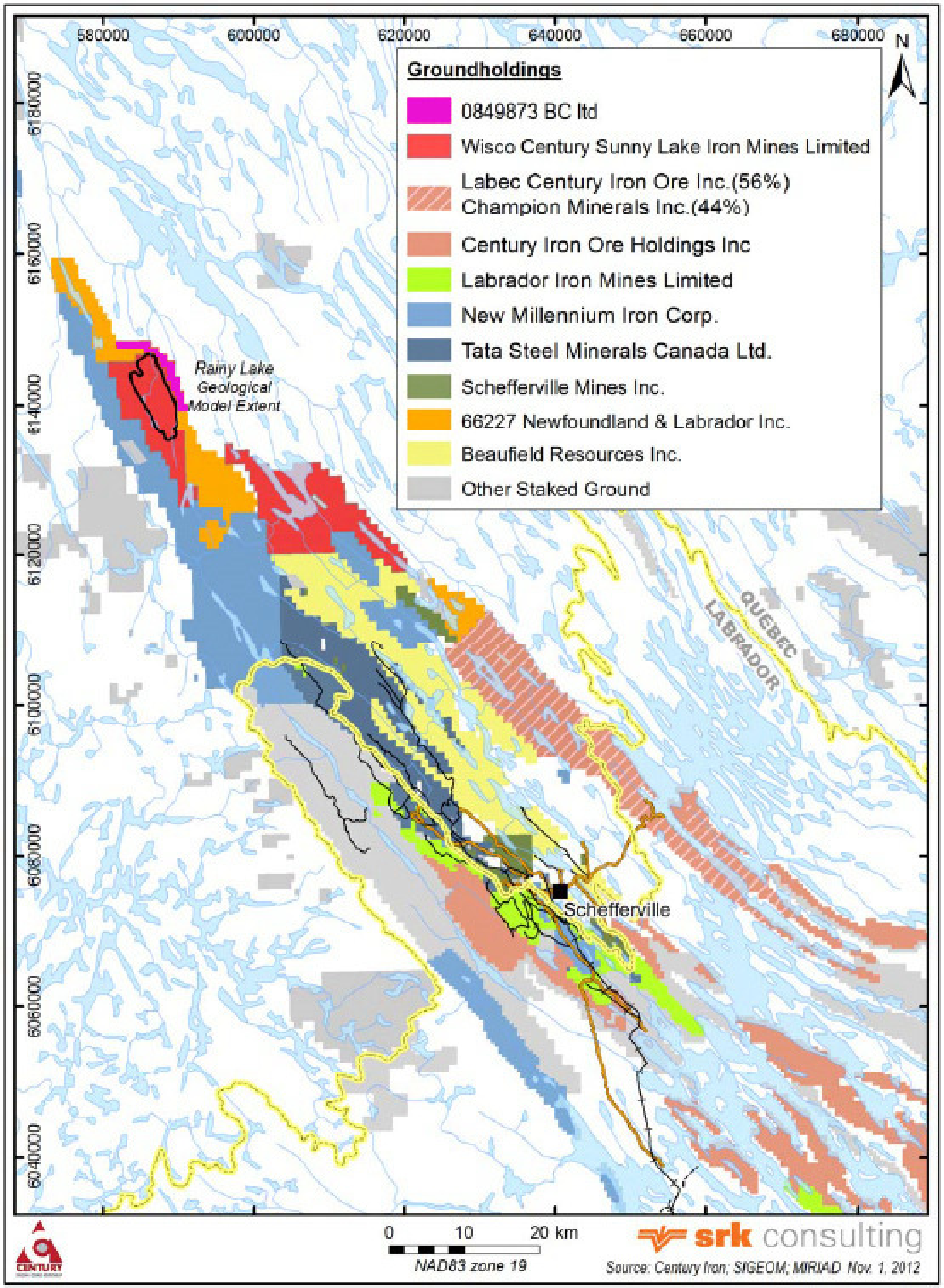


Table 4.1 – Mineral Tenure Summary of the Sunny Lake Property

Project	Registered Compagny	Registration Date	Expiry Date	No. Claims	Area (Ha)	Full Moon Deposit
Rainy Lake	WISCO Century*	23/07/2009	22/07/2015	195	9,549.47	Yes
Rainy Lake	WISCO Century*	20/07/2011	19/07/2015	183	8,951.02	
Lac Le Fer	WISCO Century*	21/07/2009	20/07/2015	59	2,900.66	
Lac Le Fer	WISCO Century*	22/07/2009	21/07/2015	150	7,223.07	
Lac Le Fer	WISCO Century*	24/07/2009	23/07/2015	35	1,717.14	
Lac Le Fer	WISCO Century*	14/04/2010	13/04/2016	55	2,704.45	
Lac Le Fer	WISCO Century*	06/08/2010	05/08/2016	36	1,766.99	
Lac Le Fer	WISCO Century*	04/10/2010	03/10/2016	24	1,181.91	
Lac Le Fer	WISCO Century*	20/07/2011	19/07/2015	127	6,244.96	
				864	**42,239.67**	

*WISCO Century Sunny Lake Iron Mines Limited

Figure 4.2 – Land Tenure Map in the Vicinity of the Sunny Lake Project



4.2 Underlying Agreements

The Sunny Lake project was acquired by staking. SRK is not aware of any back-in rights, payments or other agreements, encumbrances, or environmental liabilities to which the Sunny Lake project could be subject.

On December 19, 2011, Century entered into a joint venture agreement with WISCO on the Sunny Lake project. Under the terms of definitive joint venture agreements, WISCO could earn a 40% interest in the Sunny Lake project, has a 40 percent interest in the Sunny Lake project, including the Rainy Lake property and Full Moon iron deposit, by investing a total of C$40 million in WCSLIM. pursuant to which WISCO will invest 40M$.

4.3 Permits and Authorization

WISCO Century has obtained all permits and certifications required from governmental agencies to allow for surface drilling and exploration activities on the Sunny Lake project. To allow for drilling activities in the province of Québec, WISCO Century obtained an Authorisation de coupe de bois sur un territoire du domaine de l'État où s'exerce un droit minier from the MRNF. This permit allows for the limited cutting of trees for the purpose of exploration activities.

SRK is unaware of any other significant factors and risks that may affect access, title, or the right, or ability to perform the exploration work recommended for the Rainy Lake property.

4.4 Environmental Considerations

The Sunny Lake project and Rainy Lake property consist of undeveloped early stage exploration projects. The project areas are uninhabited and cannot be accessed by road. The area has received limited surface exploration work.

As far as WSP can determine, the environmental liabilities, if any, related to the Schefferville project are negligible.

4.5 Mining Rights in Québec

In Canada, natural resources fall under provincial jurisdiction. In the Province of Québec, the management of mineral resources and the granting of exploration and mining rights for mineral substances and their use are regulated by the Québec Mining Act that is administered by the MRNF. The act also establishes the rights and obligations of claim holders with the view of maximizing development of Québec's mineral resources. Mineral rights are owned by the Crown and are distinct from surface rights. The Québec Mining Act is currently under revision.

4.5.1 The Claim

As defined by the MRNF website (www.mm.gouv.qc.ca), the claim is the only valid exploration right in Québec. The claim gives the holder an exclusive right to search for mineral substances in the public domain, except sand, gravel, clay, and other loose deposits on the land subject to the claim. Each claim also provides access rights to a parcel of land on which exploration work may be performed. However, the claim holder cannot access land that has been granted, alienated, or leased by the Crown for non-mining purposes, or land that is the subject of an exclusive lease to mine surface mineral substances, without first having obtained the permission of the current holder of these rights. A claim holder cannot erect or maintain a construction on lands in the public domain without obtaining, in advance, the permission of the MRNF, unless such a construction is specifically allowed for by ministerial order. An application is not necessary for temporary shelters that are made of pliable material over rigid supports that can be dismantled and transported.

A claim can be obtained by map designation, henceforth the principal method for acquiring a claim, or by staking on lands that have been designated for this purpose. The accepted means to submit a notice of map designation for a claim is through GESTIM Plus (gestim.mines.gouv.qc.ca).

The term of a claim is two years, from the day the claim is registered, and it can be renewed indefinitely, providing the holder meets all the conditions set out in the Mining Act, including the obligation of paying statutory taxes and investing a required minimum amount in exploration work determined by the regulation. The Act includes provisions to allow any amount disbursed to perform work in excess of the prescribed requirements to be applied to subsequent terms of the claim.

4.5.2 Extraction Rights

There are two types of extraction right in Québec: A mining lease for mineral substances and a lease to mine surface mineral substances.

A mining lease is required to undertake commercial mining activity. A claim owner can apply to the mine registrar to obtain a mining lease granting the right to mine mineral substances over areas generally not exceeding 100 hectares (larger areas may be granted by exception). The applicant must demonstrate that the deposit is mineable and submit a written application with conditions set out by regulation and containing a description of the land, including its location, its surface area as determined by a land surveyor and a list of the claim numbers to be covered by the mining lease. The application must also include a report certified by a geologist or an engineer describing the nature and extent of the deposit and its likely value and the payment of the annual rent for the first year of the lease as set out by regulation. Rent is established by regulation and varies based the surface area of the lease, its use (mine or tailings) and its tenure (private or public land).

A mining lease is valid for a period of 20 years and can be renewed for three successive periods of 10 years (total of 50 years) by filling a renewal with the mine registrar and paying renewal fees set out by regulation. The renewal application must include the amount representing the annual rent for the first year of the renewed lease, and a report demonstrating that the holder has engaged in mineral exploitation on the land covered by the mining lease for at least two of the last 10 years for which the lease was valid. The lessee must have also complied with the provisions of the Act and of the regulation during the term of the lease. Thereafter, the MRNF can prolong the lease under conditions it determines.

The lessee of a mining lease or the concession holder has surface access and usage rights, except when the land is used as a cemetery. On public lands, access and usage rights are limited to mining purposes only. If the land covered by the lease or concession was granted or alienated by the province, the lessee or concession holder must obtain the owner’s permission to access the land and carry out work. The concession holder may acquire these rights through amicable agreement or, if necessary, by expropriation. On land leased by the province, the lessee of a mining lease or the holder of a mining concession must obtain the consent of the lessee of the land surface or pay the lessee compensation. In the event of a disagreement, a court can determine this compensation.

The lessee or concession holder may also use adjacent land for his mining activities, in compliance with other laws, in particular those relating to public lands, forests and the environment. On public lands, the lessee or concession holder may purchase or rent land to set up mine tailings or any other facility required for mining purposes. The lessee may also obtain a right of way to install transport routes or tracks, pipelines and water conduits. The location of a mill on land that is covered by a lease or outside its boundaries must be approved by the MRNF, and its location may be subjected an environmental impact assessment, or review in accordance with the Environment Quality Act, in which case the site must be approved by the government.

The lessee or concession holder may use any sand or gravel that is present at the surface of the land covered by their lease or concession for activities related to mining. This permission only applies to public lands that are not subject to an exclusive lease to mine surface mineral substances. Any mining-related activities involving sand or gravel do not require a lease to mine surface mineral substances.

The lessee or concession holder may cut wood on the land of their lease or concession, provided that this wood is only used for the purposes of erecting buildings or carrying out mining-related activities. A forest management permit must be obtained from a regional office of the Forestry Branch of the MRNF. The terms and conditions for issuing the permit vary according to amount of wood to be cut.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

The Full Moon project, part of the Rainy Lake property, is located in northeastern Québec approximately 80 km northwest of the town of Schefferville, Québec and 220 km northeast of the Brisay, Hydro-Québec power dam, Québec. The city of Schefferville is served by an airport with daily flights to Sept-Îles. There are no roads connecting the Rainy Lake property, so the Property can only be accessed by helicopter or floatplane in the summer or ski plane in the winter. There is also a trail (local people use it) up to south of Rainy Lake, and can be accessible by small trucks to Lac Frontiere, about 20 km south of Rainy Lake claims. New Millennium Limited ("NML") built a railroad up to their Timmins Mine, about 20 km NW of Schefferville.

5.2 Climate

Environment Canada's climate station at the Schefferville airport provides comprehensive year round monitoring and with a record period that is sufficient for characterizing long-term climate conditions in the project area. The station is located close to the project site (80 km in a straight line). Therefore, the Schefferville Airport Environment Canada's climate station was used to characterize the climate conditions at the project site and the average year data are summarized in Table 5.1.

Table 5.1 – Average Annual Climatic Data (Schefferville Airport)

Month	Temperature (⁰C)	Rainfall (mm)	Snowfall (cm)	Precipitation (mm)	Snow Depth (cm)
January	-24.1	0.2	57.4	53.2	62.0
February	-22.6	0.2	42.6	38.7	70.0
March	-16.0	1.6	56.6	53.3	71.0
April	-7.3	8.4	54.8	61.4	69.0
May	1.2	27.7	22.9	52.1	18.0
June	8.5	65.4	8	73.7	0.0
July	12.4	106.8	0.5	107.2	0.0
August	11.2	82.8	1.7	84.5	0.0
September	5.4	85.3	12.7	98.4	0.0
October	-1.7	24.4	57.2	80.5	7.0
November	-9.8	4.5	70.7	69.4	26.0
December	-20.6	0.9	55.4	50.7	49.0

Average wind speed and direction is presented in Table 5.2. The average annual wind speed is about 16.5 km/h and the most frequent wind direction, on an annual basis, is from the northwest.

Table 5.2 – Schefferville Airport: Average Wind Speed/Direction (1971 to 2000)

Month	Speed (km/hr)	Most Frequent Direction	Maximum Hourly Speed (km/hr)	Maximum Gust Speed (km/hr)	Direction of Maximum Gust	Days with Winds ≥ 52 km/hr	Days with Winds ≥ 63 km/hr
January	16.4	NW	85	134	W	0.7	0.7
February	16.8	NW	97	148	W	1.4	0.5
March	17.4	NW	83	148	SW	1.9	0.4
April	16.5	NW	77	130	W	1.1	0.2
May	16	NW	66	101	W	0.9	0.1
June	16.2	NW	97	126	W	0.4	0.1
July	15.1	NW	65	103	W	0.6	0.2
August	15.6	NW	61	117	W	0.4	0.1
September	16.9	NW	80	137	SW	0.8	0.1
October	17.8	NW	89	137	SW	1.1	0.1
November	17.3	NW	84	142	SW	1.8	0.3
December	16	NW	80	153	SW	2.1	0.6

5.3 Local Resources and Infrastructures

Schefferville is only accessible by train from Sept-Îles. The Quebec North Shore & Labrador Railway (“QNS&L”) was established by IOC to haul iron ore from Schefferville area mines to Sept-Îles; a distance of 571 km starting in 1954. After shipping some 150 million tons of iron ore from the area, the mining operation was closed in 1982, and, QNS&L maintained a passenger and freight service between Sept-Îles, Labrador City and Schefferville till 2005. In 2005 IOC sold the 208 km section of the railway between Emeril Yard at Ross Bay Junction and Schefferville (the Menihek Division) to Tshiuetin Rail Transportation Inc. (“TSH”), a company owned by three Quebec First Nations. The mandate of TSH is to maintain the passenger and light freight traffic between Sept-Îles and Schefferville. Passenger and general freight train departures from Sept-Îles and Schefferville occur three times a week.

Six railway companies exist in the region;

- Tshiuetin Rail Transportation Inc. which runs passengers and freight service from Schefferville to Ross Bay Junction (208 km. south of Schefferville);
- QNS&L hauls freight/passengers and iron concentrates and iron pellets from the Labrador City/Wabush area via Ross Bay Junction to Sept-Îles;
- Bloom Lake Railway hauls iron ore from the Bloom Lake ("Cliffs") mine to Wabush (currently on care and maintenance);
- Arnault Railways hauls iron ore for Wabush Mines ("Wabush") and Cliffs mine between Arnault Junction and Pointe Noire (currently on care and maintenance);;
- New Millennium Limited built a railroad up to their Timmins mine, 20 km NW of Schefferville;
- ArcelorMittal's private railway hauls iron concentrates from the Fermont area to Port-Cartier, and.

The latter railway is not connected to TSH, QNS&L, Bloom Lake or Arnault railways.

At Schefferville there is an airport with a paved runway and navigational aids for passenger jet aircraft. Air service is provided three times per week, to and from Wabush, Labrador, with less frequent service to Montreal or Quebec City, via Sept-Îles.

5.4 Physiography

The Rainy Lake Property is located within a relatively rugged physiography with rolling hills and valleys reflecting the structure of the underlying bedrock. Elevation in the Project Area can vary from 440 m in the east of the projected open pit up to 840 m at the high point.

5.5 Local Resources

It is assumed that the majority of the workforce would come from the province of Québec and a part of the employees will be recruited from the communities close to the project site.

6 History

6.1 Prior Ownership

The iron potential of the Schefferville area has been undergoing re-evaluation since 2005 by several exploration companies, including New Millennium Capital Corp. ("New Millennium"), Tata Steel Global Minerals Holdings Pte. Ltd. ("Tata Steel"), Labrador Iron Mines Ltd. ("Labrador Iron Mines"), Century Iron Mines Corporation and its joint venture partners, and others.

The following paragraphs summarize the historical exploration work completed on the Rainy Lake property of the Sunny Lake project as compiled from public records of the MRNF. Past exploration activities discussed in this report focus on the Rainy Lake property area. Information on past exploration activities on the Lac Le Fer area of the property is available in the previous technical report for the Sunny Lake project (SRK, 2010).

6.2 Past Exploration at the Rainy Lake Property

In 1950, Hollinger mapped the Helluva Lake area (GM-6845). The author reported the presence of a metallic iron formation comprising from top to bottom of: a silicate-carbonate iron formation, a jasper-chert-hematite iron formation, a thin-bedded jasper-magnetite iron formation, and a silicatecarbonate-magnetite iron formation. The total thickness of the iron formation was reported to vary from 80 to 105 meters. No assay results were reported in the report.

That same year Hollinger also mapped the Eclipse Lake area (GM-6846), to the east-northeast of ancient biotite gneiss that is occupied by the Wishart, Ruth, and Sokoman formations. The base of the iron formation is well-developed unaltered band of cherty ferro-carbonate and cherty iron silicate rock. The remainder of the iron formation is chiefly the jasper variety and is exposed over broad areas and affected by folding and faulting. There is little iron enrichment in the iron formation. The report described the Partington showing as subparallel bands of manganese and iron with grey chert beds in an enriched zone located 1.2 kilometers southwest of Blade Lake. The average grades assayed from selected surface samples collected by Hollinger were 22.16 percent iron, 17.15 percent manganese, and 31.12 percent silica.

During 1972-73, the Iron Ore Company of Canada ("IOC") completed an airborne geophysical survey over three exploration permits (Permits 554, 555, and 556; GM-31122). The survey was targeting iron-rich taconite deposit type. Several airborne electromagnetic and magnetic anomalies warranting ground surface exploration follow-up were identified within Exploration Permit 555 covering the actual Rainy Lake property.

In 1974, the IOC reported ground exploration results for exploration permits 554, 555, and 556 (GM-31123). Surface work included: ground magnetic and gravimetric surveys; geological mapping and sampling on Exploration Permit 555 covering the northwest corner of the Rainy Lake property. The gravity survey served to supplement the magnetic data and to determine the cause of several strong airborne electromagnetic anomalies close to Rainy Lake that were associated with strong magnetic responses. Gravity data showed one broad anomaly, whereas magnetic data indicated several smaller anomalies for the same area. The magnetic anomalies mapped the near surface concentrations of magnetite, whereas the gravity data integrated the entire magnetite body at depth.

Gravity Anomaly #1 covers the whole grid with values ranging from 198.0 to 202.9 miligals: the lows being attributed to fault zones and the highs corresponding to high magnetic values. The high gravity and magnetic values are not consistent in trend. They show strong variations over short distances, which indicate an irregular distribution of hematite and magnetite within the iron formation. Thirty-nine samples were collected in this zone and Davis Tube Weight Recovery ("DTWR") returned an average 25.43 percent iron. The low and moderate magnetic readings within the iron formation indicate the absence of any high grade taconite body of potential economic importance.

Gravity Anomaly #2 covers the rest of the grid southeast of line 2767. The gravity and magnetic intensities and their magnitudes are considerably lower and more variable than Anomaly #1. Gravity values vary from 193.3 to 197.7 miligals. Thirteen samples collected in this area returned an average Davis Tube weight recovery of 22.59 percent iron, which confirms, along with the gravity and magnetic results that no taconite rich iron mineralization exists in this area.

7 Geological Setting and Mineralization

7.1 Regional Geology

The Rainy Lake property is located on the extreme western margin of the Labrador Trough adjacent to Archean basement gneisses as shown in Figure 7.1. The Labrador Trough, otherwise known as the Labrador-Québec Fold Belt, extends for more than 1,000 kilometers along the eastern margin of the Superior craton from Ungava Bay to Lake Pletipi in Québec. The thrust fold-belt is about 100 kilometers wide in its central part and narrows considerably to the north and south.

The Labrador Trough is a sequence of Proterozoic sedimentary rocks, including iron formation, volcanic rocks and mafic intrusions, which form the Kaniapiskau Supergroup (Figure 7.1). The Kaniapiskau Supergroup is comprised of the Knob Lake Group in the west and the primarily volcanic Doublet Group in the east. To the west of Schefferville, rocks of the Knob Lake Group lie unconformably on Archean gneisses.

The Kaniapiskau Supergroup has been intruded by numerous diabase dikes known as the Montagnais Intrusive Suite. These dikes along with the Nimish volcanic rocks are the only rock types representing igneous activity in the western part of the Labrador Trough (Williams and Schmidt, 2004).

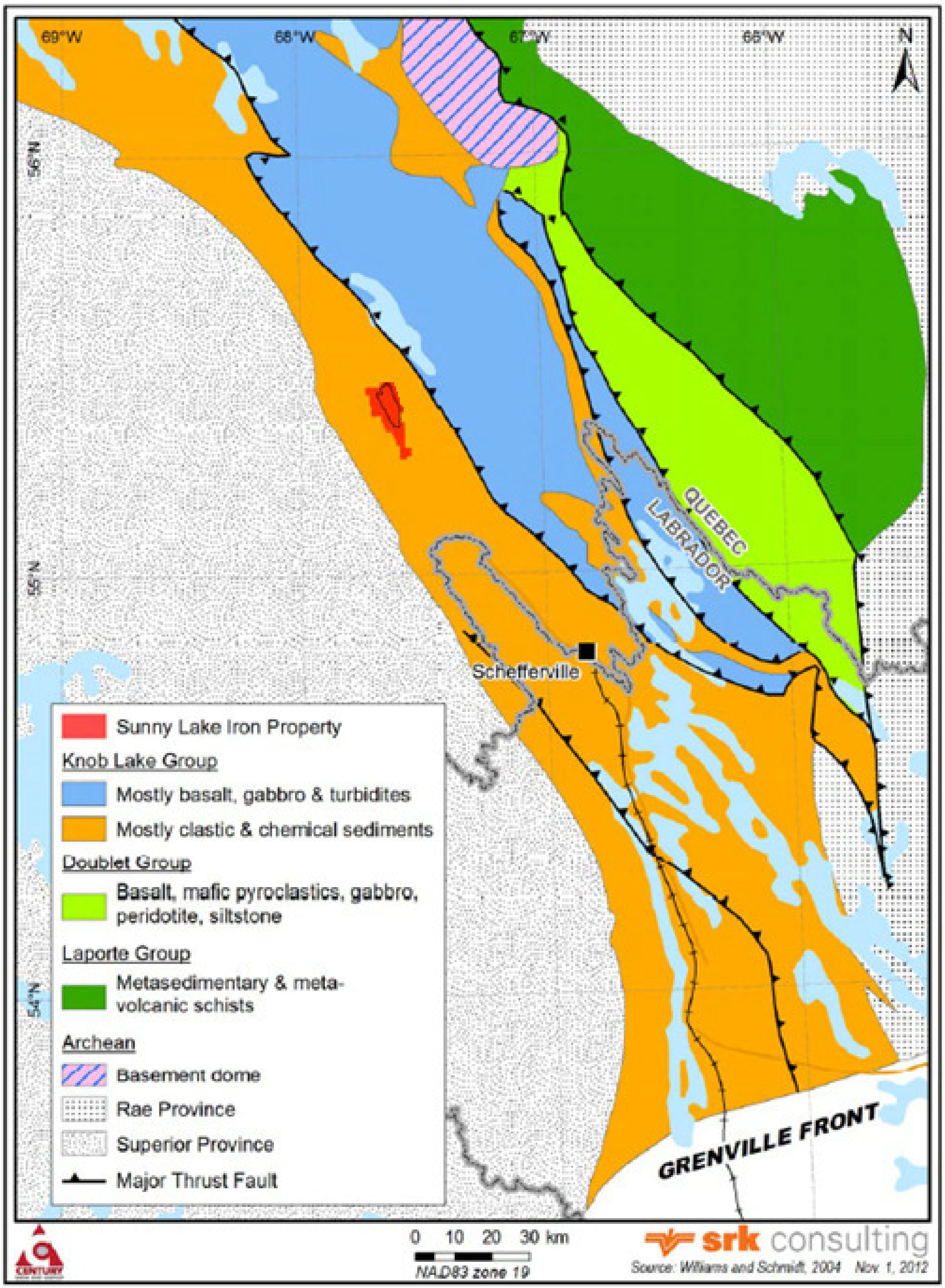
The Knob Lake Group includes the Sokoman Formation composed of iron formation is the main exploration target of the Rainy Lake property. The Sokoman Formation is a continuous stratigraphic unit that thickens and thins throughout the Labrador Trough.

The southern part of the Labrador Trough is truncated by the Grenville Front (Figure 7.1). Rocks of the Labrador Trough extend south of the Grenville Front boundary but here are strongly metamorphosed and complexly folded. Iron deposits in the Grenville part of the Labrador Trough include Lac Jeannine, Fire Lake, Mont-Wright, Mont-Reed, and the Luce Humphrey and Scully deposits in the Wabush area.

The high-grade metamorphism of the Grenville Province is responsible for the recrystallization of both iron oxides and silica in primary iron formation, producing coarse-grained sugary quartz, magnetite, and specular hematite schists.

Metamorphic grade increases from sub-greenschists assemblages in the west to upper amphibolite to granulite assemblages in the eastern part of the Labrador Trough (Dimroth and Dressler, 1978; Hoffman, 1988). Thrusting and metamorphism occurred between 1,840 and 1,829 million years during the Hudsonian Orogeny (Machado, 1990).

Figure 7.1 – Regional Geology Setting



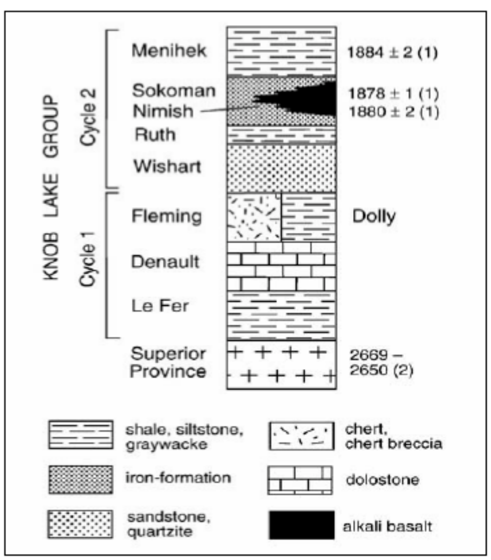
7.2 Property Geology

The Rainy Lake property area is underlain by Proterozoic sedimentary rocks that are subdivided into eight formal geological units included within the Knob Lake Group. The lowermost unit rests unconformably over Archean gneisses of the Ashuanipi Complex. From oldest to youngest, the rock units are the Seward, Lac Le Fer, Denault, Fleming, Dolly, Wishart, Sokoman, and Menihek formations (Figure 7.2 and Figure 7.3).

The sedimentary sequence of the Knob Lake Group consists of two sedimentary cycles (Wardle, 1982). Cycle 1 (the Attikamagen Subgroup) is a marine shelf (i.e., shallow water) succession comprising the Le Fer, Denault, Dolly, and Fleming formations. Cycle 2 represents deposition in a deeper water slope-rise environment. It begins with a transgressive quartz arenite (Wishart Formation) followed by shale and iron-formation of the Sokoman Formation and conformably overlain by the Menihek Formation (MSS). The Menihek Formation is composed almost entirely of grey to black, carbonaceous and locally pyritic shale, slate and siltstone, with minor feldspathic greywacke and chert. This formation is more than 300 meters thick, and is the most aerially extensive unit in the vicinity of the Schefferville project.

The Rainy Lake property is primarily underlain by the Dolly, Wishart, Sokoman and Menihek Formations. The Sokoman Formation is the most abundant geological unit underlying the property and represents the main exploration target.

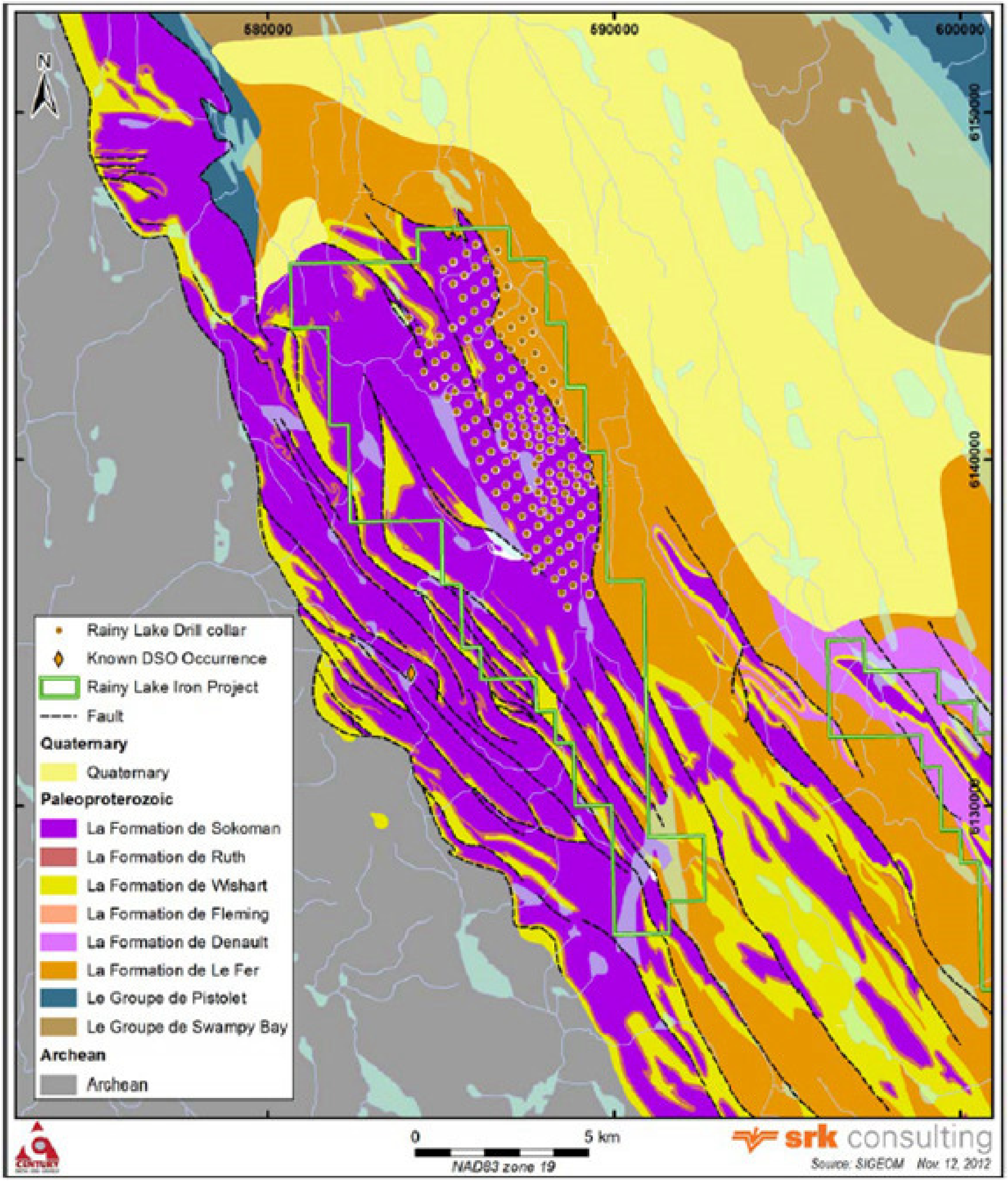
Figure 7.2 – Generalized Stratigraphy of the Knob Lake Group



(From Williams and Schmidt, 2004, with numbers representing ages of rock units in million years)

CIMA+ M03339A

Figure 7.3 – Geology of the Rainy Lake Property Area



7.2.1 The Sokoman Formation

The Sokoman Formation is the main host for iron-rich intervals in the Labrador Trough. An iron formation is a marine chemical sedimentary rock that contains more than 15 percent metallic iron. Paleomagnetic findings indicate that the 1.88 billion year old iron formations of the Sokoman Formation were deposited at approximately 30 degrees latitude south (Williams and Schmidt, 2004).

The thickness of the Sokoman Formation varies between 120 and 240 meters and is a typical Lake Superior-type iron-formation (taconite), consisting of banded sedimentary rock composed principally of layers of iron oxide, magnetite and hematite. Iron-rich bands are intercalated with cherty bands composed of variable amounts of silicate, carbonate, sulphide, ferruginous slaty iron formation, and carbonaceous shale. The Sokoman Formation is subdivided into three regionally extensive stratigraphic members:

- The Upper Iron Formation ("UIF") member (25 to 60 meters thick) consists of green, greenish grey, and pink-grey magnetite-chert iron formation with local zones of laminated to bedded siderite-magnetite-chert iron formation. It comprises the following subunits:
    - Lean Chert;
    - Jasper Upper Iron Formation; and
    - Green Chert.

It conformably overlies the:

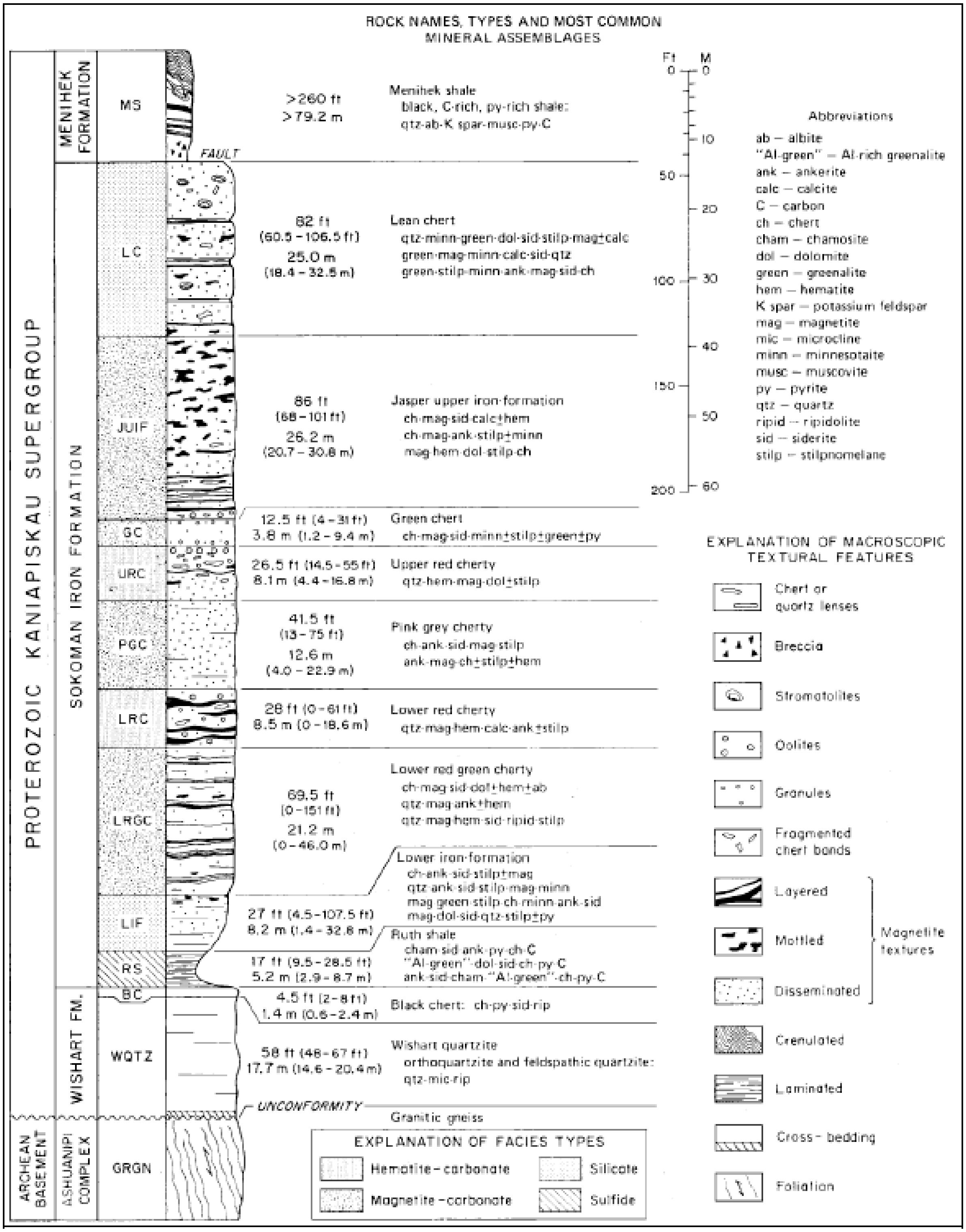
- The Middle Iron Formation ("MIF") member (15 to 60 meters thick; averaging 30 meters) consists mainly of arenaceous and argillaceous oxide-facies iron formation containing 30 percent to 70 percent iron oxides with magnetite-chert iron formation more abundant near the bottom, and jasper-magnetite-chert iron formation more abundant towards the top. This member commonly forms topographic highs. It comprises of the following subunits:
    - Upper Red Chert;
    - Pink Grey Chert; and
    - Lower Red Chert.

It conformably overlies the:

- The Lower Iron Formation member (1 to 35 meters thick) includes thin-bedded to laminated chert-siderite with thin interbeds of shale (the Ruth shale; formerly the Ruth Formation) overlain by pink, reddish-grey, and green-grey layered silicate-magnetite-carbonate (siderite) and cherty magnetite-hematite iron-formation. It comprises the following subunits:
  - Lower Red Green Cherty; and
  - Lower Iron Formation.

The stratigraphic division in Figure 7.4 is based on the interpretation of Klein and Fink (1976) on the Howells LabMag Taconite Deposit of New Millennium southwest of the Sunny Lake Project. Field representations of the URC, PGC, LRGC and LIF are shown in Figure 7.5.

Figure 7.4 – Detailed Stratigraphic Column of the Sokoman Formation



(From Klein and Fink, 1976)

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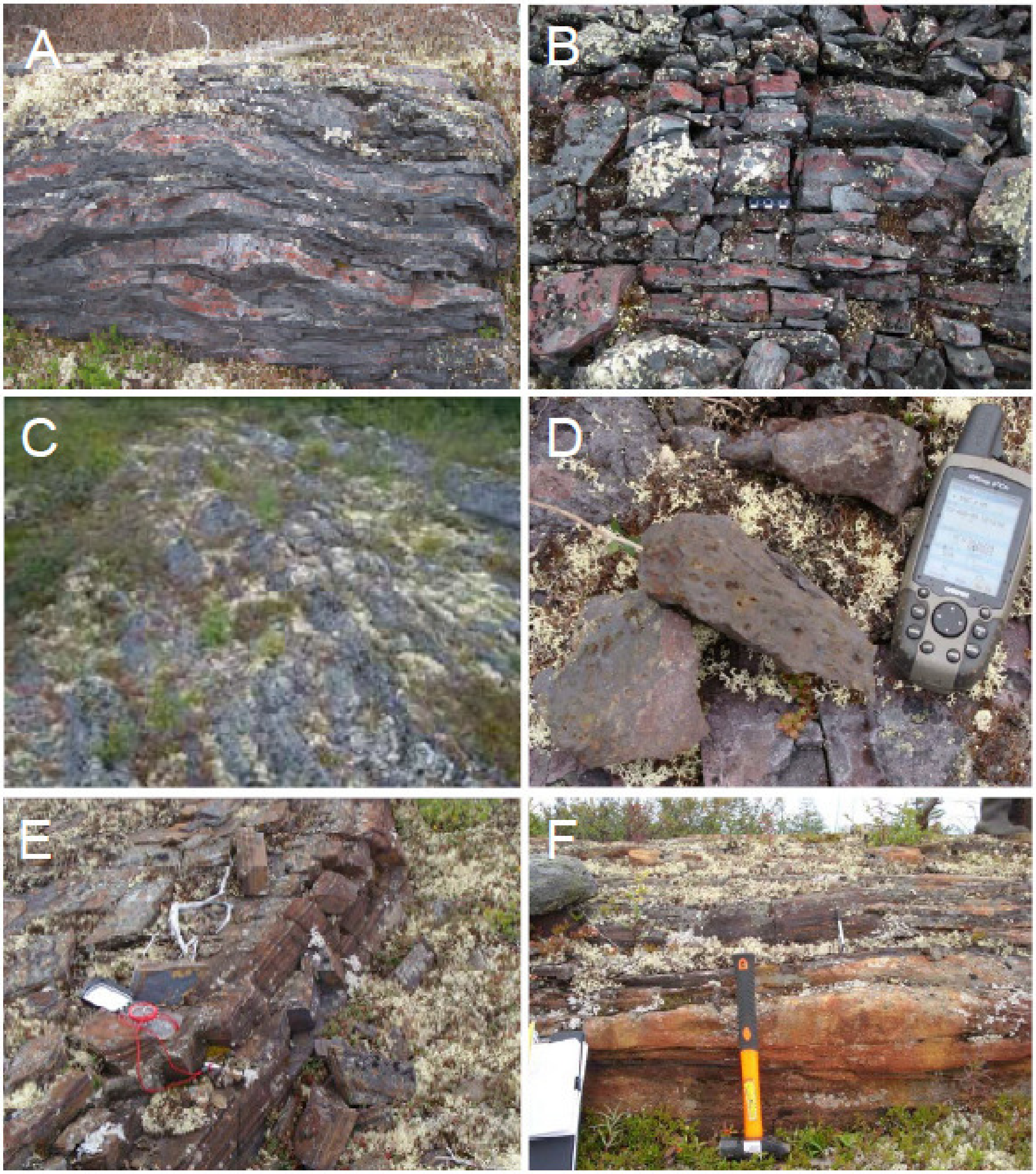
Figure 7.5 – Sokoman Formation in Outcrop in the Schefferville Area



A: URC, Rainy Lake area.

B: URC, Hayot Lake area.

C: Gently dipping PGC, Hayot Lake East area.

D: Leached PGC, Lac Sans Chef North area.

E: Gentle dipping LRGC/LIF, hinge zone of anticline, Hayot Lake area.

F: Gentle dipping LRGC/LIF, hinge zone of anticline, Hayot Lake area.

Magnetite is generally abundant in the LRC and especially in the PGC. The PGC contains several meter-scale intervals of strongly magnetic, black laminated magnetite interbedded with thin, moderately magnetic, cherty beds over approximately 20-meter thick layers. In the LRC, magnetite occurs in 5- to 20-centimeters thick strongly magnetic laminated magnetite beds intercalated with weakly magnetic red magnetite-bearing chert over a thickness of approximately 15 meters. Hematite is abundant in the URC and lower JUIF. Magnetite is present in the URC, but is more variable than in the LRC and PGC.

Historical mining activities in the Schefferville area focused on carbonate and silicate-leached rocks rich in hematite, limonite, and goethite traditionally termed Direct Shipping Ore ("DSO") iron deposits. These iron deposits resulted from intense tropical weathering of the iron formations during the Cretaceous. The DSO term is only used for historical accuracy and is not intended to imply that a positive economic study has been completed.

7.2.2 Local Geology of the Rainy Lake Property

The description of the Rainy Lake property geology is based on drilling completed by WCSLIM in 2011 and 2012. The lithological units logged and modelled are described below and shown on Figure 7.6.

Menihek Shale (MSS)

The MSS is non-magnetic fine grained black to grey and green finely laminated shale beds. It contains trace amounts of pyrite and approximately 5 percent graphite. It is generally very broken up in drill core. It contains stockworks of both quartz and carbonate veinlets, likely of hydrothermal origin. The MSS sometimes contains quartz carbonate veins up to 60 centimeters in thickness. It is the upper most unit observed in the project area.

Lean Chert (LC)

The LC is light grey to greenish grey fine- to medium-grained laminated and non-laminated iron formation. Generally laminations are observed in the green chert units. The average logged thickness is 15 meters. The LC unit is sometimes weakly magnetic but generally is non-magnetic. It is commonly very low in iron oxides, dominated with iron carbonates and iron silicates.

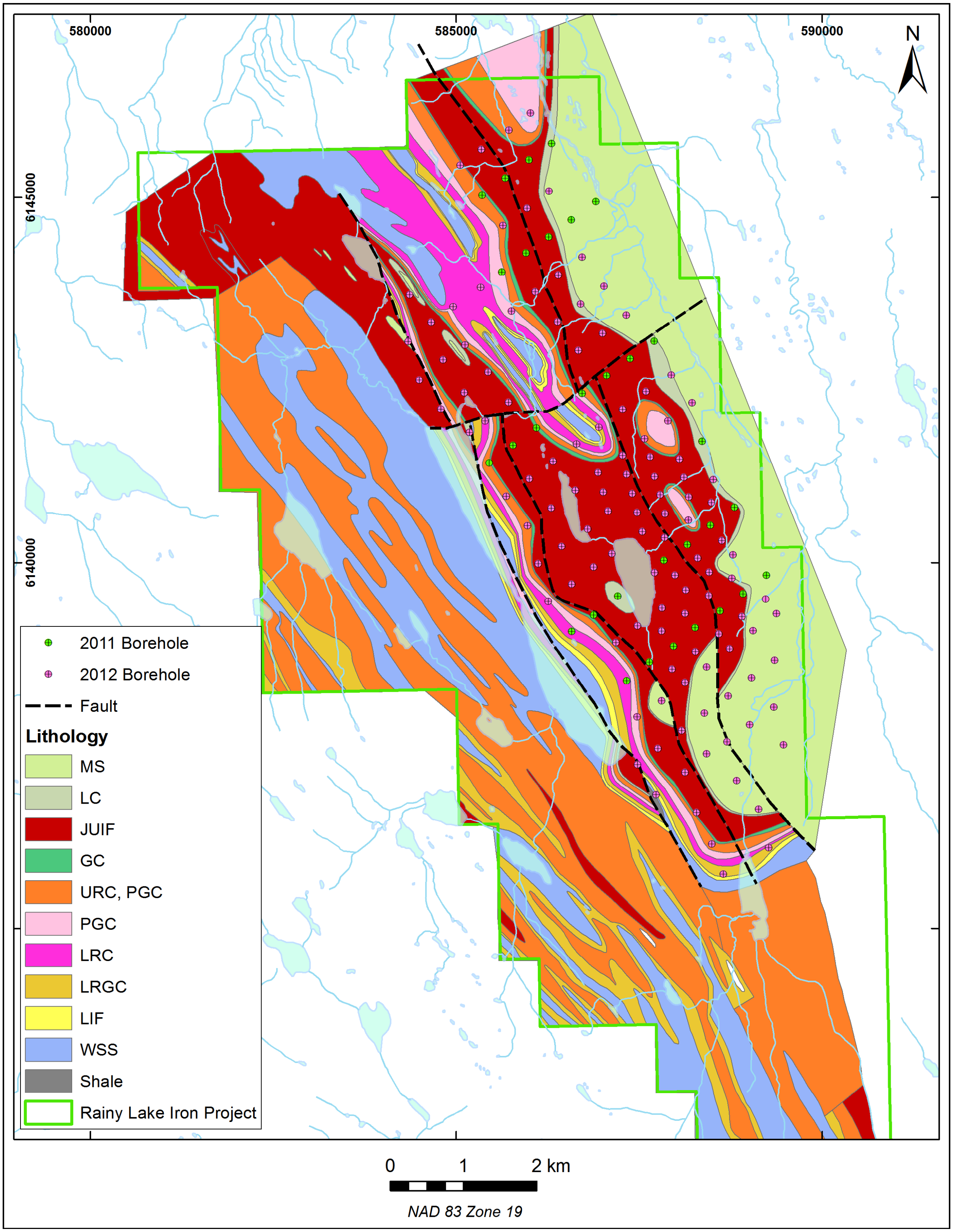
Jasper Upper Iron Formation (JUIF)

The JUIF unit is defined by grey, strongly magnetic, fine grained magnetite with blotches of red jasper chert. The chert is generally red but occasionally displays a purplish colour. The jasper is irregular in shape and in core appears banded. The unit is locally folded and banded. The average logged thickness is 70 meters, though this is slightly amplified by thrust faulting. The fine grained grey portion of the unit is made up of both magnetite and hematite and some white chert. Small carbonate pods are often observed in thick magnetite bands. Small red chert oolites are also characteristic of the unit.

Green Chert (GC)

The GC unit is typically light green to medium green with a very low to nonexistent magnetic signature. It consists of fine grained green and white chert. It is a very thin unit with an average thickness of 6 meters. The LC contains laminations of white and green chert. Occasionally contains disseminated pyrite and shale beds interlayered within the green and white chert. This unit is used as a marker horizon for determining the location in the stratigraphy.

Figure 7.6 – Surface Geology of the Rainy Lake Peoperty Area Mapped by WCSLIM



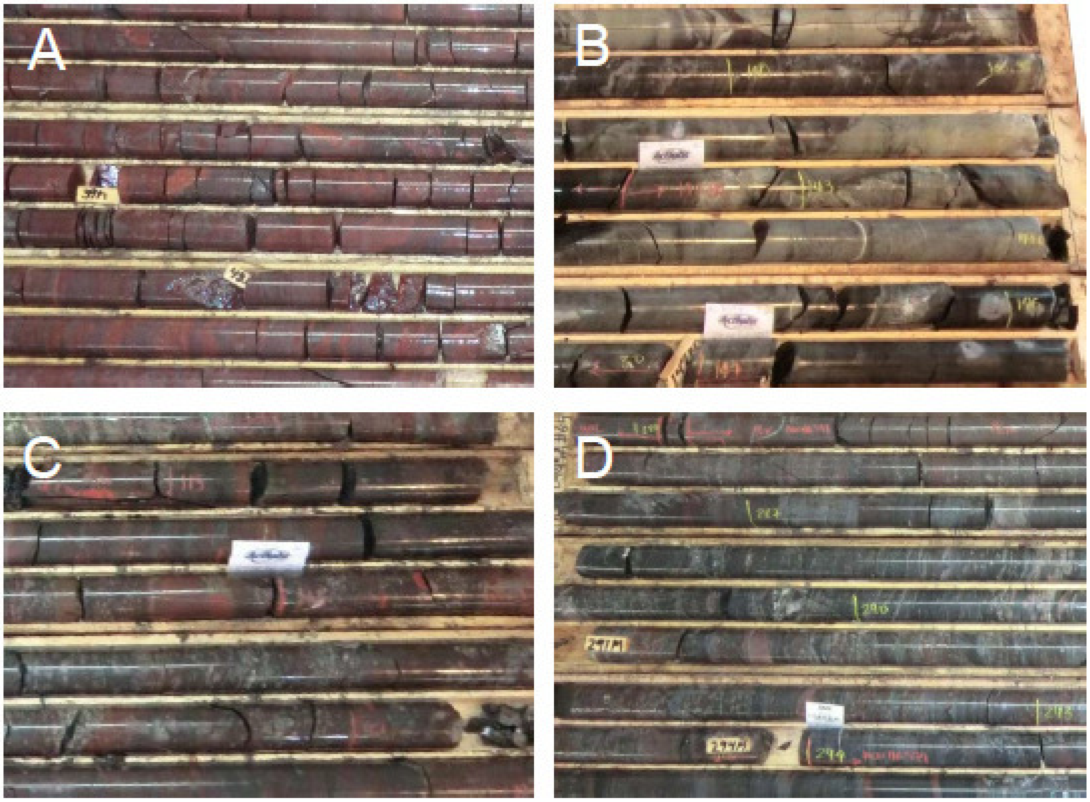
Upper Red Chert (URC)

The URC unit is defined by grey magnetic, fine grained magnetite with patches of red chert. The red chert is irregular in shape and in core appears banded. The chert is generally red but occasionally displays a purplish colour. The blotches of red chert are made up of oolites. The unit is locally folded and banded. The fine grained grey portion of the unit is made up of both magnetite and hematite and some white chert. The URC is characterized by the presence of up to 20 centimeters thick rich hematite-jasper layers within a massive hematite rich-magnetite poor cherty matrix. The upper contact of the URC is often characterized by a 30 centimeter red-beige interval. The average logged thickness is 17 meters.

Pink Grey Chert (PGC)

The PGC is defined by light pink/grey chert bands in a highly magnetic fine grained medium-grey matrix. The unit is dominated by magnetite that is present both as banded and disseminated magnetite. Typically the grey/pink clasts have a sandy texture. The upper boundary is usually identified by the appearance of millimeter to centimeter width pink chert bands and nodules. The pink chert bands can fade away towards the middle of the unit resulting in a competent, massive, and highly magnetic medium grey rock. The average logged thickness is 28 meters.

Figure 7.7 – Sokoman Formation in Core from the Rainy Lake Property



A: JUIF, Borehole RL-12-0405, 33 to 45 meters.

B: GC, Borehole RL-12-0005, 138 to 147 meters.

C: URC, Borehole RL-12-0405, 115 to 122 meters.

D: PGC, Borehole RL-12-0405A, 284 to 296 meters.

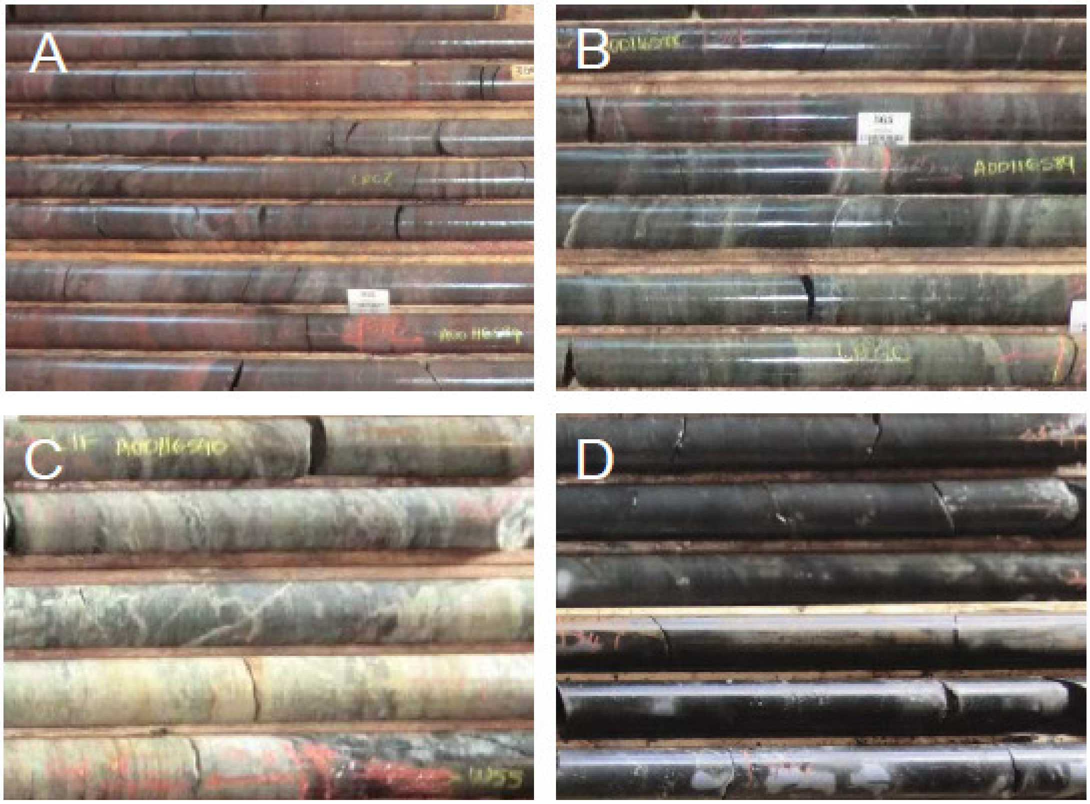
Lower Red Chert (LRC)

The LRC is identified by its dark red to dark purple colour, fine grained texture, and highly magnetic signature. Magnetite is present as a pervasive fine grained mineral throughout the unit. Some small jasper clasts are rarely observed. The average logged thickness is 25 meters.

Lower Red Green Chert (LRGC)

The LRGC is defined as dark red to dark green, fine grained, and laminated. The unit is observed as small thin bands of intercalated red and green chert in a dark grey matrix. Magnetite is present as blotchy patches of thin veinlets or as disseminated magnetite throughout the unit. The upper contact with the LRC is commonly marked by the dissipation of jasper clasts. The average logged thickness is 17 meters.

Figure 7.8 – Sokoman Formation and Wishart Formation in Core from the Rainy Lake Property



A: LRC, Borehole RL-12-0405A, 305 to 318 meters.

B: LRGC, Borehole RL-12-0405A, 323 to 330 meters.

C: LIF, Borehole RL-12-0405A, 330 to 335 meters.

D: WSS, Borehole RL-12-0405A, 336 to 345 meters.

Lower Iron Formation (LIF)

The LIF is a thin unit of light beige to medium green colour with a very low to non-magnetic signature. The upper boundary is marked by the sudden drop of magnetism as well as the sudden change in colour relative to the LRGC. Minor quartz veins are sometimes present in this unit along with some beige carbonates. The lower contact is sometimes marked by some shearing and faulting, as well as the sudden colour change of the dark shale.

Wishart Formation and Undifferentiated Shale (WSS)

The Wishart Formation is light green coloured quartzite. Underlying the Wishart Formation is the dark non-magnetic shale (Undifferentiated Shale, US) with uncertain stratigraphic relationship. For geological modelling, the WSS and US were modelled as one unit. The upper WSS boundary signifies the bottom of the Sokoman Formation at the Rainy Lake property.

7.2.3 Structural Geology

The Labrador Trough is a linear thrust fold-belt marking the junction of the Superior and Churchill structural provinces in northern Québec. The Paleoproterozoic Trans-Hudsonian orogeny is responsible for the majority of structures observed the Rainy Lake property area. The Sunny Lake project is dominated by northwest-southeast striking thrust faults interpreted based on borehole information and surface mapping. The thrust faults exhibit shallow 15 to 30 degrees easterly dip directions and are interpreted to connect to a decollement surface at the Archaean basement and sedimentary package contact. The major thrust faults in the Sunny Lake area also exhibit splay faults that on occasion are observed on surface, but generally dissipate before reaching the upper most units. The thrust faulting has contributed to the unusual thickness of the Sokoman formation on the Rainy Lake property.

Three thrusts faults were interpreted in the southeast portion of the project area and two thrust faults were interpreted in the northwest (Figure 7.6). These thrust faults are offset by a sub-vertical cross fault in the central portion of the deposit between sections 14 and 16. The cross fault was also evidenced by the sharp change in surface geology across the fault. The fold structures in the Rainy Lake area exhibit a coaxial fold interference pattern. The two folding events that produced this structure are unrelated. One set of folding, F1, is related to thrusting during the Hudsonian orogeny that exhibits northwest-southeast striking axial planes, while the other fold system, F2, is caused by a second smaller, possibly local folding event with northeast-southwest striking axial planes. F1 folds are tight and doubly plunging in the northwest and southeast directions. F2 folds are gentle and appear to be flat lying.

7.3 Mineralization

The Sokoman Formation occurring on the Rainy Lake property consists mostly of recrystallized chert and jasper with bands and disseminations of magnetite, hematite and martite, a pseudomorph of hematite after magnetite and specularite. Other observed iron-silicate minerals include minnesotaite, pyrolusite, stilpnomelane, and iron carbonate, mainly siderite. In most of the sampling programs done on the Sunny Lake project, the highest consistent concentration of magnetite occurs within the PGC unit of the Sokoman Formation. The JUIF also contains locally higher concentrations of magnetite, while hematite is most common in the LRC, URC, and JUIF submembers. Magnetite also occurs within the LIF and LRGC units, but their total iron content includes the presence of variable amounts of iron silicate and carbonate. Siderite is also common in the LRC and LIF submembers, where manganese carbonates are also present. Calcite fills some fractures, while goethite and limonite are also common as fracture coatings, and are likely due to groundwater percolating.

8 Deposit Types

The Full Moon iron deposit located within the Rainy Lake property occurs in the Sokoman Formation. This is a taconite or Lake Superior-type iron deposit. These deposits consist of a banded sedimentary unit composed principally of bands of magnetite and hematite within chert-rich rock, and variable amounts of silicate-carbonate-sulphide lithofacies. Such iron formations have been the principal sources of iron throughout the world (Gross, 1996). The salient characteristics of Lake Superior iron deposits are summarized in Table 8.1 (Eckstrand, 1984).

The minimum iron content required for a taconite deposit to be considered as economic at a given market price is generally greater than 30 percent (or approximately 40 percent iron oxide).

Lake Superior-type iron formations with low iron content locally can be brought to ore-grade through the process of enrichment (enriched ore) by leaching and deep weathering processes (DSO-type). This process involves the migration of meteoric and syn-orogenic heated fluids. DSO-type mineralization generally has an iron grade in excess of 50 percent (or approximately 70 percent iron oxide). In the case of the Labrador Trough, the Hudsonian orogenesis provided such fluids.

Hydrothermal and meteoritic fluids circulating through the banded iron formation recrystallized iron rich minerals to hematite, and leached silica and carbonate. The process may involve more than one stage (e.g., hypogene replacement of chert by carbonate, followed by supergene leaching of the carbonate, and the oxidation of magnetite to hematite). The result is an enriched iron formation that may be further enriched, whereby iron oxides (goethite, limonite), hematite, and manganese are redistributed into the openings left by the primary leaching phase, and/or deposited along fracture/cleavage surfaces and in veinlets.

Almost all the near-surface iron deposits in the Labrador Trough are enriched to some degree by these processes.

Deeper lithofacies that are not highly metamorphosed or altered by weathering are referred to as taconite. The iron deposits located in the vicinity of Schefferville are residual deposits formed by the enrichment of what was originally taconite.

As per the mining process, iron oxides of a given iron ore deposit must also be amenable to concentration (beneficiation) and the concentrates produced must be low in manganese, aluminum, phosphorus, sulphur, and alkalis.

Beneficiation involves segregating the silicate and carbonate minerals, and other rock types, interbedded within the iron formation from the iron-rich oxides. Beneficiation of taconite has resulted in the successful economic production of many contemporary iron ore deposits.

Table 8.1 – Summary Characteristics of the Lake Superior-type Iron Deposit Model (From Eckstrand, 1984)

DEPOSIT MODEL FOR LAKE SUPERIOR-TYPE IRON FORMATION
AFTER ECKSTRAND (1984)

Commodities	Fe (Mn)
Examples: Canadian - Foreign	Knob Lake, Wabush Lake and Mount Wright areas, Que. and Lab. - Mesabi Range, Minnesota; Marquette Range, Michigan; Minas Gerais area, Brazil.
Importance	Canada: the major source of iron. World: the major source of iron.
Typical Grade, Tonnage	Up to billions of tonnes, at grades ranging from 15 to 45% Fe, averaging 30% Fe.
Geological Setting	Continental shelves and slopes possibly contemporaneous with offshore volcanic ridges. Principal development in middle Precambrian shelf sequences marginal to Archean cratons.
Host Rocks or Mineralized Rocks	Iron formations consist mainly of iron- and silica-rich beds; common varieties are taconite, itabirite, banded hematite quartzite, and jaspilite; composed of oxide, silicate and carbonate facies and may also include sulphide facies. Commonly intercalated with other shelf sediments: black
Associated Rocks	Bedded chert and chert breccia, dolomite, stromatolitic dolomite and chert, black shale, argillite, siltstone, quartzite, conglomerate, red beds, tuff, lava, volcaniclastic rocks; metamorphic equivalents.
Form of Deposit, Distribution of Ore Minerals	Mineable deposits are sedimentary beds with cumulative thickness typically from 30 to 150 m and strike length of several kilometres. In many deposits, repetition of beds caused by isoclinal folding or thrust faulting has produced widths that are economically mineable. Ore mineral distribution is largely determined by primary sedimentary deposition. Granular and oolitic textures common.
Minerals: Principal Ore Minerals - *Associated* Minerals	Magnetite, hematite, goethite, pyrolusite, manganite, hollandite. - Finely laminated chert, quartz, Fe-silicates, Fe-carbonates and Fe-sulphides; primary or metamorphic derivatives
Age, Host Rocks	Precambrian, predominantly early Proterozoic (2.4 to 1.9 Ga).
Age, Ore	Syngenetic, same age as host rocks. In Canada, major deformation during Hudsonian, and in places, Grenvillian orogenies produced mineable thicknesses of iron formation.
Genetic Model	A preferred model invokes chemical, colloidal and possibly biochemical precipitates of iron and silica in euxinic to oxidizing environments, derived from hydrothermal effusive sources related to fracture systems and offshore volcanic activity. Deposition may be distal from effusive centres and hot spring activity. Other models derive silica and iron from deeply weathered land masses, or by leaching from euxinic sediments. Sedimentary reworking of beds is common. The greater development of Lake Superior-type iron formation in early Proterozoic time has been considered by some to be related to increased atmospheric oxygen content, resulting from biological evolution.
Ore Controls, Guides to Exploration	1. Distribution of iron formation is reasonably well known from aeromagnetic surveys. 2. Oxide facies is the most important, economically, of the iron formation facies. 3. Thick primary sections of iron formation are desirable. 4. Repetition of favourable beds by folding or faulting may be an essential factor in generating widths that are mineable (30 to 150 m). . 5. Metamorphism increases grain size, improves metallurgical recovery. 6. Metamorphic mineral assemblages reflect the mineralogy of primary sedimentary facies. 7. Basin analysis and sedimentation modelling indicate controls for facies development, and help define location and distribution of different iron formation facies.
Author	G.A. Gross

9 Exploration

Exploration activities discussed in this report focus on the Rainy Lake property part of the Sunny Lake Project. Further exploration details on other areas of the property are available in the previous National Instrument 43-101 technical report for the property prepared by SRK in 2010.

9.1 Geological Reconnaissance - 2009

The short 2009 exploration program carried out by WCSLIM was to confirm the presence of the Sokoman Formation as shown on the Québec provincial geological maps and to crudely assess the iron potential of the magnetic anomalies as identified from public domain SIGEOM geophysical data. The crew was based in Schefferville and access to the property was by way of fly-in fly-out on daily basis using a Bell 206 helicopter belonging to Expedition Helicopters from Cochrane, Ontario.

Forty-nine field outcrop samples, (21 from Rainy Lake and 28 from Lac Le Fer) were collected during this program. Twelve samples were submitted for mineralogical studies and preliminary metallurgical testing. Each sample consists of a composite rock chip sample (2 to 3 kilograms in weight) collected from the same outcrop using a rock pick over an area measuring approximately 5 by 5 meters.

All samples were shipped from Schefferville to the ALS Minerals Laboratory (ALS Minerals) in Val-d'Or, Québec for preparation and to its North Vancouver laboratory for assaying. Results are presented in Table 9.1.

The reconnaissance work and sampling program was successful in confirming that the Sokoman Formation is the source of the main magnetic anomalies underlying both properties. Most of the samples collected during the program were collected on relatively high topographic features associated with the PGC member of the Sokoman Formation.

The mineralogical studies of 12 samples (6 from each project) show that the sum of all valuable iron minerals (hematite + magnetite + iron oxide) varies from 29 to 75 percent for all samples from the Sokoman unit identified on the properties (PGC, URC and LRGC). The samples collected in 2009 show a strong variability in their mineralogical assemblage from hematite-rich magnetite-poor to magnetite-rich hematite-poor. Sample 178081 in the Lac Le Fer area described as from the URC contains 75 percent hematite.

Because of the limited duration of the 2009 reconnaissance program, the structural setting of the properties was not specifically evaluated. At Rainy Lake, the Sokoman Formation was measured as exposed over an area measuring 15 by 10 kilometers with an approximate thickness of 115 meters. Exposures of the iron formation over such large surface areas imply repetition by faulting and folding. It strikes fairly consistently towards the northwest with dips varying from flat to steep to the northeast or southwest.

Table 9.1 – Summary of 2009 WCSLIM Reconnaissance Surface Sampling Results on the Rainy Lake Property

Sample ID	Summary Description of Sample	Unit	Fe (%)
178055	Brown surface (carbonate), green-grey chert; 5-10 percent disseminated Mt. Medium grained.	MSS	2.6
178050	Dark bluish-black. Very fine grained. Not magnetic.	MSS	7.6
177997	Bluish grey Mt + pinkish chert. Disseminated Mt 30-70 percent. Massive, medium grained.	PGC	18.4
178000	Grey-brown yellow chert with carbonate; Mt 30-50 percent. Moderately to strongly magnetic.	PGC	22.8
178045	Grey-pinkish; glittering; 15 percent mottled + micro-brecciated red chert. Strongly magnetic.	PGC	23.3
178079	Qz-flooded; located next to MSS unit injected by Qz veining. Variably magnetic.	PGC	24.9
177998	Anticline hinge zone; bluish Mt + red-pink chert; Mt 30-70 percent.	PGC	26.6
177999	Pink-grey chert; 20-40 percent Mt as fine beds 0.2-0.5 millimeter.	PGC	27.5
178056	Blue-pink chert with disseminated to massive Mt 50 percent. Some white Qz-Carb bands with Mt.	PGC	28.6
178048	Identical to sample #178046; less than 10 percent silica and red jasper micro breccias-style material. Strongly magnetic.	PGC	29.0
178046	Idem to previous; <10 percent silica-red chert material as micro-breccias. Strongly magnetic	PGC	30.0
178057	Blue Mt bands. Massive Mt + Hm 40-70 percent; heavy. Strongly magnetic.	PGC	30.4
178054	Blue-pinkish grey Mt bands + chert; Mt 50-80 percent. Anticline hinge one.	PGC	31.5
178049	Similar to sample #178048.	PGC	31.7
178078	All grey (no mixing); not pitted. Strongly magnetic.	PGC	31.8
178077	All grey; glitters; grey-pinkish medium grained rock; local pitted surface; high grade Fe.	PGC	32.2
178058	Blue-pinkish chert with densely disseminated Mt 40-70 percent. Moderate to strongly magnetic.	PGC	33.2
178047	Medium grey, glitters, fine grained; heavy rock. Strongly magnetic.	PGC	35.9
178051	Grey-brown yellow chert with carb + 50 percent Mt. Locally as 10 centimeters beds. Moderately magnetic.	PGC	36.0
178052	D. grey to bluish Mt bands + few red chert bands. Mt 40-70 percent. f. to med gr.	PGC	36.4
178053	Rusty surface. Blue grey. Fine to medium grained chert with more than 30 percent Mt. Locally Mt beds 1 centimeter thick. Strongly magnetic.	PGC	38.4

9.2 Mineralogical Study

COREM laboratory in Québec City conducted mineralogical characterization work on samples from the Sunny Lake project reconnaissance sampling program of 2009.

The objective of that work was to characterize the mineralogy of the samples and provide preliminary mineralogy frequency analysis of their content, with a focus on identifying and quantifying iron-bearing and gangue minerals, and evaluating their relative size distribution. Results are summarized in Table 9.2.

Optical and electronic microscopy, X-ray diffraction, and X-ray fluorescence were used on samples ground to approximately 90 percent passing -4 mesh screen (4.75 millimeters).

Table 9.2 – Mineral Distribution of Samples Examined by COREM from the Lac Le Fer and Rainy Lake Properties

Sample ID	Area	Unit(1)	Hematite	Magnetite(2)	Iron Hydroxides	Sum of Valuable Mineral	Quartz	Carb	Other Fe/Mg silicates	Other(4)
178000	RL	PGC	22	17	1	40	57	2	1	n.d.
177998	RL	PGC	39	19	n.d.*	58	37	1	2.1	1
177999	RL	PGC	8	33	n.d.	41	51	2	6	1
177995	RL	PGC	11	33	1	45	51	1	2.2	1
178077	RL	PGC	n.d.	43	1	45	46	0	8	1
178053	RL	PGC	n.d.	34	5	39	21	2	38	1
178087	LLF	PGC	24	12	n.d.	36	56	1	6	n.d.
178083	LLF	PGC	19	20	n.d.	38	49	1	11	n.d.
178088	LLF	PGC	36	12	1	48	43	1	7	n.d.
178082	LLF	PGC	24	26	n.d.	50	34	1	16	n.d.
178081	LLF	URC	75	1	n.d.	75	24	0	n.d.	n.d.
178093	LLF	LRG	n.d.	25	5	29	54	3	13	n.d.

(1) PGC = Pink Grey Chert; URC = Upper Red Chert; LRGC = Lower Red Green Red Chert.

(2) Based on the Satmagan measurement.

(3) Mainly clay minerals group (talc. minnesotaite) and Amphibole group (riebeckite. etc).

(4) Pyrite, chlorite, phosphates, etc n.d. = not detected, or under 1 percent detection limit RL = Rainy Lake; LLF = Lac Le Fer.

CIMA+ M03339A

The main valuable iron-bearing minerals identified in the samples are granular and specular hematite, granular magnetite, and, in minor amount, iron hydroxides (only goethite identified). Some particles, called martitic magnetite, were observed and correspond to magnetite particles partially pseudomorphed into martite.

The gangue minerals are quartz, often observed with a microcrystalline texture and in some cases as a red opaque massive form (jasper) due to the presence of ultrafine inclusions of iron oxides. The other minerals are: riebeckite (an amphibole), minnesotaite (a clay mineral), dolomite-ankerite-siderite (containing or not calcium, iron, magnesium, and manganese), and traces of calcite and plagioclase feldspar.

Traces of accessory minerals such as pyrite, phosphate, titanium, and manganese-bearing minerals were detected but their abundance is generally lower than the detection limit (less than 1 percent by weight).

In the WCSLIM samples analyzed, the grade of total iron dosed by chemical analysis varies between 33.1 and 75.6 percent iron oxide. After completing the mineral quantification, the grades of valuable iron varied between 30 and 75 percent iron oxide (Table 9.3 and Table 9.4).

Table 9.5 presents an estimation of the magnetite and hematite mean size distribution that should be achieved to obtain optimal iron oxide liberation. The numbers are based on the observations taken from the polished sections. The main characteristic is that the valuable iron is fine-grained. Hematite and magnetite are generally smaller than 150 microns and are intimately associated with gangue minerals, mainly quartz. Sample #178088 contains specular hematite that is fine, generally less than 25 microns.

Table 9.3 – Chemical Analysis of the Head Samples Examined by COREM

Sample	Area	Unit(1)	Sat. (%)	Fe_2O_3 (%)	SiO_2 (%)	Al_2O_3 (%)	MgO (%)	CaO (%)	Na_2O (%)	K_2O (%)	MnO (%)	LOI* (%)	CO_2 (%)
178000	RL	PGC	16.9	41.4	57.3	0.09	0.28	0.57	0.10	0.02	0.23	0.34	0.7
177998	RL	PGC	19.2	59.8	38.8	0.15	0.56	0.15	0.12	0.07	0.27	0.08	0.6
177999	RL	PGC	33.2	44.1	54.6	0.13	1.23	0.29	0.1	0.04	0.09	-0.33	0.8
177995	RL	PGC	32.7	46.5	52.9	0.10	0.61	0.12	0.11	0.02	0.69	-0.28	0.4
178077	RL	PGC	42.7	49.1	50	0.08	1.38	0.11	0.11	0.03	0.09	-0.77	0.4
178053	RL	PGC	33.7	55.6	40.6	0.18	2.79	0.03	0.12	0.11	0.35	0.95	0.7
178087	LLF	PGC	12.2	39.6	59.3	0.09	0.53	0.18	0.53	0.01	0.13	0.04	0.4
178083	LLF	PGC	19.6	43.5	55.0	0.09	0.66	0.07	1.08	0.02	0.07	-0.24	0.4
178088	LLF	PGC	11.7	51.7	47.3	0.08	0.83	0.06	0.54	0.02	0.01	0.17	0.5
178082	LLF	PGC	26.1	55.4	43.1	0.08	2.00	0.04	0.71	0.02	0.04	-0.19	0.4
178081	LLF	URC	0.60	75.6	24.1	0.11	0.06	0.03	0.10	0.03	0.53	0.30	0.2
178093	LLF	LRGC	24.7	33.1	63.4	0.14	2.69	0.19	0.10	0.04	0.13	1.00	1.1

(1) PGC = Pink Grey Chert; URC = Upper Red Chert; LRGC = Lower Red Green Chert.

* LOI = Loss on ignition.

RL = Rainy Lake; LLF = Lac Le Fer.

Table 9.4 – Proportions and Grades of Iron Occurring as Valuable Iron-Bearing Minerals in Samples Examined by COREM

Sample	Area	Unit(1)	Total Iron Fe_2O_3 (%)	Total Valuable Iron (%)	Valuable Iron Fe_2O_3 (%)
178000	RL	PGC	41.4	99	40.9
177998	RL	PGC	59.8	98	58.9
177999	RL	PGC	44.1	96	42.4
177995	RL	PGC	46.5	98	45.8
178077	RL	PGC	49.1	93	45.8
178053	RL	PGC	55.6	71	39.6
178087	LLF	PGC	39.6	94	37.3
178083	LLF	PGC	43.5	90	39.3
178088	LLF	PGC	51.7	95	49.0
178082	LLF	PGC	55.4	92	51.2
178081	LLF	URC	75.6	100	75.3
178093	LLF	LRGC	33.1	91	30.0

(1) PGC = Pink Grey Chert; URC = Upper Red Chert; LRGC = Lower Red Green Chert.

RL = Rainy Lake; LLF = Lac Le Fer.

Table 9.5 – Estimation of Mean Grain Size of Magnetite and Hematite in Samples Examined by COREM

Sample	Area	Unit(1)	Magnetite (2) (micons)	Granular Hematite (microns)
178000	RL	PGC	75-150	<50
177998	RL	PGC	75-150	<50
177999	RL	PGC	75-150	50-100
177995	RL	PGC	50-75	<50
178077	RL	PGC	50-100	n.d.*
178053	RL	PGC	75-150	n.d.
178087	LLF	PGC	50-75	50-100
178083	LLF	PGC	50-75	50-100
178088	LLF	PGC	50-75	25
178082	LLF	PGC	50-75	25-75
178081	LLF	URC	n.d.	<75
178093	LLF	LRGC	75-150	n.d.

(1) PGC = Pink Grey Chert; URC = Upper Red Chert; LRGC = Lower Red Green Chert.

(2) Correspond to granular or martitic magnetite.

* n.d. = not determined. RL = Rainy Lake; LLF = Lac Le Fer.

9.3 Geophysics

9.3.1 Public Domain

The Rainy Lake and Lac de Fer properties overlap NTS map sheets 23O/03, 23O/05 and 23O/06. Within the SIGEOM database, the following airborne geophysical surveys are of particular interest:

- Québec MRNF surveys: Series of airborne MAG and EM airborne surveys carried out between 1968 and 1992, at a line spacing of 200 meters and an average flight altitude of 120 meters. The surveys encompass the entire three maps;
- Detailed surveys registered at the MRNF office. The geophysical databases covered by NTS maps 23O and 23J are part of:
  - DP-96-13 which includes the results of four distinct surveys (DP-85-20, 86-02, 86-21 and 87-04). DP-85-20 and DP-86-02 more specifically overlap locally the Lac Le Fer property;
  - DP-85-20: Airborne magnetic and time-domain electromagnetic survey carried out by Questor in 1984. The line spacing was 200 meters and the average flight altitude was 120 meters. The survey overlaps the north-western part of Lac Le Fer property; and

- DP-86-02: Helicopter magnetic and three-frequency electromagnetic survey carried out by Aerodat/Les Relevés Géophysiques in 1983 with a line spacing of 200 meters and a light altitude of 30 meters. The survey area overlaps the south-eastern part of Lac Le Fer property.

9.3.2 NOVATEM Survey - 2010

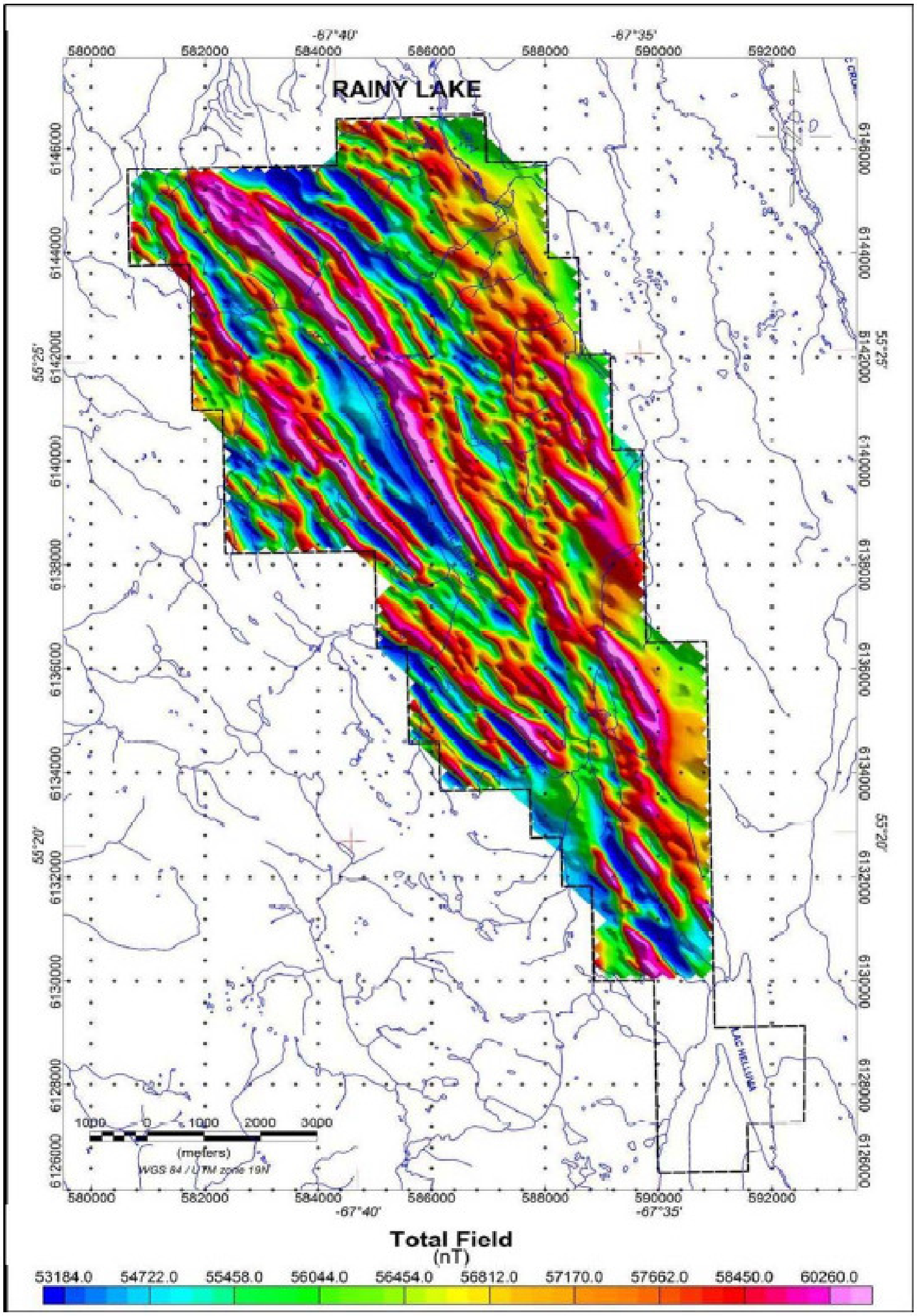
In January 2010, WCSLIM contracted NOVATEM Inc. ("Novatem") based in Mont-Saint-Hilaire, Québec to conduct a helicopter magnetic survey over the northwest part of the Lac Le Fer and most of the Rainy Lake properties (Table 9.6). The southern limit of the Lac Le Fer survey is the northern limit of the 1983 Aerodat survey. The data was reviewed by independent contractor Mr. Joel Simard of St-Donat, Québec.

Table 9.6 – Novatem Magnetic Survey Specifications

Property	Parameters	Specifications
Lac Le fer	January 21 to 24, 2010	528.3 line kilometers
Rainy Lake	January 18 to 20, 2010	536.6 line kilometers
Rainy Lake and Lac Le Fer	Survey line spacing 200 meters	Survey line spacing 200 meters
	Tie line spacing 1,000 meters	Tie line spacing 1,000 meters
	Survey line direction N045 degrees	Survey line direction N045 degrees
	Tie line direction N135 degrees	Tie line direction N135 degrees
	Sampling rate 10 Hz:3 m at 110 km/h	Sampling rate 10 Hz:3 m at 110 km/h
	Mean sensor terrain clearance 28 meters	Mean sensor terrain clearance 28 meters

At the Rainy Lake property, the helicopter-borne survey was specifically designed to improve the resolution and definition of the anomalies that were delineated on the MNRF maps in order to enhance the definition of exploration targets (Figure 9.1).

Figure 9.1 – Total Magnetic Field Map of the Rainy Lake Property (SRK, 2010)



The results of the airborne survey are in fact of better quality, which is most likely due to the lower survey height; 30 meters versus 120 meters as well as certain technological improvements. The analysis of the magnetic maps reveals the following:

- Anomalies are much less homogenous and continuous than what is seen on older survey maps;
- The amplitude of the magnetic anomalies varies from approximately 500 nanotesla ("nT") to a maximum of 10,000 nT, with relatively short wavelengths, indicative of shallow to outcropping sources;
- The lateral trend of the magnetic anomalies is most often of the order of 1 kilometer. These anomalies are most often elliptical in shape and their strike more or less northwest-southeast;
- Anomalies are indicative of formations rich in ferromagnesian minerals, such as banded iron formations. Their magnetic response is mostly due to their magnetite content; and
- The apparent thickness of these formations varies from a few tens of meters to almost 200 meters.

Airborne magnetic data are useful in assisting the mapping of the shape and geometry of magnetic bedrock sources, and in aiding exploration targeting. Magnetic inversion, a data processing technique that maps the magnetic susceptibility of an area in three dimensions, is a tool that is useful to reconcile the length, widths, depth, and dip of magnetic bedrock sources, allowing to model the major geological features, predict their three-dimensional shapes, and infer potential volumes of magnetic rock.

SRK believes that magnetic inversion is a powerful tool that can assist identification and prioritization of exploration targets for taconite type iron mineralization, and hence improve the definition of higher quality drilling targets.

WCSLIM commissioned Mira Geosciences ("Mira") from Vancouver, British Columbia to interpret the airborne magnetic data available for the Rainy Lake and Lac Le Fer properties. The scope of work included processing the magnetic data and the construction of magnetic inversion models to define the three-dimensional geometry of the magnetic bedrock sources on the properties, and infer the shape, geometry, and volume of potential taconite style iron mineralization that would produce the measured magnetic response.

The source data for the Mira work was the 2010 Novatem survey data acquired by WCSLIM and supplemented by older public domain data sourced from the MRNF databases. The magnetic inversion and modelling work focussed on three areas: two on Lac Le Fer and one on Rainy Lake. The susceptibility modelling process involves initial geological modelling; removal of regional signal from data sets; unconstrained magnetic inversion on a block model with 25 meters cubic cells; extraction into Gocad for geometrical analysis; and volume analysis using susceptibility cut-off values.

The volumetric estimates for potential taconite type iron formation derived by Mira were used to assist in the selection of exploration drilling targets and aid future exploration on the Rainy Lake and Lac Le Fer properties. The reader is cautioned that this semi-automated potential field modelling technique was conducted prior to drilling on the property, in absence of hard constraining data about the subsurface geology and geometry of the magnetic sources. Any volumetric estimates derived from this potential field mapping study were conceptual in nature and do not represent an attempt to define a mineral resource.

Mira considered that a magnetic susceptibility threshold of 0.22SI was useful to define the boundary of magnetic sources potentially related to taconite iron mineralization. This threshold was defined by visual inspection, experience, and proprietary empirical formulae derived from analysis of data from other iron oxide prospects. In Gocad, an iso surface created with a susceptibility value above 0.22SI was used to infer the volumes of potential taconite iron mineralization.

The potential volumes of taconite iron mineralization estimated by Mira are reported in Table 9.7 as a range to account for uncertainties of the models especially at depth where the unconstrained inversion model is less focused.

Table 9.7 – Global Volume and Potential Tonnage Estimates* Derived from Unconstrained Magnetic Inversion Study, Mira Geoscience, October 28, 2010

Area	Volume (m^3)	Specific Gravity Estimate**	Potential Tonnage (100 % of iso surface)	Potential Tonnage (50 % of iso surface)
Lac de fer Area 1	1,900,000,000	3.6	6,840,000,000	3,420,000,000
Lac de fer Area 2	1,740,000,000	3.6	6,264,000,000	3,132,000,000
Rainy Lake	2,520,000,000	3.6	9,072,000,000	4,536,000,000
Total			**22,176,000,000**	**11,088,000,000**

* The reader is cautioned that the estimates presented in this table should not be misconstrued with a mineral resource. At the time of this exercise, there had been insufficient exploration on the Lac Le Fer and Rainy Lake properties to define a mineral resource, and it was uncertain if further exploration would results in the discovery of a mineral resource.

** Average assumed specific gravity.

9.3.3 Ground Gravity Survey - 2010

During 2010, WCSLIM commissioned independent contractor Mr. Joel Simard of St-Donat, Québec to plan, monitor, and interpret a ground gravity survey over small areas of the Sunny Lake project. The survey was carried out by Geosig Inc. of Québec City, Québec.

The gravity method was chosen in order to discriminate between hematite and magnetite mineralization based on their density contrast. Survey profiles were laid out over ten distinct areas of interest in the Sunny Lake project including two areas in the Rainy Lake property and eight areas at the Lac Le Fer project. Gravity measurements were taken every 50 meters along northeast-trending profiles ranging between 350 and 2,050 meters in length. A real-time high resolution GPS system was used to locate the survey lines.

The positioning of the gravity stations and surveying of the gravity lines was carried out between August 16 and September 10, 2010 and produced 485 gravity stations. The quality of the gravity data was insured by repeating 3 percent of the 485 gravity stations surveyed.

The definitive coordinates of the two reference were calculated by using an on line service developed by the Canadian Government called precise point positioning. The final database contains location data in UTM coordinates (Nad 83 datum, Zone 19 north).

The gravity survey was carried out using a CG-5 micro gravimeter made by Scintrex Limited. The gravimeter has a digital screen with a large memory capacity and allows the user to obtain a field repeatability of approximately 5.0 microGals. A very low instrumental drift was observed and, in particular for the survey, an average daily drift of 0.020 milliGal was noted.

9.3.4 Ground Gravity Survey – 2011

In 2011, WCSLIM commissioned JVX Ltd. ("JVX") of Richmond Hill, Ontario, to carry out a ground gravity survey over areas of the Rainy Lake property. Ground gravity was chosen in an attempt to discriminate between hematite and magnetite bearing mineralisation based on their density contrast.

Survey profiles were laid out over nine lines within the Rainy Lake property claims and one outside the project boundary (Figure 9.2). Gravity measurements were taken every 25 meters along northeast trending profiles ranging between 300 and 3,300 meters in length. A real-time high resolution GPS system was used to locate the survey lines.

The positioning of the gravity stations and surveying of the gravity lines was carried out between March 30 and April 15, 2011 and produced 549 gravity stations. The quality of the gravity data was insured by repeating 10 percent of the 549 gravity stations surveyed. The average repeatability for the gravity stations was 0.014 microGals.

The gravity survey was also carried out using a CG-5 micro gravimeter made by Scintrex Limited. The gravimeter has a digital screen with a large memory capacity and allows the user to obtain a field repeatability of approximately 5.0 microGals. Instrumental drift correction was applied.

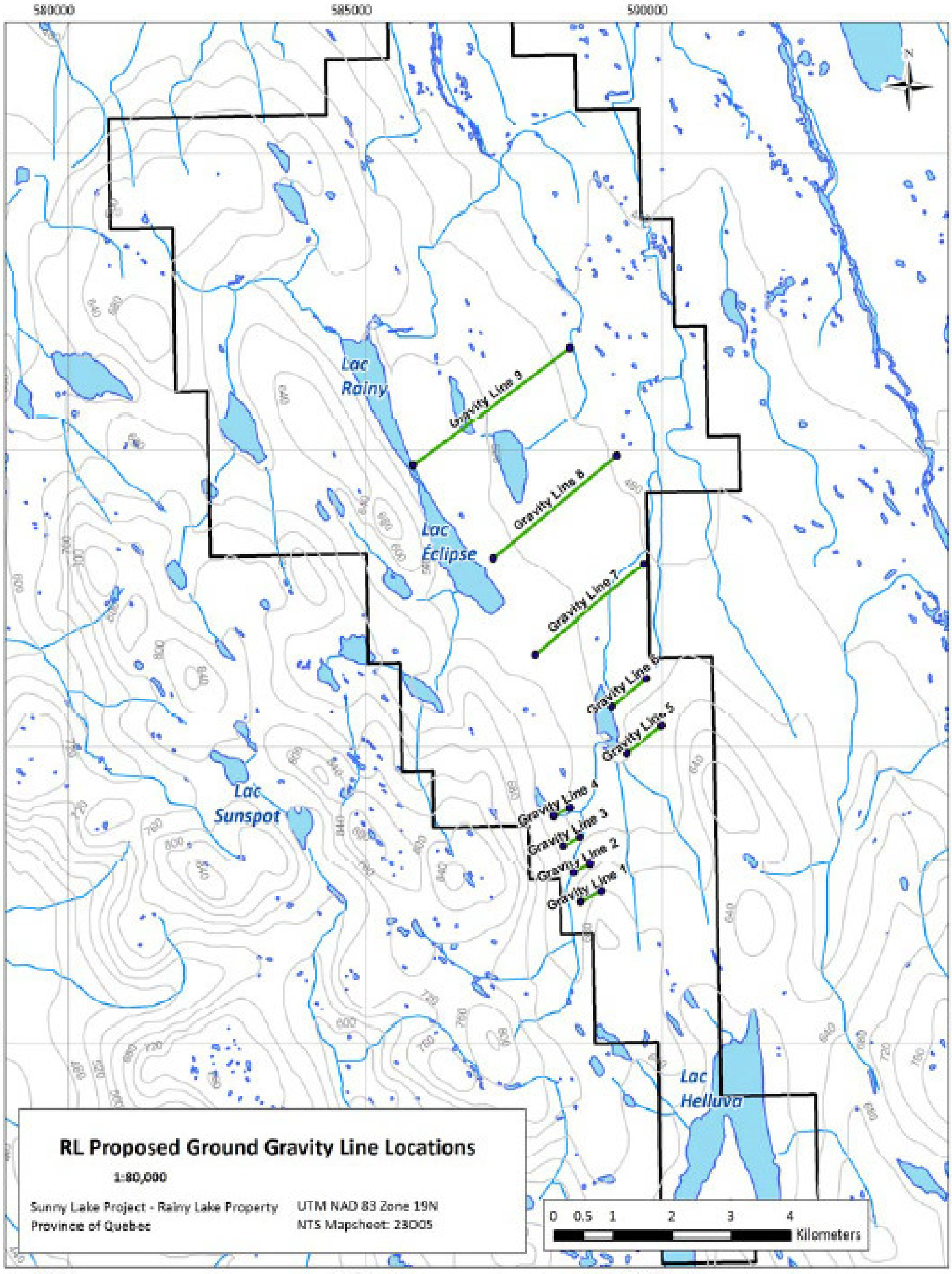
9.3.5 LiDAR Survey - 2012

In mid-2012, WCSLIM contracted XEOS Imagerie ("XEOS") of Québec City, Québec to fly a fixedwing light detection and ranging ("LiDAR") airborne remote sensing survey over the Rainy Lake property to map the topography. The primary airborne remote sensing equipment attached to a Piper Navajo aircraft was an Optech ALTM Gemini No. 07sen209 LiDAR system. A GPS precise point positioning base station was utilized for calibration.

The survey, flown August 11-12 and September 1-2, 2012, covered both the Rainy Lake property area of the Sunny Lake Project as well as the Joyce Lake project of the Attikamagen property, also operated by Century.

The survey was flown at two separate point densities. The low-density flight, flown at 1,250 meters from ground elevation, retains a point density of 1 point per meter square. The high-density flights, flown at 825 meters from ground elevation, have a point density of 4 points per meter square. According to XEOS, the precision attained in the high density flights is 15 centimeters along the easting and northing and 10 centimeters along the elevation. All data received from XEOS was in UTM coordinates NAD83 datum, Zone 19 north, in 1 by 1 kilometer tiles.

Figure 9.2 – Location of Ground Gravity Survey Grids in 2011 (JVX, 2011)



10 Drilling

This report documents the core drilling undertaken by WCSLIM on the Rainy Lake property.

10.1 Drilling 2011

In 2011, WCSLIM drilled 31 core boreholes (6,387 meters) on the Rainy Lake property. The taconite target was interpreted as a shallow dipping magnetite-rich iron formation with an expected minimum thickness of over 100 meters. The objective of the drilling program was to evaluate the taconite potential by collecting basic information such as thickness and grade continuity of the Sokoman Formation subunits along five section lines spaced by 1,000 to 2,500 meters with borehole spacing of 400 meters along each line. Access to the property for this program was by helicopter from a field camp located on the northwestern corner of Lac Rainy. Drilling was completed by Forage G4 of Val-d'Or, Québec. The drilling program started in August 2011 and was completed on by the end of November 2011. Drill rigs were moved between drilling sites using a helicopter. The drilling program at the Rainy Lake property in 2011 consisted of 31 NQ-size vertical core boreholes (Table 10.1) in an area measuring 6.5 by 2.8 kilometers (Figure 10.1).

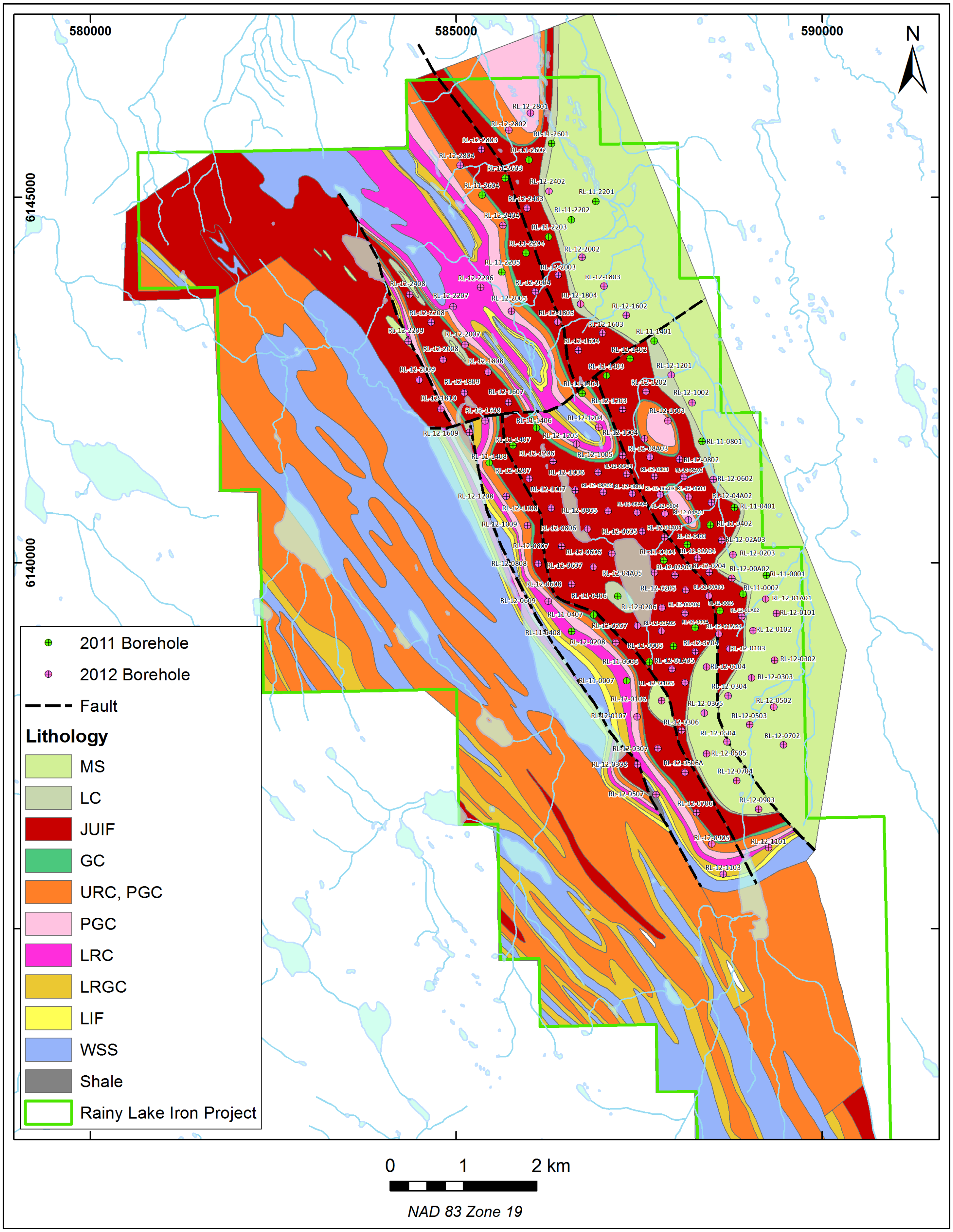
Table 10.1 – Characteristics of Core Boreholes Drilled at Rainy Lake in 2011

Borehole ID	Easting * (meter)	Northing* (meter)	Elevation (meter)	Azimuth (degree)	Plunge (degree)	Lenth (meter)	Sample (count)
RL-11-0001	589,238	6,139,830	478.15	0	-90	345.0	87
RL-11-0002	588,920	6,139,573	489.29	0	-90	384.0	118
RL-11-0003	588,604	6,139,342	495.48	0	-90	284.0	77
RL-11-0004	588,265	6,139,112	507.38	0	-90	300.0	93
RL-11-0005	587,971	6,138,860	515.09	0	-90	267.0	81
RL-11-0006	587,640	6,138,642	515.33	0	-90	198.0	57
RL-11-0007	587,328	6,138,387	537.85	0	-90	210.0	57
RL-11-0401	588,801	6,140,758	479.37	0	-90	330.0	109
RL-11-0402	588,472	6,140,516	483.34	0	-90	271.8	88
RL-11-0403	588,159	6,140,248	500.60	0	-90	177.0	45
RL-11-0404	587,837	6,140,034	505.86	0	-90	300.0	84
RL-11-0406	587,209	6,139,546	512.45	0	-90	206.8	61
RL-11-0407	586,876	6,139,284	529.73	0	-90	96.0	26
RL-11-0408	586,580	6,139,061	546.05	0	-90	66.0	11
RL-11-0801	588,359	6,141,664	465.77	0	-90	321.0	82
RL-11-1401	587,707	6,143,035	458.37	0	-90	165.0	49
RL-11-1402	587,375	6,142,794	477.07	0	-90	243.0	72
RL-11-1403	587,055	6,142,564	491.07	0	-90	175.5	50
RL-11-1404	586,718	6,142,317	517.88	0	-90	90.0	24
RL-11-1406	586,094	6,141,845	511.45	0	-90	78.0	20
RL-11-1407	585,778	6,141,605	531.27	0	-90	185.0	51
RL-11-1408	585,450	6,141,368	546.43	0	-90	99.0	28
RL-11-2201	586,906	6,144,943	431.10	0	-90	279.5	48
RL-11-2202	586,573	6,144,696	446.87	0	-90	249.0	58
RL-11-2203	586,264	6,144,463	466.55	0	-90	153.0	42
RL-11-2204	585,952	6,144,240	486.63	0	-90	126.0	35
RL-11-2205	585,623	6,143,981	517.50	0	-90	103.1	27
RL-11-2601	586,304	6,145,738	432.24	0	-90	258.0	76
RL-11-2602	585,992	6,145,515	452.42	0	-90	165.0	48
RL-11-2603	585,668	6,145,266	465.66	0	-90	123.0	36
RL-11-2604	585,351	6,145,032	503.57	0	-90	138.0	46
Total 31 boreholes						**6,386.70**	**1,786**

* UTM Coordinates (Nad83, Zone 19)

CIMA+ M03339A

Figure 10.1 – Location of Core Boreholes Drilled at Rainy Lake between 2011 and 2012



CIMA+ M03339A

Borehole locations were planned and marked by WCSLIM geologists using a handheld GPS device. All boreholes were drilled vertically. The collar location was surveyed with a differential GPS unit by Allnorth Consultant Limited after completion of drilling. Downhole surveys were completed for all boreholes using a Reflex Instruments Flexit device at the end of the borehole in non-magnetic rock. The deviation of the vertical boreholes is not expected to be material. Core retrieved from boreholes was moved from the drilling sites to the core shack in Schefferville by helicopter. Core was examined for consistency, its distance markings were verified, and recovery and rock quality designation were measured by a trained technician. Magnetic susceptibility was measured using a multi-parameter probe ("MPP") magnetic susceptibility meter. Core was logged by qualified geologists. All core was photographed prior to sampling. Descriptive data was recorded electronically in Microsoft Excel spreadsheets.

A total of 1,786 samples (5,367 meters) were collected from half core split lengthwise with a mechanical splitter. Sampling intervals were determined by a geologist according to the uniformity of the iron mineralization and the geological boundaries. Sample lengths vary from 1.0 to a maximum of 5.0 meters with 70 percent of samples collected at 3.0 meter intervals. The remaining half core was replaced in the core box and archived. Samples were organized into batches and sent by freight from Schefferville to Activation Laboratories Ltd. ("Actlabs") in Ancaster, Ontario for preparation and testing.

10.2 Drilling - 2012

In 2012, 117 core boreholes (24,555 meters) were completed on the Rainy Lake property. Drilling took place between March and October and was contracted to Forage G4 of Val d'Or, Québec. One hundred thirteen holes were drilled using NQ coring equipment and 4 holes were drilled using HQ coring equipment for metallurgical sampling. Access to the property for this program was by helicopter from a field camp on the northwestern corner of Lac Rainy. Drill rigs were moved between drilling sites using a helicopter.

The purpose of the 2012 drilling program was to expand the drilling to an area of approximately 10.5 by 3.5 kilometers. The boreholes are distributed on section lines spaced at 500 meters and borehole spacing on each section line of 400 meters. In the central part of the deposit, in-fill drilling was conducted at line spacing of 250 meters. The physical characteristics of the boreholes are presented in Table 10.2. The distribution of the boreholes drilled in 2012, and 2011, is shown in Figure 10.1.

Borehole locations were planned and marked by WCSLIM geologists using a handheld GPS. The collar location was surveyed with a differential GPS unit by Allnorth Consultant Limited after completion of drilling. All boreholes were drilled vertically. Downhole surveys were completed for all boreholes using a Reflex Instruments Flexit device at the end of the borehole in non-magnetic rock. The deviation of the vertical boreholes is not expected to be material. Core retrieved from boreholes was moved from the drilling sites to the core shack in Schefferville by helicopter. Core was examined for consistency, its distance markings were verified, and recovery and rock quality designation was measured by a trained technician. Magnetic susceptibility was measured using a multi-parameter probe magnetic susceptibility meter. All core was photographed, prior to sampling. Descriptive data was recorded electronically in Microsoft Excel spreadsheets.

A total of 4,372 samples (19,563 meters) were collected from half core split lengthwise with a mechanical splitter. Sampling intervals were determined by a geologist according to the uniformity of the iron mineralization and the geological boundaries. Sample lengths vary from 0.6 to a maximum of 10 meters. Remaining half core was replaced in the core box and archived. Samples were organized into batches and sent by freight from Schefferville to Actlabs in Ancaster, Ontario or SGS Canada Inc. ("SGS") in Lakefield, Ontario for preparation and testing.

Table 10.2 – Summary Characteristics of Core Boreholes Drilled at Rainy in 2012

Borehole ID	Easting * (meter)	Northing* (meter)	Elevation (meter)	Azimuth (degree)	Plunge (degree)	Lenth (meter)	Sample (count)	
							Received	Pending
RL-12-00A02	588,769	6,139,787	488.53	0	-90	315.0	0	59
RL-12-00A03	588,451	6,139,548	498.46	0	-90	249.3	0	40
RL-12-00A04	588,129	6,139,306	507.60	0	-90	234.0	0	43
RL-12-00A05	587,809	6,139,066	510.89	0	-90	267.0	0	53
RL-12-0101	589,376	6,139,307	477.24	0	-90	307.0	37	0
RL-12-0102	589,057	6,139,070	493.51	0	-90	374.0	0	71
RL-12-0103	588,737	6,138,824	499.47	0	-90	336.0	70	0
RL-12-0104	588,422	6,138,576	510.39	0	-90	225.0	36	0
RL-12-0105	588,127	6,138,360	523.59	0	-90	282.0	56	2
RL-12-0106	587,806	6,138,111	525.63	0	-90	285.0	48	0
RL-12-0107	587,470	6,137,895	542.61	0	-90	204.0	37	0
RL-12-01A01	589,229	6,139,505	478.14	0	-90	360.0	0	60
RL-12-01A02	588,907	6,139,265	489.70	0	-90	330.0	0	66
RL-12-01A03	588,589	6,139,027	498.92	0	-90	258.4	0	40
RL-12-01A05	587,947	6,138,546	518.24	0	-90	288.0	0	57
RL-12-0203	588,781	6,140,106	484.51	0	-90	363.0	0	62
RL-12-0204	588,460	6,139,869	493.43	0	-90	232.0	0	40
RL-12-0205	588,139	6,139,626	504.67	0	-90	231.1	0	47
RL-12-0206	587,814	6,139,380	508.75	0	-90	270.0	0	56
RL-12-0207	587,477	6,139,140	508.96	0	-90	222.0	0	41
RL-12-0208	587,182	6,138,907	529.17	0	-90	93.7	0	16
RL-12-02A03	588,628	6,140,307	482.91	0	-90	312.0	0	65
RL-12-02A04	588,300	6,140,062	497.67	0	-90	231.0	0	43
RL-12-02A05	587,990	6,139,829	505.94	0	-90	369.0	0	73
RL-12-0302	589,353	6,138,668	489.25	0	-90	358.5	51	4
RL-12-0303	589,034	6,138,426	500.00	0	-90	372.0	56	0
RL-12-0304	588,717	6,138,177	506.07	0	-90	405.0	62	0
RL-12-0305	588,393	6,137,944	523.85	0	-90	264.0	40	0
RL-12-0306	588,081	6,137,708	536.07	0	-90	225.0	42	0
RL-12-0307	587,759	6,137,455	540.35	0	-90	111.0	17	0
RL-12-0308	587,475	6,137,238	539.37	0	-90	195.0	36	0
RL-12-0405A	587,710	6,139,865	506.91	0	-90	345.0	0	75
RL-12-04A02	588,487	6,140,826	477.93	0	-90	264.0	0	54
RL-12-04A03	588,170	6,140,587	492.32	0	-90	225.0	0	46
RL-12-04A04	587,846	6,140,348	504.71	0	-90	192.0	0	36
RL-12-0502	589,341	6,138,028	500.35	0	-90	345.0	43	1
RL-12-0503	589,012	6,137,786	507.17	0	-90	378.0	50	0
RL-12-0504	588,701	6,137,549	516.42	0	-90	330.0	40	0
RL-12-0505	588,423	6,137,383	532.24	0	-90	240.0	36	0
RL-12-0506A	588,125	6,137,133	544.77	0	-90	183.0	33	0
RL-12-0507	587,731	6,136,824	553.92	0	-90	30.0	0	0
RL-12-0602	588,513	6,141,142	472.87	0	-90	348.5	0	68

Borehole ID	Easting *	Northing*	Elevation	Azimuth	Plunge	Lenth	Sample (count)	
RL-12-0603	588,178	6,140,901	487.92	0	-90	176.8	32	0
RL-12-0604	587,851	6,140,673	502.90	0	-90	199.0	39	0
RL-12-0605	587,542	6,140,430	507.93	0	-90	249.0	47	0
RL-12-0606	587,124	6,140,124	507.87	0	-90	234.0	0	45
RL-12-0607	586,879	6,139,939	516.98	0	-90	210.5	0	44
RL-12-0608	586,575	6,139,706	535.21	0	-90	237.0	0	50
RL-12-0609	586,259	6,139,471	552.09	0	-90	153.0	0	30
RL-12-06A02	588,110	6,141,168	484.85	0	-90	195.0	0	37
RL-12-06A03	587,789	6,140,930	501.78	0	-90	159.2	0	28
RL-12-06A04	587,471	6,140,688	509.21	0	-90	267.6	0	56
RL-12-0702	589,476	6,137,506	503.99	0	-90	432.0	47	0
RL-12-0704	588,835	6,137,022	520.82	0	-90	255.0	36	0
RL-12-0706	588,285	6,136,581	550.02	0	-90	137.0	28	0
RL-12-0802	588,049	6,141,418	482.38	0	-90	276.0	63	4
RL-12-0803	587,711	6,141,180	505.16	0	-90	180.0	33	0
RL-12-0804	587,405	6,140,948	511.26	0	-90	258.0	55	0
RL-12-0805	587,076	6,140,706	511.83	0	-90	258.0	52	0
RL-12-0806	586,795	6,140,463	514.51	0	-90	204.0	41	0
RL-12-0807	586,439	6,140,225	527.48	0	-90	181.4	40	0
RL-12-0808	586,119	6,139,988	557.27	0	-90	120.0	21	4
RL-12-08A03	587,649	6,141,451	503.73	0	-90	132.0	0	25
RL-12-08A04	587,330	6,141,209	511.55	0	-90	143.0	0	26
RL-12-08A05	587,010	6,140,969	515.39	0	-90	192.0	0	39
RL-12-0903	589,133	6,136,626	525.06	0	-90	320.0	48	4
RL-12-0905	588,495	6,136,148	545.26	0	-90	170.0	18	0
RL-12-1002	588,225	6,142,192	460.98	0	-90	390.0	0	77
RL-12-1003	587,893	6,141,942	477.09	0	-90	213.0	29	4
RL-12-1004	587,575	6,141,699	503.67	0	-90	222.0	38	0
RL-12-1005	587,274	6,141,470	512.18	0	-90	111.0	19	0
RL-12-1006	586,936	6,141,237	523.32	0	-90	135.0	23	0
RL-12-1007	586,628	6,140,993	515.52	0	-90	126.1	27	0
RL-12-1008	586,300	6,140,750	523.43	0	-90	153.0	33	0
RL-12-1009	585,974	6,140,509	553.62	0	-90	84.0	19	0
RL-12-1101	589,272	6,136,099	533.02	0	-90	117.0	7	0
RL-12-1103	588,651	6,135,741	536.79	0	-90	90.0	10	0
RL-12-1201	587,938	6,142,574	456.66	0	-90	177.0	0	29
RL-12-1202	587,595	6,142,344	484.82	0	-90	126.0	0	26
RL-12-1203	587,274	6,142,098	498.22	0	-90	162.0	0	33
RL-12-1204	586,946	6,141,858	516.68	0	-90	79.0	0	5
RL-12-1205	586,646	6,141,625	527.54	0	-90	90.6	0	13
RL-12-1206	586,323	6,141,389	519.24	0	-90	54.0	0	6
RL-12-1207	585,999	6,141,150	527.44	0	-90	60.0	0	10
RL-12-1208	585,683	6,140,910	556.14	0	-90	144.0	0	28
RL-12-1602	587,327	6,143,388	461.95	0	-90	263.7	0	41

Borehole ID	Easting *	Northing*	Elevation	Azimuth	Plunge	Lenth	Sample (count)	
RL-12-1603	587,000	6,143,140	473.50	0	-90	267.0	0	51
RL-12-1604	586,671	6,142,909	502.52	0	-90	170.0	0	40
RL-12-1607	585,715	6,142,194	528.93	0	-90	159.0	0	31
RL-12-1608	585,394	6,141,940	535.33	0	-90	78.0	0	11
RL-12-1609	585,182	6,141,783	534.06	0	-90	211.1	0	42
RL-12-1803	587,017	6,143,784	456.05	0	-90	282.0	45	0
RL-12-1804	586,700	6,143,540	475.73	0	-90	231.0	48	0
RL-12-1805	586,383	6,143,299	495.79	0	-90	177.0	28	0
RL-12-1808	585,433	6,142,611	543.43	0	-90	165.0	0	33
RL-12-1809	585,109	6,142,330	532.18	0	-90	249.0	0	50
RL-12-1810	584,790	6,142,103	539.65	0	-90	111.0	0	20
RL-12-2002	586,718	6,144,183	454.08	0	-90	216.0	30	0
RL-12-2003	586,393	6,143,939	476.28	0	-90	204.0	37	0
RL-12-2004	586,081	6,143,709	486.07	0	-90	135.9	26	0
RL-12-2005	585,752	6,143,448	500.32	0	-90	84.0	16	0
RL-12-2007	585,119	6,142,981	571.10	0	-90	134.0	24	0
RL-12-2008	584,820	6,142,781	555.15	0	-90	153.0	0	31
RL-12-2009	584,497	6,142,500	559.62	0	-90	213.0	0	44
RL-12-2206	585,335	6,143,768	525.06	0	-90	60.0	0	9
RL-12-2207	584,956	6,143,507	590.83	0	-90	60.0	0	11
RL-12-2208	584,656	6,143,294	588.77	0	-90	261.0	0	54
RL-12-2209	584,342	6,143,028	568.94	0	-90	123.0	0	1
RL-12-2402	586,265	6,145,088	450.56	0	-90	66.0	0	4
RL-12-2403	585,968	6,144,852	465.98	0	-90	129.0	0	25
RL-12-2404	585,637	6,144,620	493.84	0	-90	138.0	0	27
RL-12-2408	584,364	6,143,670	607.50	0	-90	192.0	0	35
RL-12-2801	586,012	6,146,161	445.09	0	-90	72.0	0	14
RL-12-2802	585,720	6,145,924	449.97	0	-90	102.2	0	15
RL-12-2803	585,342	6,145,655	480.50	0	-90	180.0	0	36
RL-12-2804	585,053	6,145,444	500.85	0	-90	147.0	0	28
RL-12-3704	588,268	6,138,783	510.52	0	-90	198.0	0	31
Total 117 boreholes						**24,554.6**	**1,847**	**2,525**

* UTM Coordinates (Nad83, Zone 19)

10.3 SRK Comments

In the opinion of SRK, the sampling procedures used by WCSLIM conform to industry best practices and the resultant drilling pattern is sufficiently dense to interpret the geometry and the boundaries of the iron mineralization with confidence. All drilling sampling was conducted by appropriately qualified personnel under the direct supervision of appropriately qualified geologists.

11 Sample Preparation, Analysis and Security

This section documents the sampling preparation, analyses and security procedures for the core sampling informing the mineral resources discussed herein. Information on sampling conducted on the other areas of the Sunny Lake project are not relevant to this technical report and are therefore not discussed here. Refer to the previous technical report prepared for the property by SRK in 2010.

11.1 Sample Preparation and Analyses

11.1.1 Geological Reconnaissance Samples – 2009

Independent geologists collected samples averaging 2 to 3 kilograms of broken rock into a numbered plastic bag containing a distinct laboratory sample tag. Field samples were submitted to the ALS Minerals in Val-d'Or, Québec for preparation using standard preparation procedures (drying, weighting, crushing, splitting, and pulverization to 85 percent passing 75 microns).

Prepared samples were sent to the ALS Minerals in North Vancouver for assaying for a suite of 31 elements including iron using an aqua regia digestion and inductively coupled plasma optical emission spectroscopy (ICP-AES; method code ME-ICP61).

The management system of the ALS Group laboratories is accredited to ISO 9001 by QMI and the North Vancouver laboratory is accredited ISO 17025 by the Standards Council of Canada for a number of specific test procedures, including the method used to assay samples submitted by WCSLIM. ALS laboratories also participate in a number of international proficiency tests, such as those managed by CANMET and Geostats.

Twelve samples prepared by ALS Minerals were submitted to the COREM Laboratory in Québec City, Québec for mineralogical characterization. The testing included optical and electronic microscopy, X-ray diffraction, and X-ray fluorescence on samples ground to approximately 90 percent passing -4 mesh screen (4.75 millimeters).

The management system of the COREM laboratory is accredited to ISO 9001 by Bureau de Normalisation du Québec. The analytical laboratories are also certified ISO 17025 by the Standards Council of Canada for certain testing procedures including those used to assay the WCSLIM samples.

Three composite samples prepared by ALS Minerals in Val-d'Or were also submitted to SGS in Lakefield, Ontario for preliminary beneficiation tests. The SGS Lakefield laboratory is accredited ISO 17025 by Standards Council of Canada for certain testing procedures.

11.1.2 Core Drilling Sampling - 2011

All core samples collected in 2011 were submitted to Actlabs in Ancaster, Ontario by freight in rice bags tied with tamper resistant security tags. Actlabs is accredited to ISO/IEC Guideline 17025:2005 by the Standards Council of Canada for a number of specific test procedures, including the method used to assay the samples submitted by WCSLIM.

Samples were assayed for iron and a suite of 11 other elements reported in oxide form (SiO_2, Al_2O_3, Fe_2O_3 reported in Total Fe, MnO, MgO, CaO, Na_2O, K_2O, TiO_2, P_2O_5, Cr_2O_3, V_2O_5, and LOI) using lithium fusion and X-ray fluorescence spectrometry (method code QOP FUSION XRF 4C). Samples were also assayed for sulphur using combustion and infrared analysis (method code QOP Carbon & Sulphur 4F-F, S Infrared).

In early 2012, WCSLIM submitted 19 core samples from the 2011 drilling program for Davis Tube testing.

11.1.3 Core Drilling Sampling - 2012

Core samples collected in 2012 were submitted to both Actlabs in Ancaster, Ontario and SGS in Lakefield, Ontario by freight in rice bags tied with tamper resistant security tags. Both labs are accredited to ISO/IEC Guideline 17025:2005 by the Standards Council of Canada for a number of specific test procedures, including the method used to assay samples submitted by WCSLIM.

Samples at Actlabs were assayed for iron and a suite of 11 other elements reported in oxide form (SiO_2, Al_2O_3, Fe_2O_3 reported in Total Fe, MnO, MgO, CaO, Na_2O, K_2O, TiO_2, P_2O_5, Cr_2O_3, V_2O_5 and LOI) using lithium fusion and X-ray fluorescence spectrometry (method code QOP FUSION XRF 4C). Samples were also assayed for sulphur using combustion and infrared analysis (method code QOP Carbon & Sulphur 4F-F, S Infrared).

At SGS samples were assayed for iron and a suite of 11 other elements reported in oxide form (SiO_2, Al_2O_3, Fe_2O_3, MnO, MgO, CaO, Na_2O, K_2O, TiO_2, P_2O_5, Cr_2O_3, V_2O_5 and LOI) using borate fusion and X-ray fluorescence spectrometry (method code GO/GC/GP/GU_XRF76V / R).

Throughout 2012, WCSLIM submitted core samples for Davis Tube testing.

11.2 Specific Gravity Data

Specific gravity measurements were made using the water displacement method at intervals of 2 to 4 meters in the Sokoman Formation. Specific gravity was determined by weighing dry core samples in air and immersed in water. Generally a representative piece of core, 10 to 15 centimeters in length, was selected prior to splitting. Results were recorded directly into a Microsoft Excel spreadsheet. Specific gravity was measured on every sample during the 2011 and 2012 drilling program.

11.3 Quality Assurance and Quality Control Programs

Quality control measures are typically set in place to ensure the reliability and trustworthiness of exploration data. These measures 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 implemented to monitor the precision and accuracy of the sampling, 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 regularly duplicating and replicating assays and inserting quality control samples to monitor the reliability of assaying results throughout the sampling and assaying process. Check assaying is normally performed as an additional test of the reliability of assaying results; it generally involves re-assaying a set number of sample rejects and pulps at a secondary umpire laboratory.

The exploration work conducted by WCSLIM was carried out using a quality assurance and quality control program meeting industry best practices for delineation stage exploration properties. Standardized procedures were used in all aspects of the exploration data acquisition and management including mapping, surveying, drilling, sampling, sample security, assaying, and database management.

During the 2011 and 2012 core drilling programs, the analytical quality control measures implemented by WCSLIM include the use of control samples (sample blanks, certified reference materials, in-house reference materials, and field duplicate samples) at a rate of 10 control samples every 100 samples, in addition to choosing ISO accredited primary laboratories.

Certified reference materials were sourced from Natural Resources Canada's CANMET Mining and Mineral Sciences Laboratories ("CANMET") in Ottawa, Ontario. WCSLIM also commissioned ALS Minerals to create a set of in-house control samples with samples from Rainy Lake. The in-house standards were prepared from PGC material from the Rainy Lake property. ALS Minerals submitted the samples to a set of four different laboratories for round robin testing where the laboratories analysed each standard five times.

WCSLIM used five distinct reference materials, with certified assay values ranging from 27.36 to 60.73 percent iron (Table 11.1).

Field duplicates were also inserted within the samples submitted for assaying in 2011 and 2012. Field duplicate samples were collected by splitting the remaining half core in half and assigning a separate sample number out of sequence from the original samples.

Table 11.1 – Specifications of the Certified Control Samples Used by WCSLIM during 2011 and 2012 Drilling at Rainy Lake

Reference Material	Fe (%)	Std Dev. (Fe %)	95 % Confidence Interval (Fe %)	Number of Samples
STD-4	27.36	0.23	-	29
STD-3	30.30	0.21	-	38
STD-2	31.65	0.10	-	21
STD-1	39.05	0.08	-	13
SCH-1	60.73	-	0.09	32

11.4 SRK Comments

In the opinion of SRK the sampling preparation, security and analytical procedures used by WCSLIM are consistent with generally accepted industry best practices and are, therefore, adequate for the purpose of mineral resource estimation.

12 Data Verification

12.1 Verifications by WCSLIM

The exploration work carried out on the Rainy Lake property was conducted by WCSLIM personnel and qualified subcontractors. WCSLIM implements a series of routine verifications to ensure the collection of reliable exploration data. All work is conducted by appropriately qualified personnel under the supervision of qualified geologists. In the opinion of SRK, the field exploration procedures used at Rainy Lake generally meet industry practices.

The quality assurance and quality control program implemented by WCSLIM is comprehensive and supervised by adequately qualified personnel. Exploration data were recorded digitally to minimize data entry errors. Core logging, surveying, and sampling were monitored by qualified geologists and verified routinely for consistency. Electronic data were captured and managed using an internally managed Microsoft Access database, and backed up daily.

Assay results were delivered by the primary laboratories electronically to WCSLIM and were examined for consistency and completeness.

12.2 Verifications by SRK

12.2.1 Site Visit

In accordance with National Instrument 43-101 guidelines, Dominic Chartier, P.Geo., (OGQ#874, PEGNL#06306) of SRK visited the Rainy Lake property on October 12 and 13, 2011 accompanied by Wenlong Gan, P.Geo., (APGO#2043) and Matthew Chong of WCSLIM. Mr. Chartier and Dr. Jean-Francois Couture, P.Geo., (OGQ#1106, APGO#0197) also visited the project on May 15 to 17, 2012 accompanied by Mr. Gan, Mr. Chong and Zhihuan Wan, P.Geo (APGO#2072) of WCSLIM. At the time of the visits, drilling activities were ongoing on the project. The purpose of the site visits was to review the exploration database and validation procedures, review exploration procedures, define geological modelling procedures, examine drill core, interview project personnel, and collect all relevant information for the preparation of an initial mineral resource model and the compilation of a technical report.

SRK was given full access to relevant data and conducted interviews with WCSLIM personnel to obtain information on the past exploration work, to understand procedures used to collect, record, store, and analyze historical and current exploration data.

12.2.2 Verifications of Analytical Quality Control Data

WCSLIM made available to SRK exploration data in the form of a Microsoft Access database. This database aggregated the assay results for the quality control samples received to date, and was accompanied by comments from WCSLIM personnel.

SRK aggregated the assay results for the external quality control samples for further analysis. Sample blanks and reference materials data were summarized on time series plots to highlight the performance of the control samples. Paired data (field duplicates) were analyzed using bias charts, quantile-quantile, and relative precision plots. The analytical quality control data produced by WCSLIM in 2011 and 2012 are summarized in Table 12.1.

Table 12.1 – Summary of Analytical Quality Control Data Produced by WCSLIM in 2011 and 2012 on the Rainy Lake Property

	2011 Drilling	(%)	2012 Drillings	(%)	Total	(%)	Comment
Sample Count	1,786		1,599		3 385		
Blanks	72	4.03 %	70	4.38 %	142	4.19 %	
Standards	77	4.31 %	56	3.50 %	133	3.93 %	
STD-4	18		11		29		WCSLIM (27.36 % Fe)
STD-3	24		14		38		WCSLIM (30.30 % Fe)
STD-2	9		12		21		WCSLIM (31.65 % Fe)
STD-1	13		13				WCSLIM (39.05 % Fe)
SCH-1	26		6		32		CANMET (60.73 % Fe)
Field Duplicates	22	1.23 %	35	2.19 %	57	1.68 %	
Total QC Samples	**171**	**9.57 %**	**161**	**10.07 %**	**332**	**9.81 %**	

In general, the performance of the control samples inserted with samples submitted for assaying is acceptable. The performance of the in-house control samples show that Actlabs in 2011 performed above two standard deviations for STD-3 and STD-2 and above the mean (within two standard deviations) for STD-4. Whereas, the performance of the in-house samples by Actlabs and SGS in 2012 performed better within two standard deviations for STD-4 and STD-3 and within or below two standard deviations for STD-2 and STD-1.

However, the expected values and standard deviations calculated by ALS Minerals for the in-house standards are generally inconsistent with the values received from Actlabs and SGS. The performance of the control samples at both laboratories is nonetheless consistent.

Actlabs and SGS appears to have had difficulty in delivering assay results for certified reference material SCH-1 with 25 percent of samples falling below four times the 95 percent confidence interval.

Paired assay data produced by Actlabs and SGS and examined by SRK suggest that iron grades can be reasonably reproduced. Ranked half absolute difference ("HARD") plots show that 96.3 percent of the field duplicate sample pairs by Actlabs and 100 percent of the field duplicate sample pairs by SGS have HARD below 10 percent, which suggests excellent analytical reproducibility of the total iron grades.

In the opinion of SRK, the analytical results delivered by Actlabs and SGS for the core samples from the Rainy Lake property are sufficiently reliable to support mineral resource evaluation. SRK recommends that for future drilling programs, WCSLIM should review assay results of analytical quality control samples using bias charts when assays are received from the primary laboratory to monitor reliability and detect potential assaying problems. Batches under review for potential failures should be recorded in a quality control spreadsheet, investigated, and corrective measures taken when required.

13 Mineral Processing and Metallurgical Testing

This section of the report summarizes the previous PEA testwork and discusses the metallurgical testwork that was conducted in 2011-2012 at COREM, Quebec City, on drill core samples from the Rainy Lake deposit. Based on the metallurgical results and the benchmark from nearby projects, a process flowsheet is proposed and a weight recovery model developed.

13.1 Historical Testwork prior to PEA

This section presents a summary of the results of interest from the testwork that was realized prior to the PEA. Complete results can be found in the reports submitted from the different laboratories and that are listed in Section 27.

13.1.1 COREM T1119: Mineralogical Characterization of Samples from the Sunny Lake Project

In 2005, 0849873 BC Inc., an affiliate of Century Iron Mines, mandated COREM to perform mineral characterization on twelve (12) samples from the Sunny Lake project. COREM received six (6) samples from the Lac Le Fer ("LLF") deposit of PGC, URC and LRGC lithologies, and six (6) samples from the Rainy Lake ("RL") deposit of the PGC lithologies. Samples received were already crushed and ground to 90 % 4.75 mm.

Table 13.1 presents the main characterization results for the RL samples.

Table 13.1 – T1119 - Rainy Lake PGC Samples Composition

Sample	Head Assay (%)			Mineralogy (%)				
	Total Fe[1]	Magnetite[2]	SiO_2	Hematite	Magnetite	Iron Hydroxides	Sum of Valuable Minerals	Quartz
178000	28.9	17	57.3	22	17	1	40	57
177998	41.8	19	38.8	39	19	n.d.	58	37
177999	30.8	33	54.6	8	33	n.d.	41	51
177995	32.5	33	52.9	11	33	1	45	51
178077	34.3	43	50.0	n.d.	43	1	45	46
178053	38.9	34	40.6	n.d.	34	5	39	21

[1] Based on % Fe_2O_3

[2] Based on Satmagan measurement

n.d. not detected or under 1 % limit

CIMA+ M03339A

Chemical analysis and mineralogical characterization determined that the total iron grade varied from 29 % to 42 %. The main valuable iron-bearing minerals identified by microscopy and X-ray diffraction ("XRD") were iron oxides (hematite and magnetite) and iron hydroxides (goethite). The magnetite content varied from 17 % to 43 %.

The main gangue mineral identified was quartz. High density contaminants (i.e. sulphides, manganese bearing minerals and carbonates) were also detected in the samples. A preliminary study concluded that these contaminants will tend to report to the concentrate stream along with the iron in a gravity separation process.

Scanning electron and optical microscopic observations of the head samples showed very fine hematite and magnetite grains intimately associated with quartz grains. Liberation of the fine magnetite and hematite grains in mixed particles would require grinding to less than 150 µm for the magnetite and less than 100 µm for the hematite.

13.1.2 SGS 12385-001: Mineralogical Characteristics of Samples from the Sunny Lake Project

In March 2010, BBA Inc, on behalf of 0849873 BC Inc., mandated SGS Minerals Services to perform mineral characterization on three (3) samples from the Sunny Lake project. SGS received two (2) samples from the Lac Le Fer deposit of PGC and URC lithologies, and one (1) sample from the Rainy Lake deposit of PGC lithology. The samples received had already been finely ground (90-95 % passing-75 µm), which limited the characterization testwork to chemical assays and mineralogy. The chemical testwork consisted of head assays and size-by-size assays, whereas the mineralogy testwork consisted of determining the mineral abundance, deportment, average grain size and association.

Table 13.2 summarizes the results for the head assays and mineralogy testwork for the Rainy Lake sample. A good correlation is observed between the magnetite values obtained by chemical and mineralogical analyses.

Table 13.2 – SGS 12385-001 - Rainy Lake Samples Composition

Sample	Head Assay (%)			Mineralogy (%)				
	Total Fe[1]	Magnetite[2]	SiO_2	Magnetite	Iron-Oxides	Fe Deportment in Iron-Oxides	Quartz	Micas/ Clays
RL-PGC	32.9	30.4	48.7	31.1	43.3	92.2	40.6	13

[1]: Based on % Fe_2O_3
[2]: Based on Satmagan measurement

CIMA+ M03339A

The main gangue mineral was quartz, followed by mica/clays. The magnetite-to-hematite ratio was approximately 70:30. The iron was predominantly found as iron oxides (over 92 %), these later being well liberated (more than 85 % of the oxides were liberated).

13.1.3 COREM T1268-1 Identification of Iron-Bearing Minerals in Drilled Core Samples from the Sunny Lake Project

In 2011, WCSLIM mandated COREM to perform mineral characterization on eleven (11) samples from the Sunny Lake project. Five (5) samples of different lithologies from the Rainy Lake deposit were selected by WCSLIM from a previous project from COREM (T1217). Samples were already pulverized to 90 % - 200 µm.

Table 13.3 presents the main characterization results for the RL samples.

Table 13.3 – T1268-1 - Rainy Lake Samples Main Composition

Sample	Unit	Head Assay (%)			Mineralogy (%)		
		Total Fe[1]	Magnetite[2]	SiO_2	% of Valuable Total Fe	Grade in Valuable Fe	Major Iron Bearing Mineral
1056463	PGC	47.8	48.3	25.0	92	44	Magnetite
1056029	URC	41.2	9.1	40.8	97	40	Hematite
1056194	LRGC	29.1	18.6	55.3	83	24	Hematite / Magnetite
1056305	JUIF	35.4	18.5	49.7	97	34	Hematite / Magnetite
1056028	LC	14.0	3.7	78.8	26	4	-

[1]: Based on % Fe_2O_3
[2]. Based on Satmagan measurement

Chemical analysis and mineralogical characterization determined that the total iron grade varied from 14 % to 48 %. The main iron-bearing minerals identified by microscopy and X-ray diffraction were iron oxides (hematite and magnetite) and iron hydroxides (goethite). The PGC lithology sample contained mainly magnetite as valuable iron-bearing mineral. The URC lithology sample contained mainly hematite while the JUIF lithology sample contained magnetite and hematite in similar proportions. The LC sample was mainly gangue material. For all samples except the LC sample, more than 83 % of the iron is valuable iron.

The magnetite content in samples was found to vary widely (between 4 % and 48.3 % for RL deposit). The main gangue mineral identified in all samples was quartz, and in some cases minnesotaite.

Scanning electron and optical microscopic observations of the head samples showed very fine hematite and magnetite grains intimately associated with gangue minerals.

13.2 Benchmark

The Rainy Lake property is located in a very active geological exploration area. Table 13.4 presents the nearby projects that were used as benchmarking for process selection. The data shown in this table is available to the public. The table presents the general production targets and the processes involved.

Table 13.4 – Benchmarking of Nearby Iron Ore Projects

Parameter	Unit	Century Duncan Lake	New Millenium KéMag/LabMag	Adriana Resources Otelnuk Lake	Alderon Kami
Ore					
Crude Iron	% Fe	24.8	32.5	29.8	29.5
Crude Magnetite	% Magnetite	n/a	27.0	17.6	21.1
General Concentrator					
Run of Mine	Mtpy	41.3	86.1	182.0	22.9
Concentrate Production	Mtpy	11.6	22.0	49.1	8.0
Concentrate Silica Content	% w/w	5.0	1.5	4.0	3.0
Concentrate Total Fe Content	% w/w	67.6	70.9	68.5	65.5
Weight Recovery	% w/w	28.0	25.5	27.0	35.1
Concentrate Size	µm	P_{85} = 75	P_{100} = 53	P_{80}=53	P_{80}=260
Concentrator - Circuit Comparison					
Crushing	-	Gyratory Crusher	Gyratory + Cone Crushers	Gyratory + Cone Crushers	Gyratory Crusher
Primary Grinding	-	SAG Mill	HPGR	HPGR	AG Mill
Secondary Grinding	-	Ball Mill	Ball Mill	Ball Mill	Ball Mill
Primary Separation	-	Magnetic-Cobbers	Magnetic-Cobbers	Magnetic-Cobbers	Spirals
Secondary Separation	-	Magnetic-Cleaner & Finishers	Magnetic-Rougher & Finishers	Magnetic-Rougher & Finishers	Magnetic-Cobber, Rougher & Finisher
Final Concentration	n/a	n/a	Regrind + Magnetic Separation & Flotation	n/a	n/a
General Pellet Plant					
Pellets Type	n/a	Acid	BF & DR	BF & DR	n/a
Pellets Production Target	Mtpy	12.0	17.0	50.0	n/a
Pellet Plant - Circuit Comparison					
Flotation	-	No	No	No	n/a
Grinding Equipment	-	No	No	No	n/a
Balling Equipment	-	Balling Disc	Balling Disc	Balling Disc	n/a
Grate Furnace Area	m^2	n/a	2 X 816 m^2	6 X 848 m^2	n/a

13.3 PEA Study Metallurgical Testwork

This section addresses the metallurgical testwork program undertaken at COREM. Part of the testwork was undertaken under Soutex's supervision. The project and testwork program were suspended mid-2012 therefore all testwork initially planned has not been completed.

13.3.1 PEA Testwork Plan

One of the main objectives of the PEA study is to develop a process flowsheet, using Rainy Lake deposit characteristics that would allow the economical assessment of the project. The few characterization tests that were done before the PEA indicated that the ore was mainly magnetic and fine grinding was required to liberate the valuable iron. The data gathered was not yet sufficient to complete the PEA study.

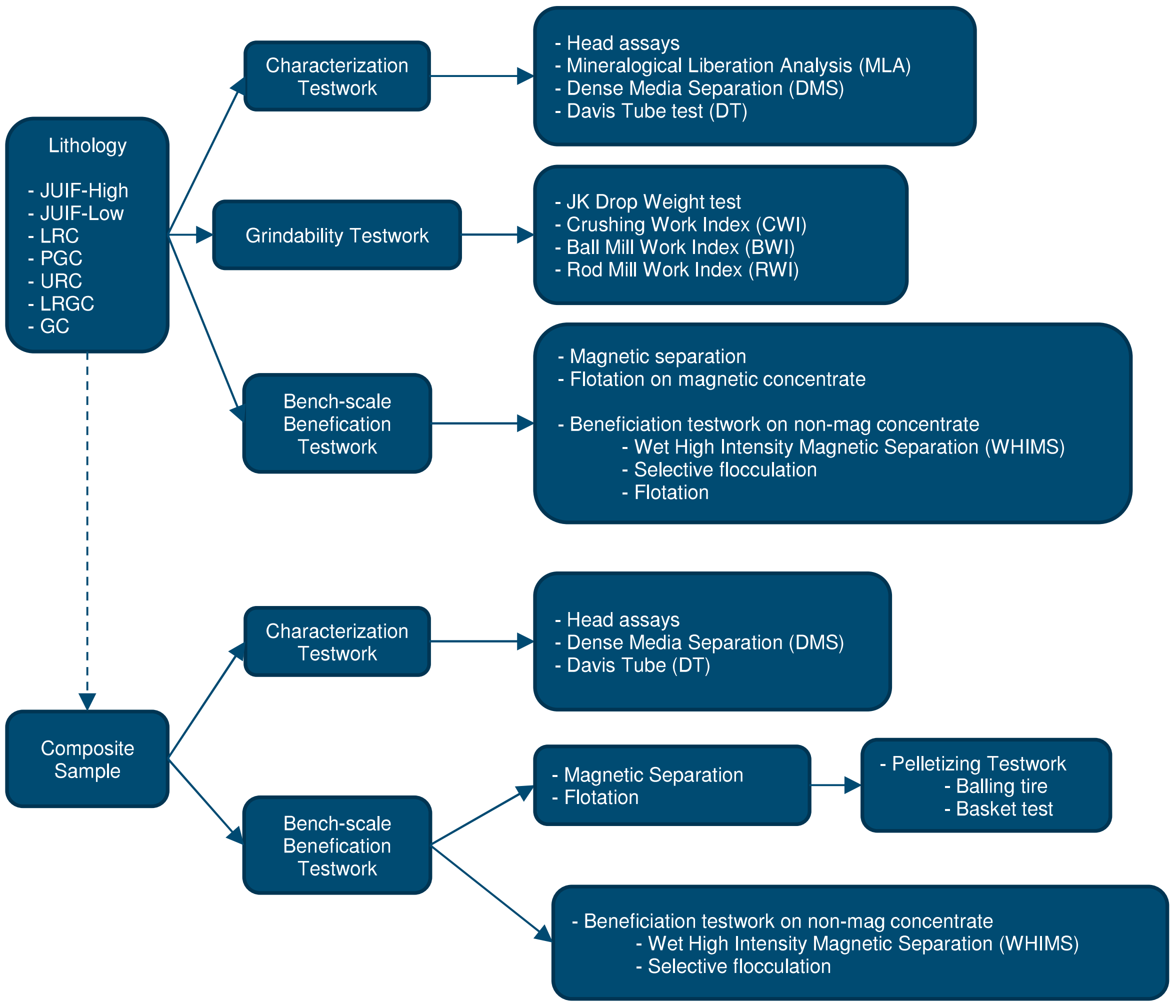
A metallurgical testwork program was hence developed by Soutex at the early stages of the study, in order to characterize the Rainy Lake deposit ore body. The objectives of the testwork program were to:

- Characterize each lithology unit of the deposit;
- For each lithology unit and for a composite sample:
  - Evaluate the ore's amenability to be processed by magnetic separation in order to produce a commercially acceptable, quality product. An important part of the testwork consisted in evaluating the iron liberation size;
  - Evaluate the potential of gravity separation to produce a commercially acceptable, quality product or to contribute to an increase in iron and weight recoveries.
- Perform preliminary pelletizing tests.

Figure 13.1 presents the high level metallurgical test plan for the project. All metallurgical testwork was carried out at COREM. Only relevant results to the study are presented later in sections 13.3.3 to 13.3.7. A complete description of the testwork is available in COREM's reports [4, 5].

The testwork results were then used in defining a process flowsheet, which is described later in this section of the report (Section 13.4). A recommended testwork program for subsequent testwork required for the next study phase of the project is also presented (Section 13.5).

Figure 13.1 – Metallurgical Testwork Plan



13.3.2 Sample Selection and Preparation

13.3.2.1 Sample Selection

The material requirements to perform the metallurgical testwork presented above in Figure 13.1 were evaluated by Soutex [6].

Based on the geological studies of the Rainy Lake deposit, the extractable taconite is situated in the following six (6) stratigraphic units, which differ in iron grade and oxidation potential:

- Jasper Upper Iron Formation;
- Green Chert;
- Upper Red Chert;
- Pink Grey Chert;
- Lower Red Chert;
- Lower Red Green Chert.

Given this assemblage of the deposit, WCSLIM geologists selected the drill core hole's location so that the material extracted would:

- Have enough proportion of each lithology to realize the testwork described in Figure 13.1 for each lithology;
- Representative of the first years of operation.

In 2012, the following four (4) HQ drill core samples of 200-250 m in length were extracted to perform the metallurgical testwork: RL 12 0203; RL 12 3704; RL 12 0606; and RL 12 1201.

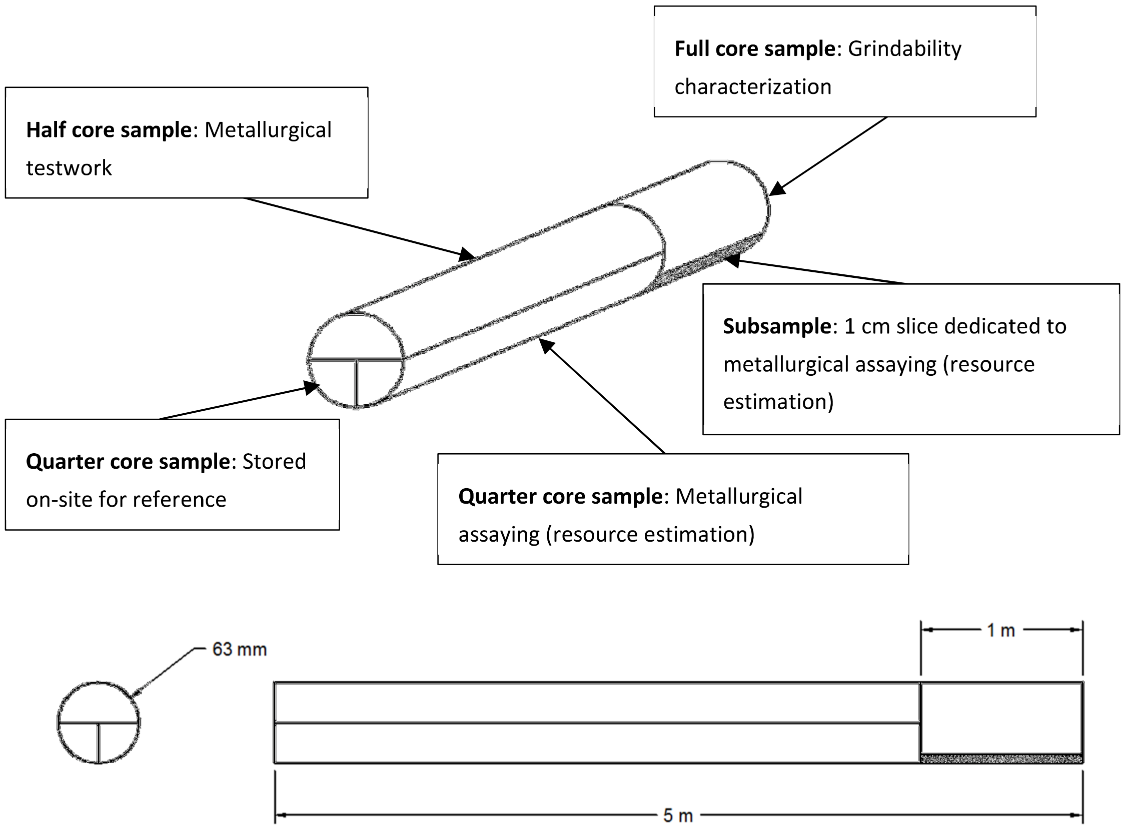
During the sample selection process, it was observed that the JUIF lithology presented different levels of magnetic susceptibility, which is indicative of different iron magnetic grades. An analysis determined that there exist significant differences in the iron magnetic grades within the JUIF lithology unit to warrant its separation into strongly magnetic and weakly magnetic samples [7]. The JUIF sample was thus reclassified into two (2) different lithology units:

- Jasper Upper Iron Formation – strongly magnetic;
- Jasper Upper Iron Formation – weakly magnetic.

13.3.2.2 Sample Preparation

The drill cores from drilling were split as depicted in Figure 13.2 [8]. Only samples with no lithology intercept were prepared for metallurgical testwork.

Figure 13.2 – Typical Sample Partitions and Dimensions



A composite sample was produced as per the lithologies' proportions recommended by the geologists and that are presented in Table 13.5.

Table 13.5 – Composite Sample Composition

Lithology	%
JUIF-High	25
JUIF-Low	8
LRC	18
PGC	20
URC	14
LRGC	13
GC	2

13.3.3 Grindability Testwork

Grindability testwork was conducted on 150 kg full core samples and, includes JK Drop Weight, Crushing Work Index ("CWI"), Rod Mill Work Index ("RWI"), Ball Mill Work Index ("BWI") and Abrasion Index ("AI") tests, as depicted in Figure 13.1.

Testwork results on GC lithology units, as well as LRGC JK Drop Weight and CWI test results are not available since both samples were accidentally mixed during sample preparation. No grindability testwork was conducted on the composite.

The grindability test results obtained are summarized in Table 13.6 and Table 13.7. These results were used to perform preliminary sizing and power evaluation of the crusher, primary grinding mill and the regrind mill.

Table 13.6 – Grindability Tests Results Summary

Lithology	Drop Weight Test			Abrasion JK	Crushing Work Index (kWh/t)	Specific Gravity (g/cm3)	Rod Mill Work Index (kg/kWh)	Ball Mill Work Index (kg/kWh)
	A	b	A x b	ta				
JUIF High	92.45	0.33	30.70	0.16	13.48	3.24	17.23	16.38
JUIF High (CWI repeat)	n.a.	n.a.	n.a.	n.a.	15.99	3.42	n.a.	n.a.
JUIF Low	68.64	0.54	36.99	0.22	15.42	3.30	17.41	18.90
LRC	90.18	0.36	32.37	0.22	13.81	3.18	15.22	14.90
PGC	92.73	0.33	31.04	0.23	14.25	3.33	14.79	15.39
URC	100.00	0.30	29.62	0.19	15.57	3.43	15.27	15.68
LRGC	n.a.	n.a.	n.a.	n.a.	n.a.	n.a.	16.70	16.21

Table 13.7 – Bond Abrasion Tests Results Summary

Lithology	Bond Abrasion Test					
	Bond Abrasion Index (AI) (g)	Wet Rod Mill for Rods (kg/kWh)	Wet Rod Mill for Liners (kg/kWh)	Wet Ball Mill for Balls (kg/kWh)	Wet Ball Mill for Liners (kg/kWh)	Crusher for Gyratory, Jaw and Cone (kg/kWh)
JUIF High	0.7389	0.1486	0.0144	0.1425	0.0107	0.0395
JUIF Low	0.5990	0.1423	0.0135	0.1327	0.0100	0.0338
LRC	0.5468	0.1397	0.0131	0.1286	0.0098	0.0316
PGC	0.6225	0.1435	0.0137	0.1345	0.0102	0.0347
URC	0.6533	0.1449	0.0139	0.1367	0.0103	0.0360
LRGC	0.5021	0.1372	0.0128	0.1249	0.0095	0.0298

The drop weight test results indicate that, in terms of impact breakage, the URC lithology unit is considered a very hard ore type (A x b < 30), whereas all other lithology units are classified as hard ore types (A x b lies between 30 – 38). In terms of abrasion breakage, all lithology units are classified as very hard ore types (ta < 0.24).

Crusher Work Index values are within the same range (between 13.5-16 kWh/t). They indicate that all lithology units classify as hard ore types (CWI between 14-20 kWh/t) except the first repetition of the JUIF-High and LRC which are in the high limit of the medium range of CWI values (CWI between 9-14 kwh/t).

With values between 14-20 kWh/t, both Rod Mill and Ball Mill Work Index test results indicate all lithology units classify as hard ore types.

The Abrasion Indexes determined for all lithology units are in accordance with the typical abrasion index for taconite (AI = 0.6237).

13.3.4 Ore Characterization

Characterization testwork is depicted in Figure 13.1. It includes head assays, Mineralogical Liberation Analysis , Dense Media Separation and Davis Tube tests. Testwork was done on 20 Kg of each lithology pre-crushed to -1.7 mm.

13.3.4.1 Head Assays

Table 13.8 present the lithologies' head grades measured on the half core samples pre-crushed to 1.7 mm. The head assays measured show some differences with the historical data, the PGC lithology having less magnetite while the URC and LRGC lithologies have a higher magnetic content.

The GC lithology unit presents the lowest total iron and magnetite grades (19.2 and 3.3 % respectively) and as such is considered as waste. The JUIF-Low lithology unit has the next lowest magnetic content with 8.9 %. All the other lithology units exhibit a higher concentration of magnetite, between 18.0 % and 28.9 %.

The average magnetite content of the optimized pit for the PEA study is 27 % (Section 13.4.2.2), which was used as the basis for the mass balance and sizing of the equipment. At 21 %, the magnetite content of the composite sample is lower than the optimized pit.

In general, the lithology samples analysed do not point to any issues with regards to manganese, sulfur and phosphorus, these elements being of relatively low to moderate concentration.

Table 13.8 – Rainy Lake Samples Head Grades

Lithology	S.G. g/cm³	Analysis (%)														
		Magnetite [2]	Fe [1]	SiO_2	Al_2O_3	MgO	CaO	Na_2O	K_2O	TiO_2	MnO	P_2O_5	Cr_2O_3	LOI	S_T	C_T
JUIF-High	3.40	28.9	31.04	44.40	0.61	1.75	2.94	0.1	0.11	0.06	0.84	0.04	0.03	4.63	0.02	1.49
JUIF-Low	3.34	8.9	29.08	44.90	0.57	2.19	3.22	0.1	0.12	0.09	0.94	0.04	0.02	6.56	0.02	1.94
LRC	3.37	21.0	29.43	45.90	0.10	1.98	5.24	0.1	0.04	0.05	0.52	0.03	0.04	4.63	0.01	1.33
PGC	3.41	18.0	30.97	44.50	0.10	1.91	4.27	0.1	0.02	0.05	0.54	0.03	0.04	4.77	0.01	1.43
URC	3.43	20.2	31.39	40.60	0.20	2.42	3.94	0.1	0.04	0.05	1.10	0.03	0.03	6.80	0.08	2.08
LRGC	3.30	24.7	28.87	45.50	0.20	3.81	2.50	0.1	0.05	0.05	0.63	0.02	0.01	5.87	0.02	1.60
GC	3.04	3.3	19.22	43.80	1.50	5.14	4.01	0.1	0.36	0.19	1.39	0.05	0.02	15.50	0.45	4.03
Composite - measured	**3.44**	**20.9**	**30.27**	**44.20**	**0.30**	**2.24**	**3.56**	**0.1**	**0.07**	**0.05**	**0.72**	**0.04**	**0.02**	**5.55**	**0.04**	**1.55**
Composite - calculated	**3.37**	**21.4**	**30.11**	**44.33**	**0.32**	**2.29**	**3.75**	**0.1**	**0.07**	**0.06**	**0.75**	**0.03**	**0.03**	**5.49**	**0.03**	**1.63**

[1]: Based on % Fe_2O_3
[2]. Based on Satmagan measurement

13.3.4.2 Mineralogical Liberation Analysis

In order to define the iron distribution in the minerals contained in the samples and the liberation size of the valuable iron minerals, some mineralogical characterization tests were performed. The results of this testwork were to be used to orient the beneficiation testwork and to better understand and interpret the beneficiation results.

All lithology units were stage-ground to 100 % -106, -75, -45, -25 µm and MLA was performed on the top size fraction of each ground product, i.e. -106+75, -75+53, -45+25 µm.

The mineral characterization showed that:

- The main gangue mineral is quartz but other gangue minerals such as silicates, carbonates, micas, and also chlorite are also present. Hence, JUIF-High and Low, LRC, PGC and URC lithology units contained 35-45 % quartz, 10-18 % silicates and 2-13 % carbonates. With around a third of quartz, a third of silicates and a third of carbonates, the GC lithology unit is different from the other lithologies;
- The iron is mainly distributed in valuable iron-bearing minerals (more than 85 % found in iron oxides) in all samples except the GC and LRGC. GC sample contains less than 20 % of the total iron as valuable iron-bearing mineral, 50 % of the iron being in carbonates-bearing minerals;
- Excluding the GC sample, all lithologies contain between 30-50 % of iron oxides. Containing around 30 % of hematite, samples JUIF-Low and PGC contain the higher proportion of hematite, while JUIF-High and LRGC contain the higher proportion of magnetite (20-35 %);
- The proportion of free iron particles (>90 % area of iron oxides) is around 56 % for the ground fraction -106+75 µm, 65 % -75+53 µm and 78 for the fraction -45+25 µm.

Non-liberated iron particles are mainly binary particles associated with quartz. Between 5-15 % of the iron particles occur as ternary particles.

13.3.4.3 Dense Media Separation Tests

DMS tests were performed on each lithology pre-crushed at -1.7 mm and for eight (8) size fractions between 1180 µm and 75 µm. The finer fractions (-75 µm) were not processed because they do not respond well to DMS. The objective was to evaluate whether the ore was amenable to gravity concentration. The dense media used for the tests had a 3.3 density.

The results showed that producing a concentrate with gravity concentration would be at the cost of a very low iron recovery.

13.3.4.4 Davis Tube Testwork

Preliminary DT tests were performed on each lithology pre-crushed at -1.7 mm and for each of the particle size fractions. Results showed that a silica grade below 5 % was only reached for the -45 µm fraction, suggesting that the Rainy Lake ore would require a grind fineness of at least 100 % passing 45 µm.

Liberation DT tests were performed on the material that was prepared for the MLA testwork (Section 13.3.4.2). All lithology units were stage-ground to -106, 75, 45, 25 µm and DT were performed on each ground product.

Table 13.9 presents a summary table of the DT test results. The following general observations can be made in analyzing these results:

- A silica grade below the 4.5 % target grade is reached for all lithology units ground to -25 µm, except the JUIF-Low lithology, for which the silica grade obtained is 4.7 %;
- At a -45 µm grind, the 4.5 % target silica grade is only reached with three (3) lithology units (URC, LRGC, GC). The other lithology units (JUIF-High, LRC, PGC) are close to the target with a silica grade below 5 % while JUIF-Low lithology has a magnetic concentrate at 7.4 % silica;
- Coarser grind sizes do not permit the target silica grade to be reached for all lithology units;
- Magnetite recovery is good (96-98 %) except for the GC lithology unit for which magnetite recovery is in the 80-90 % range.

All DT results show that a target grind size of around 35-45 µm should lead to a final concentrate with the required 4.5 % silica grade.

Table 13.9 – DT Tests on Ground Products - Magnetite Concentrate Summary Results

Samples	Size (µm)	Grade			Recovery		
		Fe (%)	Magnetite (%)	SiO_2 (%)	Fe (%)	Magnetite (%)	Weight (%)
JUIF-High	-106	60.2	74.8	12.9	66.6	96.4	34.8
	-75	63.5	81.5	9.2	68.8	97.2	35.7
	-45	67.7	88.2	4.8	64.9	96.8	30.9
	-25	68.5	90.9	4.0	67.4	97.4	33.2
JUIF-Low	-106	57.2	61.0	15.3	55.7	97.3	15.6
	-75	60.6	67.7	12.1	57.9	97.2	13.7
	-45	65.1	77.2	7.4	53.8	96.1	12.9
	-25	67.5	83.4	4.7	57.8	96.3	14.1
LRC	-106	61.6	72.0	11.7	64.3	97.2	28.2
	-75	64.9	79.2	8.0	65.9	97.6	25.5
	-45	67.9	86.2	4.8	62.0	97.7	26.2
	-25	69.6	90.7	3.1	64.2	97.4	23.6
PGC	-106	59.8	63.6	13.5	59.6	96.6	25.4
	-75	64.9	75.7	8.2	61.1	97.0	23.3
	-45	67.8	85.1	4.9	56.7	96.9	22.0
	-25	69.1	87.3	3.4	60.4	97.1	21.7
URC	-106	62.8	74.4	10.3	60.9	96.8	27.5
	-75	65.8	80.9	7.4	62.9	97.6	25.7
	-45	69.0	89.2	3.5	58.3	96.6	22.2
	-25	69.1	89.8	3.4	61.2	97.4	24.7
LRGC	-106	63.9	84.5	9.5	67.9	97.8	26.9
	-75	67.7	91.4	5.0	71.0	97.9	27.3
	-45	69.2	93.2	3.3	66.1	97.0	27.7
	-25	69.6	93.3	2.9	68.0	96.9	25.5
GC	-106	65.7	82.4	7.2	28.0	91.4	9.4
	-75	65.4	81.6	7.3	14.1	84.1	4.3
	-45	68.2	86.6	4.3	13.5	80.5	4.1
	-25	69.0	87.8	3.8	24.9	88.8	8.3

13.3.5 Magnetite Plant Benchscale Beneficiation Testwork.

Ore characterization testwork confirmed that the best concentration route for the Full Moon project is magnetite beneficiation and that a magnetite concentrate at 4.5 % SiO_2 should be obtained at 100 % 45 µm. The beneficiation test program conducted included the following:

- Dry coarse cobbing to investigate the optimum feed size for early gangue rejection and downstream grinding energy minimization;

- Low intensity magnetic separation on material ground at 100 % 45 µm to confirm concentrate grades and the achievable recovery;
- Flotation tests to establish the parameters for silica removal and determine the recovery to produce a low silica content final concentrate (<1.5 %);
- Preliminary beneficiation testing on non-magnetic material to establish the potential to increase the recovery in a hematite scavenging plant.

13.3.5.1 Cobber Magnetic Separation

A sample of each lithology was submitted for dry magnetic separation at 100 % passing -4.0,-2.8 and 2.0 mm. The results of the tests are presented in Table 13.10.

A part from JUIF-Low for which mass rejection is around 48 % for a magnetite recovery of 93-95 %, all the other lithology units have a mass rejection of 15-20 % for a magnetite recovery of 98-99 %. The high mass rejection obtained for the JUIF-Low sample is due to the fact that this sample has a lot less magnetite than the other samples (Table 13.8).

The target particle size for dry cobbing was 100 % passing 3 mm; this size was selected on the basis of benchmarking with existing operations and after discussions with HPGR vendors.

Results show that for all the samples, mass rejection barely varies over the range of sample feed sizes and that the previously selected dry cobbing particle size can be kept.

Table 13.10 – Cobber Dry Magnetic Separation Results - Concentrate Specifications

Lithology Unit	Size (mm)	Weight Recovery (%)	Fe Recovery (%)	Magnetite Recovery (%)	Fe Grade (%)	Magnetite Grade (%)	SiO_2 Grade (%)
JUIF-High	-4.0	83.8	92.7	98.8	34.1	32.4	42.4
	-2.8	83.5	92.1	98.7	34.1	32.3	43.9
	-2.0	82.7	91.9	98.9	33.9	32.3	43.5
JUIF-Low	-4.0	52.4	61.7	93.1	35.4	17.0	37.3
	-2.8	52.5	62.5	93.5	35.4	16.9	37.8
	-2.0	52.8	62.9	95.3	34.7	16.3	38.4
LRC	-4.0	81.0	85.8	97.9	30.9	23.3	47.1
	-2.8	79.6	85.3	97.7	31.5	23.3	45.9
	-2.0	78.9	85.1	97.8	31.9	24.1	45.4
PGC	-4.0	84.5	86.7	98.2	31.6	19.2	45.9
	-2.8	82.4	85.8	98.0	32.8	20.1	45.0
	-2.0	81.6	85.2	98.0	32.4	19.9	45.0
URC	-4.0	84.7	89.3	98.6	34.8	25.1	41.5
	-2.8	83.2	88.2	98.4	35.0	25.0	40.1
	-2.0	83.0	88.4	98.7	35.3	24.1	40.7
LRGC	-4.0	84.1	89.3	98.9	30.5	27.7	46.5
	-2.8	83.1	88.4	98.8	30.7	28.3	46.8
	-2.0	81.9	87.9	98.8	31.3	30.0	46.0

13.3.5.2 Ball Mill Work Index on Cobber Concentrates

BWI with a 270 mesh (53 µm) reference screen were conducted on cobber magnetic concentrates for regrinding sizing. Table 13.11 presents the results. The results indicate that lithology units are classified as hard ore types.

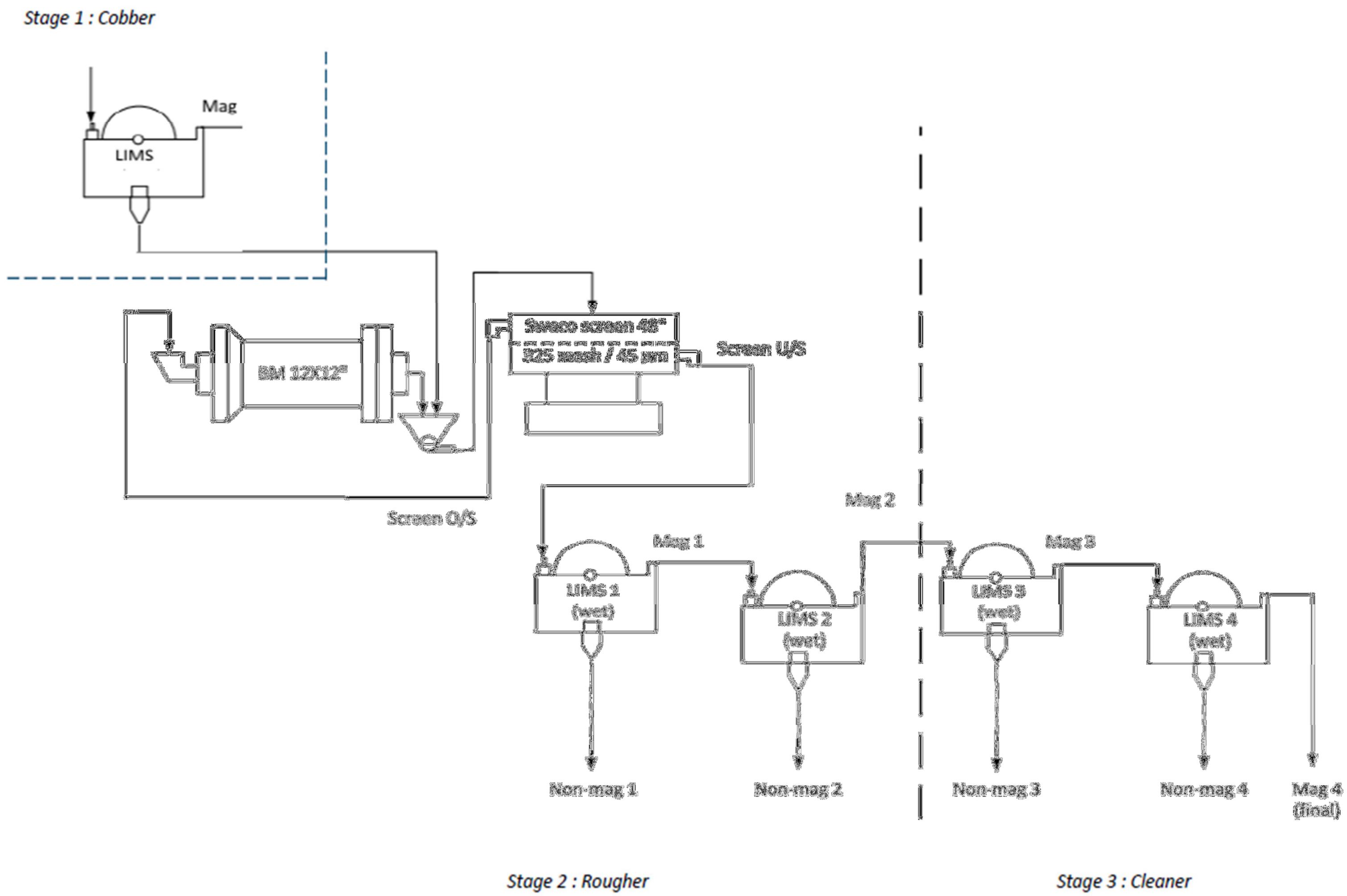
Table 13.11 – BWI on Cobber Concentrates

Lithology Unit	Feed Size (mm)	BWI (kWh/t)
JUIF-High	-2.8	17.3
JUIF-Low	-2.8	18.7
LRC	-4.0	15.4
	-2.8	15.3
	-2.0	14.7
PGC	-2.8	15.9
URC	-2.8	16.3
LRGC	-2.8	17.2

13.3.5.3 Low Intensity Magnetic Separation

To produce material for the next testwork steps (flotation and pelletizing), a semi-continuous mini pilot was used. The production was performed in three (3) separate stages. The first stage was the dry cobber stage performed on material ground at -2.8 mm. The second stage included regrinding at 95 % passing 45 µm and rougher magnetic separation. A cleaning of the rougher concentrate was realized in a third stage. Figure 13.3 presents the flowsheet of the circuit.

Figure 13.3 – Wet Low Intensity Magnetic Separation Circuit



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Aside from producing material for the subsequent tests, the other objective of this semi-pilot was to confirm the Davis Tube liberation testwork results in terms of concentrate grades and magnetite recovery. However, a significant amount of very fine material was lost during the filtration of non-magnetic products and the weight recoveries could not be confirmed.

Table 13.12 presents the results of the cobber stage and Table 13.13 presents the final magnetic concentrate characteristics. These results confirm the benchscale cobber results and the feasibility to reach the final 4.5 % SiO_2 concentrate grade with a regrinding at 95 % passing 45 µm and a magnetic beneficiation process.

Table 13.12 – Semi-pilot Cobber Concentrate

Samples	% Weight	Magnetite		Total Iron		SiO_2
		Grade (%)	Dist(%)	Grade (%)	Dist(%)	Grade (%)
JUIF-High	83.4	33.0	98.6	34.3	91.1	42.1
JUIF-Low	49.0	21.0	94.7	36.1	59.6	37.9
LRC	78.7	24.0	98.8	31.6	85.1	45.3
PGC	81.2	20.0	98.2	32.4	85.1	44.2
URC	81.6	26.0	98.6	35.5	87.7	40.0
Composite	77.5	26.0	98.7	33.3	84.8	43.4

Table 13.13 – Semi-pilot Final Magnetic Concentrate

Samples	Magnetite	Total Iron	SiO_2
	Grade (%)	Grade (%)	Grade (%)
JUIF-High	93.3	67.4	5.4
JUIF-Low	90.6	67.2	5.4
LRC	92.0	68.3	4.3
PGC	89.9	68.2	4.4
URC	93.5	68.4	4.4
Composite	95.0	68.7	4.3

13.3.5.4 Flotation

Flotation tests were conducted on magnetic concentrate to validate the feasibility to produce a low-silica content concentrate (<1.5 %). Reverse flotation tests were performed using an amine collector and starch depressant.

The tests performed were as follows:

- Rougher flotation tests on JUIF-High and composite samples to adjust the reagents dosage;
- Qualitative mineralogical analysis of composite rougher froth;
- Rougher flotation and cleaning flotation of rougher froth for all samples to confirm preliminary rougher results and increase iron recovery with the cleaning step.

Table 13.14 shows the summary results of rougher flotation tests. The target 1.5 % SiO_2 concentrate grade was reached for the composite sample at a 69 % iron recovery. It was not the case for the JUIF High sample which had a higher SiO_2 feed grade than normally expected. The following rougher/cleaner tests were performed using the composite flotation recipe.

Table 13.14 – Rougher Flotation - Sink Characteristics

Test #	Feed	Sink			
	SiO_2 (%)	% Weight	Total Iron		SiO_2 (%)
			Grade (%)	Recovery (%)	
JH - M - FL -03	6.2	87.5	69.4	90.7	3.4
JH - M - FL -04	6.2	94.3	67	96.3	4.3
JH - M - FL -05	6.2	78	69.7	81.7	2.4
JH - M - FL -06	6.2	69.4	70.6	73.8	1.7
JH - M - FL -07	6.2	72.7	69.1	75.8	1.7
Comp - M - FL - 01	4.3	66.6	71.3	69.1	1.3

A qualitative mineralogical analysis of composite rougher froth samples showed that gangue particles were generally coarser than iron oxide particles and mixed but only with a small proportion of iron oxides.

Table 13.15 presents the results of the Rougher/Cleaner tests. Tests performed on Composite, LRC, PGC and URC led to a final combined concentrate within the target specifications. The target 1.5 % SiO_2 concentrate was not reached for JUIF-High and JUIF-Low samples. As for the rougher tests, this is due to the feed SiO_2 grade being higher than 4.5 %.

Cleaning the rougher froth led to a 7-15 % increase in iron recovery. The combined concentrate iron recovery is still low, between 71 % and 87 % for the LRC, PGC, JUIF-High and Composite samples and between 49 and 58 % for the URC and JUIF-Low samples. No mineralogical analysis of the concentrate and tails was realized at the time of the testwork but, given the mineralogical analyses of the composite rougher froth, it is suspected that regrinding the rougher tails will be necessary to maximize the iron recovery.

Figure 13.4 – Schematic Representation of Rougher-Cleaner Tests

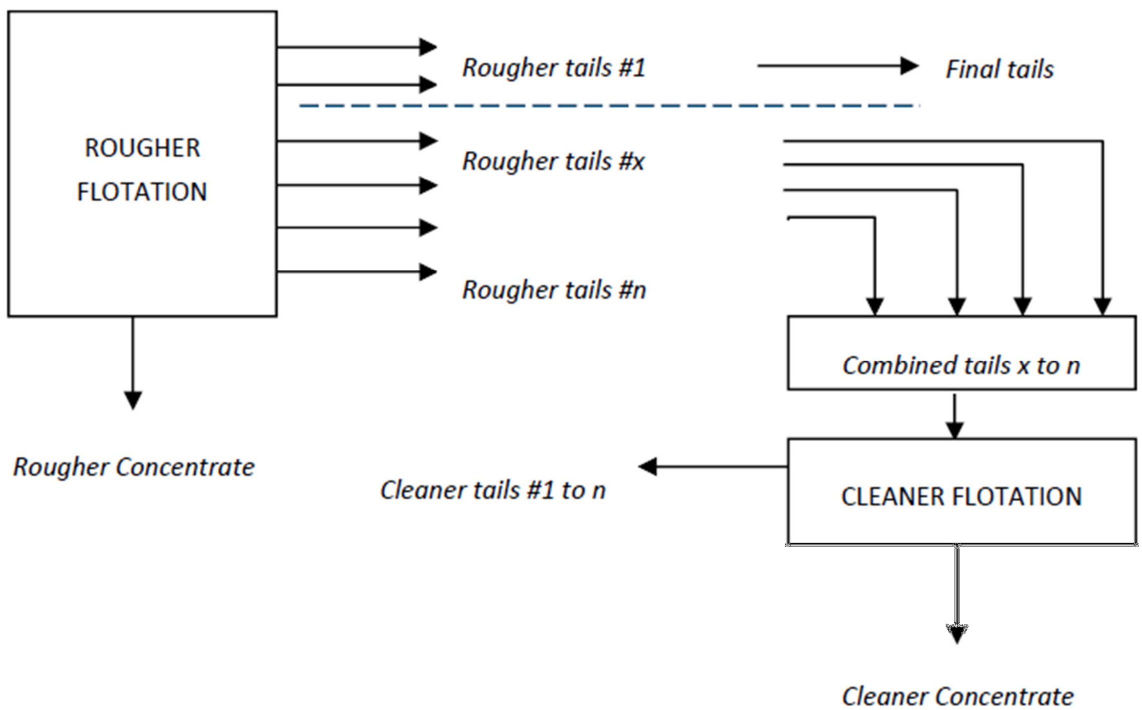


Table 13.15 – Rougher-Cleaner Flotation - Sink Characteristics

Test #	Feed	Rougher Concentrate				Froth Cleaner Concentrate				Combined Concentrate			
			Total Iron				Total Iron				Total Iron		
	SiO_2 (%)	% Weight	Grade (%)	Rec. (%)	SiO_2 (%)	% Weight	Grade (%)	Rec. (%)	SiO_2 (%)	% Weight	Grade (%)	Rec. (%)	SiO_2 (%)
JH-M-FL-08	6.2	72.6	70.6	76.9	1.9	10.5	67.3	10.6	4.5	83.1	70.2	87.5	2.2
JH-M-FL-09	6.2	73.3	69.9	77.5	2	6.8	67.7	6.9	3.9	80.1	69.7	84.4	2.2
Comp-M-FL-02	4.3	63.9	71.3	66.5	1.3	12.1	69.9	12.3	2.7	76.0	71.1	78.8	1.5
JL-M-FL-01	5.4	47.5	70.6	49.8	2.1	8.1	68.8	8.2	3.5	55.6	70.3	58	2.3
LRC-M-FL-01	4.3	62.4	69.1	64	1.2	7	70.6	7.4	2	69.4	69.3	71.4	1.3
PGC-M-FL-01	4.4	64.2	69.8	66.2	1.2	7.6	70.6	8.0	1.2	71.8	69.9	74.1	1.2
URC-M-FL-01	4.4	33.8	70.6	34.9	1.2	13.9	71.3	14.5	1.5	47.7	70.8	49.4	1.3

13.3.6 Hematite Plant Beneficiation Testwork

Beneficiation testwork was conducted on the non-magnetic products from the semi-pilot to evaluate the potential iron recovery of a hematite scavenging plant. It includes the following tests: dense media separation, high intensity magnetic separation, selective flocculation and flotation.

13.3.6.1 Dense Media Separation Test – Cobber Tails

DMS tests were performed on Cobber tails of JUIF-high and low -2.8 mm samples for each of the particle size fractions. The objective was to evaluate whether the non-magnetic ore was amenable to gravity concentration. The dense media used for the tests had a 3.3 density. No final concentrate was obtained with these tests and iron recovery was very low.

13.3.6.2 Wet High Intensity Magnetic Separation

WHIMS tests were performed on the rougher and cleaner non-magnetic products of the semi-pilot. Test conditions are shown in Figure 13.5.

Figure 13.5 – WHIMS Testing Conditions

Matrix	Steal ball ¼"
Wash water	adjusted at ≈3 psi
Sample weight	≈50 g
% solids	≈25%
Magnetic field	2000G, 4000G, 6000G and 8000G

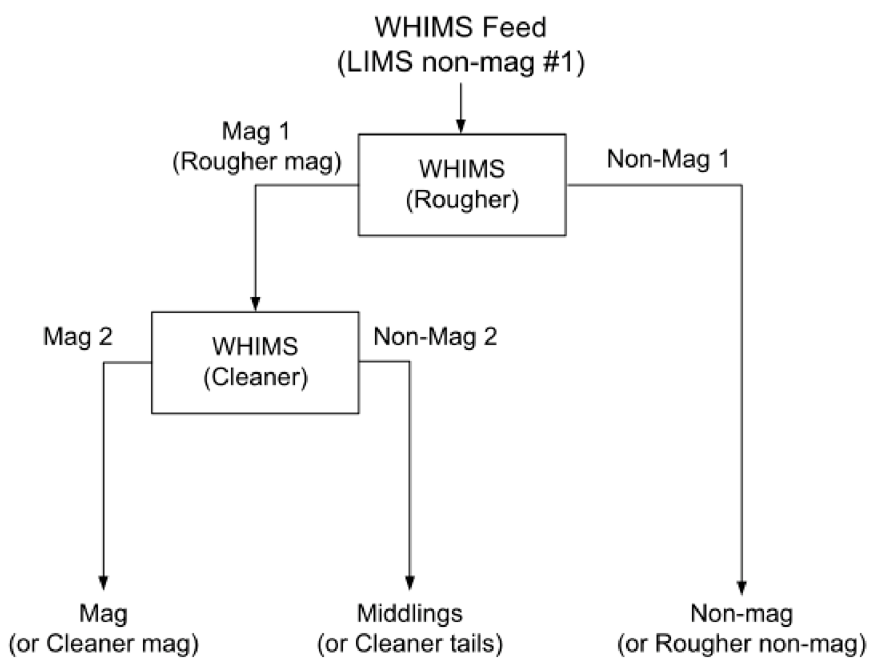


Table 13.16 presents the results of the tests conducted at 8000G. Iron recoveries of 76-89 % were obtained with a mass rejection of 43-62 %, showing that WHIMS could be used as a rougher to treat the non-magnetic tails.

Table 13.16 – WHIMS Magnetic Concentrate Charateristics - 8000G

Samples	% Weight	FeT (%)	FeT Recovery (%)
JUIF-High	53.7	38.0	83.0
JUIF-Low	61.7	41.5	80.9
LRC	49.5	41.0	88.6
PGC	43.1	50.7	81.9
URC	44.3	43.9	76.0

13.3.6.3 Selective Flocculation

High level selective flocculation testwork was conducted in order to assess the amenability of the magnetite circuit tailings (hematite ore) for the selective flocculation process.

A literature review and qualitative and quantitative tests were performed. The literature review allowed the selection of the reagents and the test conditions. Table 13.17 presents the testing conditions.

Table 13.17 – Reagent and Variable Selected

	Dispersion Step	Coagulation/Flocculation Step
Reagent	Sodium hexametaphosphate (SHMP)	Wheat Dextrin (WW82)
Concentration Ranges	50 to 150 mg/L	20 to 80 mg/L
Solids Percent	15 %	8 % to 15 %
Auqeous Matrix	Tap water and NaCl solutions	Tap water and NaCl solutions

The selective flocculation tests were performed on the composite wet LIMS non-magnetic #1-2 product of the semi-pilot. The composition of the sample is presented in Table 13.18.

Table 13.18 – Selective Flocculation Test – Feed Sample Composition

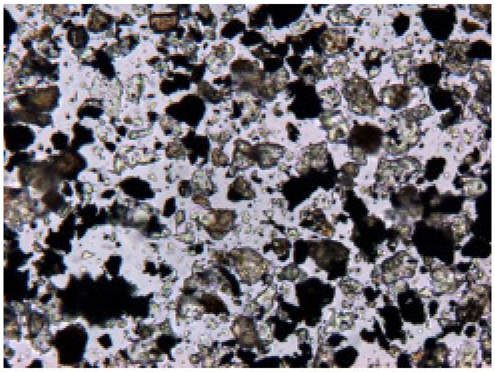
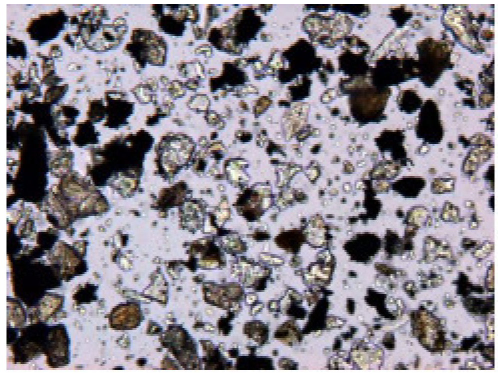
Product	Magnetite (%)	FeT (%)	SiO_2 (%)	Al_2O_3 (%)	Fe_2O_3 (%)	MgO (%)	CaO (%)	Na_2O (%)	K_2O (%)	TiO_2 (%)	MnO (%)	P_2O_5 (%)	Cr_2O_3 (%)	LOI (%)
Wet LIMS - Stage 1 non-magnetic #1-2	1	19.2	59.3	0.4	27.4	2.45	4.02	0.004	0.07	0.04	0.8	0.03	0.01	6.5

During the tests, the setting conditions were fixed at 3 and 10 minutes. Table 13.19 shows the sink product characteristics while Figure 13.6 shows pictures of the sink product.

Table 13.19 – Summary of the Selective Flocculation Test Results (Sink Product)

Settling Time (minute)	% Weight	FeT (%)	FeT Recovery (%)	SiO_2 (%)	MgO (%)	CaO (%)	LOI (%)
3	18.7	41	39.2	31.1	1.54	2.75	5.2
10	64.9	24.7	80.1	52.6	1.98	3.68	6.1
Head	100	19.2	100	59.3	2.45	4.02	6.5

Figure 13.6 – Picture of the Sink Pproduct (10 Minutes Settling Time)

The main conclusions of those preliminary tests can be stated as follows:

- The dispersant selected brings good results and did agree with the literature;
- The selectivity was achieved using the dispersant for a short settling time;
- A short settling time is critical to the selectivity of the separation process, the difference between 3 and 10 minutes settling time was significant, with grades of sinks of 40 % Fe and 24 % Fe;
- The microscopic observations show that, in the sink product, most of the iron oxide and impurities appeared to be free.

Even if the results did not give interesting enough results to select the process for the flowsheet definition, the selective flocculation process should not be discarded for the next project phases. Selective flocculation tests performed on similar ore show interesting results regarding the iron grade and recovery that can be achieved with this process. For the next project phase, it is suggested to continue selective flocculation benchscale tests by:

- Varying the pulp pH to assess the impact;
- Testing other reagents used as dispersant and flocculent;
- Increasing the desliming steps to allow maximum silica removal;
- Decreasing the settling time for each step in order to increase the underflow grade.

13.3.6.4 Flotation

Exploratory flotation tests were conducted on rougher and cleaner non-magnetic products of the semi-pilot. As per Section 13.3.5.4, reverse flotation was performed with the difference that a higher proportion of the weight was floated. The same equipment and reagents as for the magnetite concentrate flotation were used with the addition of another collector, a phosphoric acid for the flotation of carbonates.

Table 13.20 shows the summary of the flotation tests conducted. Although exploratory, these tests show that it would be difficult to reach a final concentrate grade by using flotation directly on non-magnetic products, even at the cost of a high iron recovery loss.

Table 13.20 – Non-Magnetic Flotation Results Summary

Test #	% Weight	Total Iron		SiO_2 (%)	MgO (%)	CaO (%)
		Grade (%)	Rec. (%)			
Feed	100	19.2	100	59.3	2.45	4.02
Comp - NM - FL - 01	27.4	45	61.7	21.5	2.68	3.22
Comp - NM - FL - 02	9.7	56.5	27.7	7.9	1.9	2.26
Comp - NM - FL - 03	16.9	49.8	41.3	22	1.99	1.01
Comp - NM - FL - 04	14	53.4	37.6	12.1	2.16	2.13
Comp - NM - FL - 05	5.7	54.3	15.6	10.3	2.03	2.41

Testwork to define a potential hematite plant was exploratory and not completed. Testwork was performed on non-magnetic products of the semi-pilot only. Cobber tails were not tested. However, the following conclusions could be reached:

- WHIMS seems promising as a rougher step;
- Further testwork on flotation is required and regrinding seems to be necessary. It is supposed at this stage that hematite has the same liberation size as magnetite. See MLA results in Section 13.3.4.2.

13.3.7 Pelletizing Testwork

The pelletizing tests were conducted at COREM and aimed to investigate the suitability of the ore for producing commercial grade pellets. The scope of these tests was preliminary and the composite sample was used.

13.3.7.1 Material Preparation and Characterization

The iron ore concentrate, mainly composed of magnetite and produced in the wet LIMS step from the composite sample, was homogenized and characterized. Size analyses, Blaine specific surface and Satmagan measurements were carried out. Major elements were analyzed by X-ray fluorescence ("XRF") and results are reported as oxides except for total iron which is reported in its elemental form. Differential thermal analyses, for orienting the pelletizing testwork, were performed at the Université de Sherbrooke.

13.3.7.2 Laboratory Balling Tests

Laboratory balling tests were carried out using a balling tire and a COREM standard procedure. The magnetite concentrate produced for the pelletizing testwork had a P80 of 38 µm (93.8 % -45 µm) and a Blaine of 1815 cm^2/g. These features are appropriate for pelletizing and the material was used as obtained. Green pellets with a good drop number and wet strength were obtained with a bentonite ratio of 7 to 9 kg/t of ore and a green pellet moisture of between 9.7 to 10.2 %.

13.3.7.3 Basket Test

COREM had conducted one (1) basket test with three (3) blast furnace pellet chemistries: two (2) acid pellets and one (1) fluxed pellet. They were charged in duplicate in the six-compartment basket. After basket firing, these duplicates were unloaded and blended together to generate enough samples to perform all the physical and metallurgical characterization. Table 13.21 presents the results for the three (3) selected blast furnace pellet chemistries.

Table 13.21 – Pellet Feed Properties and Fired Pellet Chemistry – Basket Tests

Parameter		Unit	BF Acid Pellets 0.21 CaO/SiO_2 - 0.24 % MgO	BF Acid Pellets 0.14 CaO/SiO_2 - 0.51 % MgO	BF Fluxed Pellets 1.0 CaO/SiO_2 - 1.0 % MgO
Pellet Feed Properties	Blaine	cm^2/g	1845	1845	1940
	-45 µm	%	93.2	93.2	90.8
	Satmagan	%	89.8	89.8	82.7
	Bentonite	%	0.78	0.78	0.72
	Limestone	%	1.39	0	5.95
	Dolomite	%	0	1.39	3.26
Fired Pellet Chemestries	SiO_2	%	4.7	4.7	4.5
	CaO	%	1.00	0.68	4.47
	MgO	%	0.26	0.52	1.01
	Fe	%	65.7	65.9	62.5
	FeO	%	< 0.1	< 0.1	0.2
	Al_2O_3	%	< 0.2	< 0.2	0.2

Table 13.22 presents the physical and metallurgical properties measured. It can be observed that all three (3) pellet samples showed good physical and metallurgical properties. The fluxed pellet sample had the lowest quality of all three (3) samples; however, it is likely that optimizing firing conditions specifically for fluxed pellets would improve its quality. It should be mentioned that fired pellets from basket tests have a better quality than commercial pellets and it is strongly recommended to validate these results with full pot grate tests.

Table 13.22 – Fired Pellet Physical and Metallurgical Properties of the Basket Test Samples

Parameter	Unit	BF Acid Pellets 0.21 CaO/SiO_2 - 0.24 % MgO	BF Acid Pellets 0.14 CaO/SiO_2 - 0.51 % MgO	BF Fluxed Pellets 1.0 CaO/SiO_2 - 1.0 % MgO
CSS	kg / pellet	465	408	340
Mini-tumble	% +6.3 mm	98.3	97.8	98.8
	% - 0.5 %	1.7	2.2	1.2
Satmagan		0.8	0.6	0.4
Porosity		21.0	19.6	18.3
R_{40}	% O_2 / min	0.66	0.72	1.04
Swelling	% vol.	13.3	14.3	15.1
	% red.	44.0	43.3	59.0
Dynamic LTD	% +6.3 mm	98.8	88.2	77.8
	% - 0.5 %	0.6	2.9	3.1

13.4 Process Flowsheet Development

The results from the above-mentioned testwork, as well as historical test data and adjacent properties' process information, were used to develop a preliminary process flowsheet for the Rainy Lake deposit. The following considerations were taken into account to develop the flowsheet:

- Historical test data showed that Rainy Lake ore was a magnetic ore, with most of the iron in the form of iron-oxides and a magnetite-hematite ratio of around 70:30;
- PEA testwork confirmed that that Rainy Lake ore was a magnetic ore;
- Heavy liquid separation tests and liberation analysis showed that gravity concentration was not a viable option;
- Recovery by magnetic separation requires a grind size of at least 100 % passing 45 µm to achieve a concentrate grade of below 4.5 % SiO_2;
- Flotation is required to produce a low silica content concentrate (1.5 % SiO_2) and regrinding is necessary to maximize iron recovery;
- The liberation size for the recovery of hematite has not been confirmed through testwork and is currently considered similar to the magnetite liberation size;
- Recovery of hematite through a combination of regrinding, WHIMS and flotation seems achievable.

The following is a presentation of the proposed flowsheet. Section 13.4.2.1 presents the weight recovery model for this flowsheet.

13.4.1 Proposed Flowsheet

13.4.1.1 Description

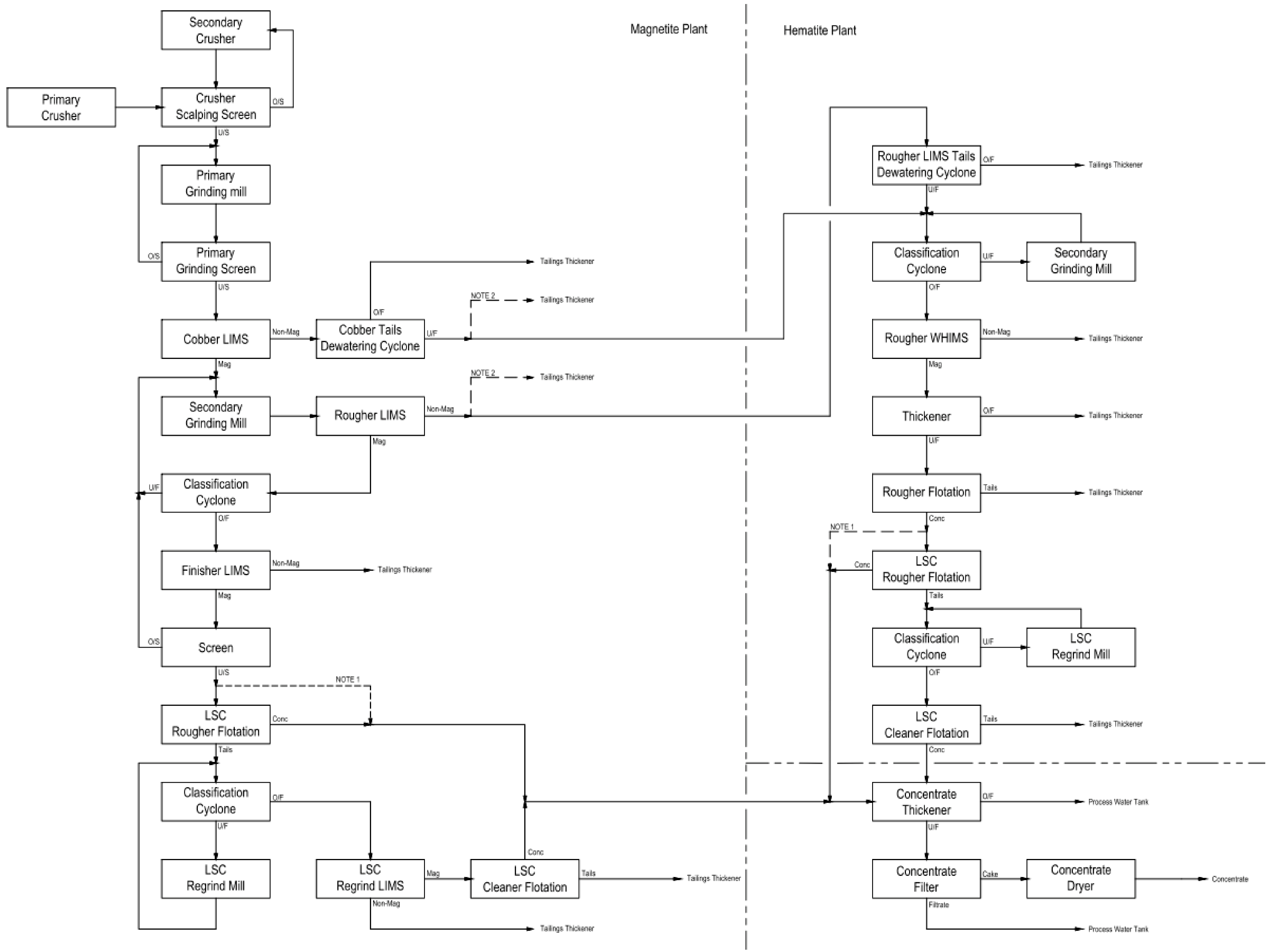
Figure 13.7 presents the selected flowsheet. Two (2) stages of crushing followed by a grinding stage via HPGR are required in order for the ROM to reach the optimum grain size for processing. The magnetite beneficiation process then consists of a magnetic separation circuit followed by a flotation circuit.

The magnetic separation circuit is a three (3) stage process whose purpose is to separate the magnetite from the non-magnetic material. Grinding is added after the cobber magnetic separation stage in order to increase particle liberation. The regrind product is fed to a rougher LIMS whose role is to immediately reject the non-magnetic particles that have been liberated through grinding before re-circulating them into the mill. This reduces the grinding energy requirements. The regrind product is then further processed in a finishing magnetic step followed by a final classification to achieve the targeted iron and high silica content grade (4.5 %) targets. To produce a magnetic LSC (1.5 %), the HSC undergoes further regrinding. The regrind product is fed to a magnetic separator to remove the liberated silica and the target silica grade concentrate is then achieved via a final flotation step.

The hematite plant is a scavenging plant that treats the magnetite plant cobber and the rougher LIMS tailings. The material is first reground in order to increase particle liberation. Hematite is then recovered in a WHIMS step and sent to a desliming thickener for dewatering and for slime particles removal. The target high silica grade concentrate is then achieved via a final flotation step. As for the magnetite plant, the HSC has to undergo further regrinding to produce a low silica grade concentrate. The reground product is sent to a final flotation step to produce the LSC.

Finally, the magnetite and hematite concentrates are combined, thickened, filtered and dried for transport and pellet production.

Figure 13.7 – Simplified Process Flowsheet



13.4.1.2 Discussion

The flowsheet presents the following key features:

- Building a hematite plant together with a magnetite plant will maximize the iron recovery from the Rainy Lake deposit;
- The magnetite and hematite plant are totally independent. This is an advantage as building the magnetite and the hematite plant in two (2) separate steps is considered;
- The numerous magnetic separation stages of this flowsheet allow rejection of the liberated non magnetic material early in each processing stage. This presents the advantage of minimizing the grinding energy consumption;
- No WHIMS have been included in the hematite plant regrind loop as the feed material is likely to be too coarse for this equipment;
- Water management is important in this flowsheet to balance the required pulp densities in various stages.

13.4.2 Weight Recovery Model

13.4.2.1 Weight Recovery Model

A weight recovery model was developed for the above proposed flowsheet using the geological Davis Tube results database and the metallurgical testwork results. This weight recovery model was used for the mine pit design and mine planning (Section 16). It had to be expressed as a function of the total iron since the block model does not provide the magnetite or the hematite content for each block nor does it provide the amount of recoverable iron.

The Weight Recovery ("WR") is expressed as follows:

Total Weight Recovery = Magnetite Plant Weight Recovery + Hematite Plant Weight Recovery

Where:

- The magnetite plant weight recovery is obtained from the geological Davis Tube results (Section 11);
- The hematite plant weight recovery is obtained from the hematite plant iron recovery. This latter is calculated from the MLA results that give the iron distribution within the different minerals (Section 13.3.4.2).

Table 13.23 presents the total weight recovery correlation obtained for each lithology.

Table 13.23 – Total Weight Recovery Models per Lithology

Samples	Correlation	R^2
JUIF	Total WR = 1.0411 x Feed Fe_Tot + 3.3655	R^2 = 0.4710
LC	Total WR = 1.5992 x Feed Fe_Tot - 12.7290	R^2 = 0.9007
LRC	Total WR = 1.4233 x Feed Fe_Tot - 0.9894	R^2 = 0.9504
LRGC	Total WR = 1.7700 x Feed Fe_Tot - 23.9900	R^2 = 0.6457
PGC	Total WR = 1.3293 x Feed Fe_Tot + 0.8395	R^2 = 0.9351
URC	Total WR = 0.9113 x Feed Fe_Tot + 8.4630	R^2 = 0.5568
GC	Total WR = 1.3285 x Feed Fe_Tot - 16.0600	R^2 = 0.4885

A correlation between the total iron feed grade and the total weight recovery was then developed for each lithology. This correlation is corrected to consider the following aspects:

- The production of a concentrate at an average of 4.5 % SiO_2;
- The Davis Tube tests represent the perfect magnetic separation. A production circuit will be less efficient.

13.4.2.2 Process Plant Feed Design Criteria

Since the block model does not provide the magnetite or the hematite content for each block but only the total iron feed grade, the geological DT results database was processed to select the plant magnetite feed characteristics:

- Filtering using a cut-off DTWR of 18 %;
- Filtering using a cut-off concentrate SiO_2 of 8 % in order to obtain an average concentrate SiO_2 of 4.5 %.

Table 13.24 presents the resulting average feed composition. This composition corresponds to an average DTWR of 27.1 % and a hematite plant weight recovery of 10.2 % for a total weigh recovery of 37.3 %. These recoveries are ideal and do not include losses that could occur in the process during dewatering and thickening steps.

Table 13.24 – Average Feed Composition Based on Filtered Davis Tube Results

Filtering Criteria	Head Total Fe Grade (%)	Head Total SiO_2 Grade (%)	Head Magnetite Grade (%)
Cut-off 18 % DTWR & DT SiO_2 <8 %	31.3	44.5	27

13.5 Recommended Testwork for the Next Engineering Phases

Additional testwork will be required to bring the project to the Pre-Feasibility and Feasibility Studies level of detail. The major testwork planned to be performed is shown in Table 13.25.

Table 13.25 – Major Testwork Planned for the Pre-Feasibility and Feasibility Studies

Phase	Testwork
	Benchscale Testwork - Process Development
Pre-Feasibility	LSC circuit confirmation
	MLA analysis of HSC magnetic
	Flotation tests to confirm the flowsheet
	With MLA analysis of products and tails
	With regrinding of flotation tails
	Lock-cycle flotation tests on the confirmed flowsheet
	Hematite recovery testwork
	MLA analysis of non-magnetic streams
	Confirmation of regrinding size
	WHIMS testing
	Selective flocculation
	Flotation tests
	Benchscale Testwork - Process Variability
Feasibility	Grindability tests
	CW_i,BW_i,RW_i,SMC
	LABWAL & ATWAL HPGR tests
	Jar-mill tests (Tower Mill)
	Beneficiation confirmation testwork
	Magnetite recovery
	Hematite recovery
	Benchscale Pelletizing Testwork
Pre-Feasibility/ Feasibility	On combined HSC magnetite and hematite
	Blaine evaluation
	Ball tire agglomeration
	Green ball characterization
	Pot grate furnace tests
	On combined LSC magnetite and hematite
	Blaine evaluation
	Ball tire agglomeration
	Green ball characterization
	Pot grate furnace tests

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Phase	Testwork
	Pilot Scale Testwork
Feasibility	Flowsheet sections to pilot
	Grinding circuit
	Magnetic plant for HSC production
	Magnetic LSC flowsheet
	Hematite plant for HSC production
	Hematite LSC flowsheet
	Production of material for
	Pelletizing testwork
	Concentrate thickening, filtering & drying tests
	Tailings thickening & drying tests
	Equipment Testing
Feasibility	HPGR tests
	Tower mill tests
	Cyclone simulation
	Stack sizer tests
	Thickening tests
	Filtering tests
	Drying tests
	Pot grate furnace tests

14 Mineral Resource Estimate

14.1 Introduction

The Mineral Resource Statement presented herein represents the maiden mineral resource evaluation prepared for the Rainy Lake property in accordance with the Canadian Securities Administrators' National Instrument 43-101.

The mineral resource estimation process was a collaborative effort between SRK and WCSLIM staff. WCSLIM provided to SRK an exploration database and a geological interpretation comprising a series of vertical cross-sections through the areas investigated by core drilling and surface mapping. The geology model was constructed by Mr. Dominic Chartier, P.Geo. (OGQ#874, PEGNL#06306). The geostatistical analysis, variography, selection of resource estimation parameters, construction of the block model, and the conceptual pit optimization work were completed by Mr. Filipe Schmitz Beretta under the supervision of Mr. Mark Campodonic, MAusIMM (CP#225925), both employees of SRK Consulting (UK) Ltd. The project was conducted under the overall supervision of Dr. Jean-Francois Couture, P.Geo. (OGQ#1106, APGO#0197). The site visit was completed by Mr. Chartier and Dr. Couture. By virtue of their education, work experience that is relevant to the style of mineralization and deposit type under consideration and to the activity undertaken, and membership to a recognized professional organization, Mr. Campodonic, Mr. Chartier, and Dr. Couture are Qualified Persons pursuant to National Instrument 43-101 and independent from WCSLIM. The effective date of the Mineral Resource Statement is October 22, 2012.

This section describes the resource estimation methodology and summarizes the key assumptions considered by SRK. In the opinion of SRK, the resource evaluation reported herein is a reasonable representation of the iron mineralization found in the Full Moon iron deposit at the current level of sampling. The mineral resource has been estimated in conformity with generally accepted CIM Estimation of Mineral Resource and Mineral Reserves Best Practices Guidelines and is reported in accordance with the Canadian Securities Administrators' National Instrument 43-101. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve upon application of modifying factors.

The database used to estimate the mineral resource was audited by SRK. SRK is of the opinion that the current drilling and sampling information is sufficiently reliable to interpret with confidence the boundaries of iron mineralization, and that the assay data are sufficiently reliable to support mineral resource estimation.

14.2 Resource Estimation Procedures

The resource estimation methodology involved the following procedures:

- Database compilation and verification;
- Construction of wireframe models for the boundaries of the Sokoman Formation;
- Definition of resource domains;
- Data conditioning (compositing and capping) for statistical analysis, geostatistical analysis, and variography;
- Block modelling and grade interpolation;
- Resource classification and validation;
- Assessment of "reasonable prospects for economic extraction" and selection of appropriate reporting cut-off grades; and
- Preparation of the Mineral Resource Statement.

14.3 Mineral Resource Database

The resource database available for geology and mineral resource modelling comprises core borehole information acquired by WCSLIM in 2011 and 2012. The borehole database considered for mineral resource modelling comprises 124 core boreholes (22 853 meters) for which complete assay results were available. The boreholes are distributed on section lines generally spaced at 500 meters and borehole spacing along each section line of 400 meters. The assay database comprises 3 633 sample intervals from 121 boreholes assayed for the common major oxide elements.

The borehole data were received as an electronic database and the sectional interpretation of the main geological units and faults were received as polylines. SRK also received a digital topographic surface created from a digital elevation model constructed from LiDAR survey completed in 2012. Upon receipt of the project data SRK performed the following validation steps:

- Check of collar locations against topography. The topography of the area is generally flat;
- Check of minimum and maximum values for each table value field; and
- Check for gaps, overlaps, and out of sequence intervals for assay and lithology tables.

Mr. Chartier visited the Rainy Lake property on October 12 and 13, 2011 to inspect the property, discuss and review the exploration work undertaken by WCSLIM. Mr. Chartier and Dr. Couture returned to the Rainy Lake property on May 15 to 17, 2012 to review drilling procedures and discuss geological modelling. SRK is satisfied that the exploration work carried out by WCSLIM was conducted in a manner consistent with industry best practices and that the exploration data and the drilling database are sufficiently reliable for the purpose of supporting a mineral resource estimate.

14.4 Geological Interpretation and Modelling

The Full Moon iron deposit is a very large taconite iron deposit hosted in banded iron formations of the Proterozoic Sokoman Formation. The iron mineralization is strata-bound and sedimentary in origin and its original geometry was modified by folding and thrust faulting.

Based on a sectional geological interpretation of the core drilling data and surface geology mapping provided by WCSLIM, SRK created a three-dimensional model for the main stratigraphic rock units of the Sokoman Formation using the Leapfrog software. The three-dimensional model honours drilling data. The resulting geological/mineralisation model incorporates eight members of the Sokoman Formation, namely: LC, JUIF, GC, URC, PGC, LRC, LRGC, and LIF (Section 7.2.1). The bottom of the overlying MSS and the top of the underlying WSS were also modelled. Domains were created in Leapfrog software by interpolating a surface boundary from polylines set on several vertical sections spaced at 500 meters.

Each lithological unit exhibits different iron content and variable magnetite and hematite proportions. For this reason each lithological unit was considered as a separate domain for resource modelling and grade estimation. Figure 14.1 shows a plan view of the mineralized domains in relation to the boreholes. Figure 14.2 shows an oblique three-dimensional view of the mineralized domains.

Figure 14.1 – Plan of the Full Moon Iron Deposit and Distribution of Drilling Information Available for Resource Modelling

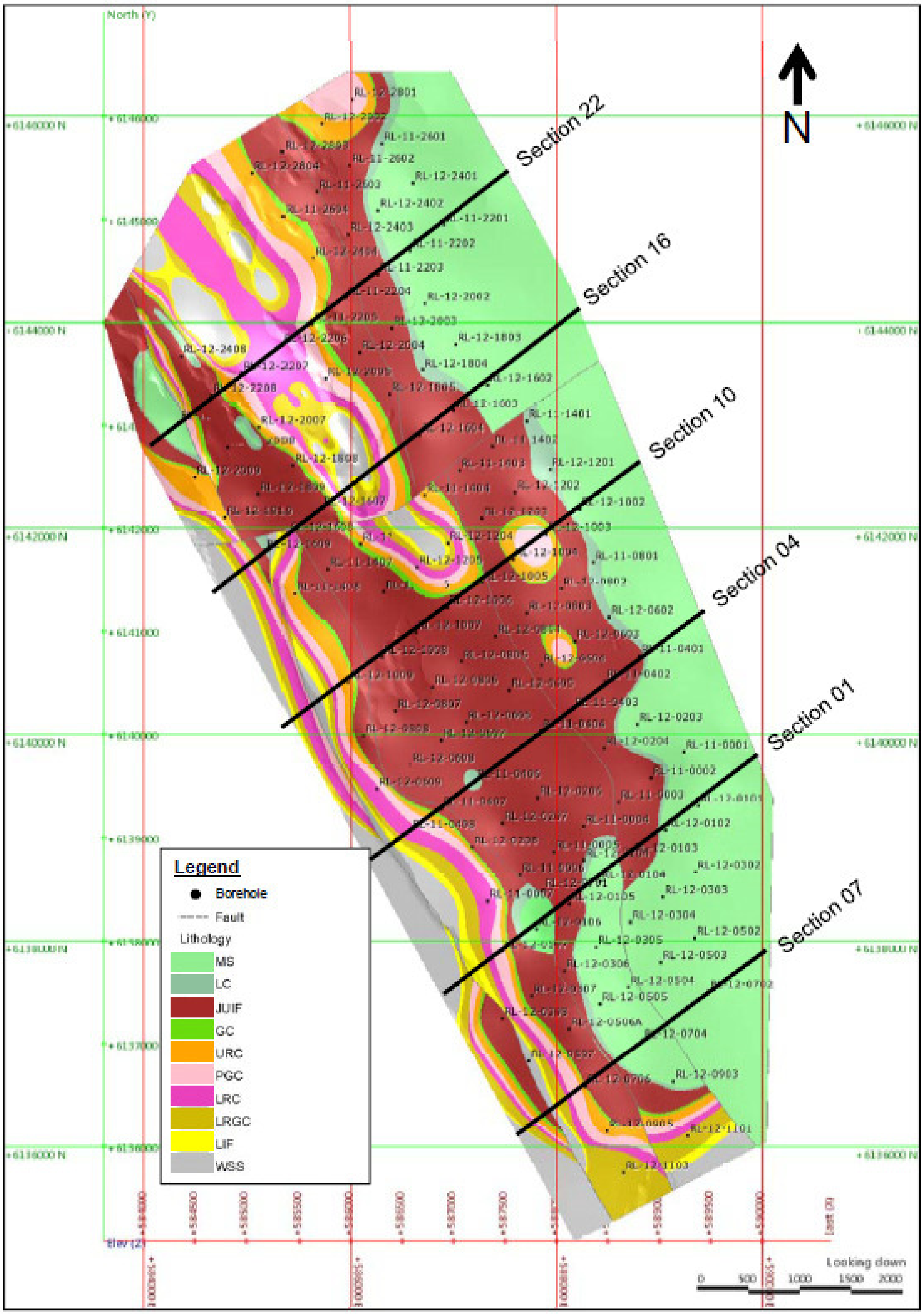
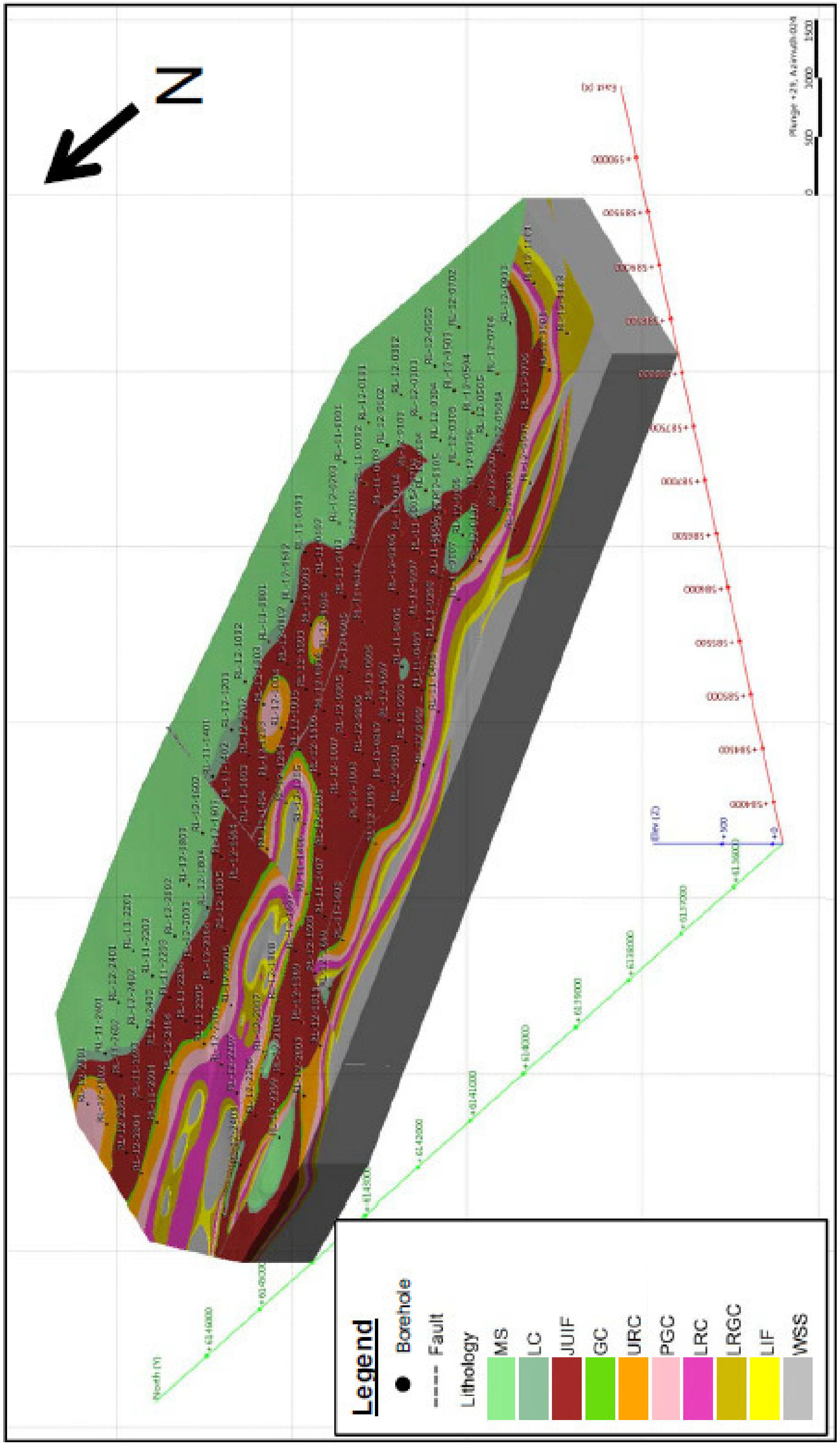


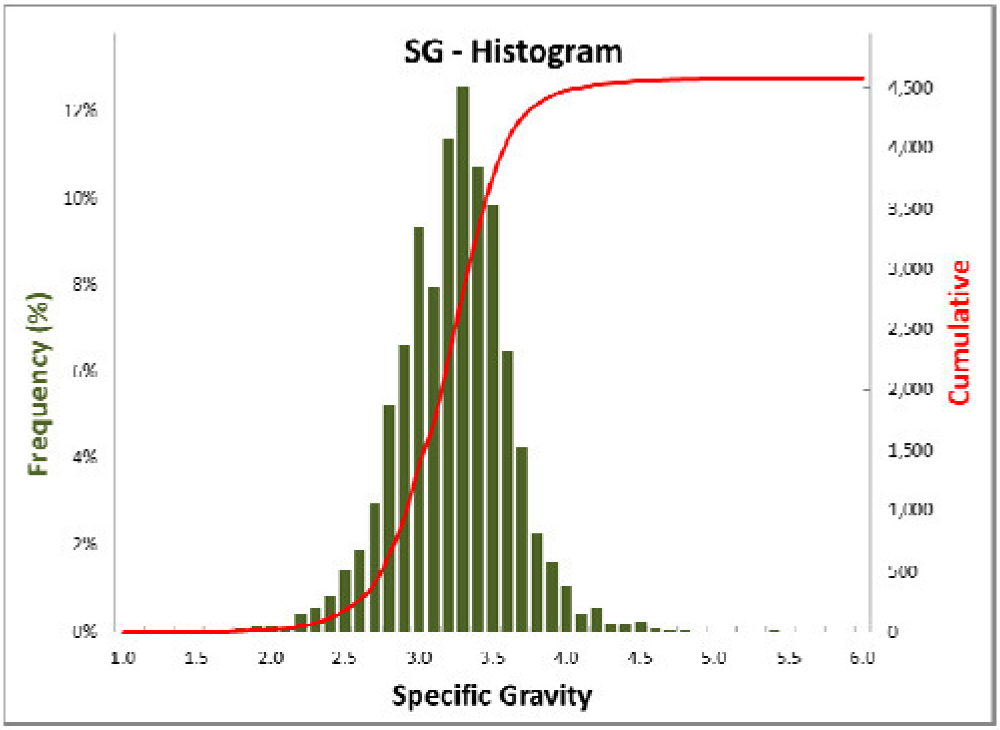
Figure 14.2 – Oblique View Looking Northeast of the Full Moon Iron Deposit Showing the Sokoman Formation Domains Considered for Resource Estimation



14.5 Specific Gravity

Specific gravity was measured by WCSLIM using a standard weight in air/weight in water methodology on representative core samples (Section 11.2). A total of 4 614 specific gravity measurements were taken for all lithological units (Figure 14.3).

Figure 14.3 – Frequency and Cumulative Histogram of Specific Gravity Data

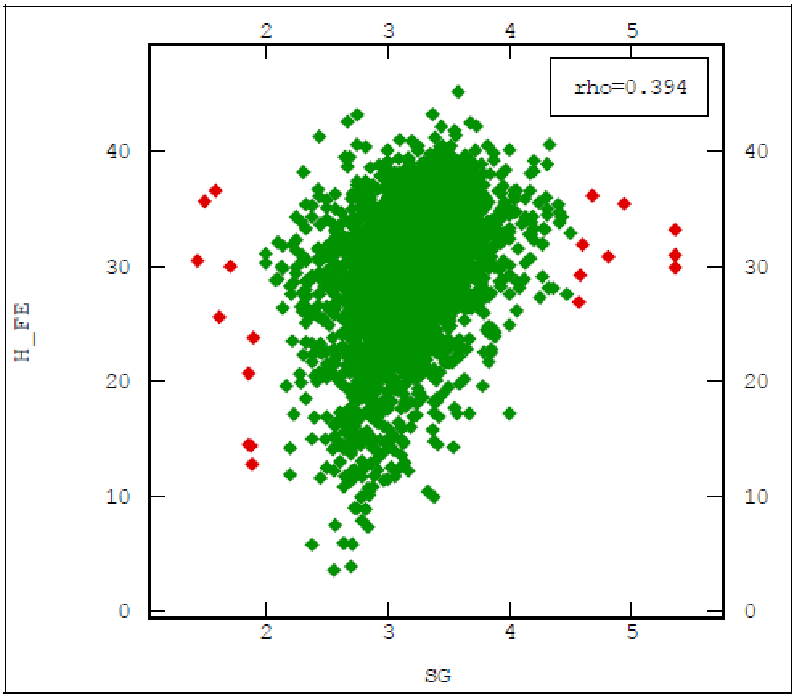


The raw specific gravity database shows outliers, which would affect the estimation process as shown in Figure 14.4. For this reason, specific gravity values were restricted to only those values between 2.0 and below 4.5.

Table 14.1 shows the basic statistics for specific gravity in the 5-meter composites for each domain.

Density was estimated in the block model using ordinary kriging, as there is no obvious relationship between oxide analysis results and specific gravity.

Figure 14.4 – Scatterplot between Iron (%) and Specific Gravity Composites



(Outliers marked in red)

Table 14.1 – Specific Gravity Composites Statistics by Domain

Domain	Count	Minimum	Maximum	Mean	Std. Dev.	Variance
JUIF	1,328	2	4.5	3.2	0.35	0.12
URC	395	2	4.5	3.3	0.33	0.11
PGC	671	2	4.5	3.3	0.32	0.10
LRC	643	2	4.5	3.2	0.32	0.10
LRGC	427	2	4.5	3.1	0.29	0.09
Total	**3,464**	**2**	**4.5**	**3.2**	**0.33**	**0.11**

14.6 Compositing and Statistics

The statistical study was completed for five Sokoman Formation domains modelled. The domains are subdivided into subdomains by faults (Figure 14.1). Raw statistics were examined for each subdomain to ensure they could be grouped in domains for statistic and geostatistical studies. The major oxide variables studied were iron (%), SiO_2 (%), Al_2O_3 (%), P_2O_5 (%), MnO (%) and loss on ignition ("LOI") (%). Table 14.2 shows the basic statistics for the geological domains. The highlighted domains are those considered as mineralized and object of this study. Data compositing was undertaken to reduce the inherent grade variability that exists within the domained populations and to generate samples more appropriate to the scale of the mining operation envisaged. It was also necessary for the estimation process, as all samples were assumed to be of equal weighting, and should, therefore, be of equal length. A review of outliers suggests that capping is not necessary.

Table 14.2 – Raw Un-composited Sample Statistics by Domain

Variable	Domain	Count	Min.	Max	Mean	Std. Dev.	Variance
Fe (%)	MS	0	0.00	0.00	0.00	0.00	0.00
	LC	24	5.83	27.84	16.37	4.66	21.73
	JUIF	1,292	9.00	43.23	29.44	4.72	22.28
	GC	110	7.34	26.23	17.23	3.98	15.81
	URC	379	12.80	41.00	33.49	4.19	17.58
	PGC	705	19.40	42.50	31.14	3.73	13.91
	LRC	627	19.80	45.18	30.57	3.16	9.96
	LRGC	410	12.30	42.18	27.22	4.83	23.30
	LIF	79	3.59	29.38	17.44	4.82	23.25
	WSS	8	3.91	23.20	14.73	6.29	39.61
SiO_2 (%)	MS	0	0.00	0.00	0.00	0.00	0.00
	LC	24	34.60	65.10	47.28	7.25	52.60
	JUIF	1,292	23.23	74.80	45.08	5.52	30.43
	GC	110	20.90	72.40	50.09	6.92	47.87
	URC	379	31.92	66.40	40.52	4.22	17.82
	PGC	705	28.10	57.99	43.56	4.27	18.27
	LRC	627	23.10	59.40	45.66	3.64	13.25
	LRGC	410	37.13	60.90	47.33	4.38	19.15
	LIF	79	30.40	78.13	50.72	8.47	71.73
	WSS	8	42.30	87.00	57.84	13.82	191.04
Al_2O_3 (%)	MS	0	0.00	0.00	0.00	0.00	0.00
	LC	24	0.27	14.80	2.42	4.20	17.68
	JUIF	1,244	0.01	9.91	0.51	0.70	0.50
	GC	109	0.02	10.00	0.97	1.29	1.66
	URC	365	0.01	8.51	0.15	0.44	0.19
	PGC	682	0.01	1.79	0.13	0.14	0.02
	LRC	612	0.01	2.13	0.14	0.11	0.01
	LRGC	392	0.01	12.70	0.19	0.84	0.71
	LIF	76	0.01	13.80	2.82	3.78	14.28
	WSS	7	0.04	14.30	4.51	5.26	27.70

Variable	Domain	Count	Min.	Max	Mean	Std. Dev.	Variance
P_2O_5 (%)	MS	0	0.00	0.00	0.00	0.00	0.00
	LC	24	0.03	0.25	0.07	0.05	0.00
	JUIF	1,260	0.01	0.56	0.03	0.03	0.00
	GC	97	0.01	0.93	0.04	0.08	0.01
	URC	366	0.01	0.11	0.02	0.01	0.00
	PGC	686	0.01	0.09	0.02	0.01	0.00
	LRC	619	0.01	0.06	0.02	0.01	0.00
	LRGC	390	0.01	0.07	0.02	0.01	0.00
	LIF	69	0.01	0.13	0.04	0.02	0.00
	WSS	5	0.01	0.06	0.04	0.02	0.00
MnO (%)	MS	0	0.00	0.00	0.00	0.00	0.00
	LC	24	0.04	0.85	0.49	0.18	0.03
	JUIF	1,292	0.12	6.89	0.92	0.61	0.37
	GC	110	0.43	4.28	1.71	0.72	0.51
	URC	379	0.14	3.79	0.91	0.52	0.27
	PGC	705	0.07	1.95	0.60	0.28	0.08
	LRC	627	0.09	5.08	0.54	0.42	0.17
	LRGC	410	0.14	1.82	0.66	0.25	0.06
	LIF	79	0.04	2.08	1.01	0.51	0.26
	WSS	8	0.07	2.54	0.77	0.77	0.59
LOI (%)	MS	0	0.00	0.00	0.00	0.00	0.00
	LC	24	4.70	24.30	16.09	5.31	28.23
	JUIF	1,292	-0.09	23.66	5.78	3.56	12.70
	GC	110	4.37	61.12	14.14	5.48	30.07
	URC	379	0.54	19.70	5.17	2.55	6.49
	PGC	705	-0.17	15.69	4.96	2.22	4.95
	LRC	627	0.00	16.58	3.96	1.78	3.15
	LRGC	410	0.04	19.40	6.49	3.23	10.43
	LIF	79	1.28	25.00	12.96	5.11	26.13
	WSS	8	2.15	18.20	9.94	5.81	33.80

Figure 14.5 shows a histogram of sample length from the mineralized domains. As shown, more than 98 percent of the samples are less than or equal to 5 meters and as such a 5-meter composite length was chosen. Table 14.3 illustrates the statistical comparison between the raw data and the 5-meter composites for the main elements with both the raw and 5-meter composites averaging around 30 percent iron.

The estimation process assumes an equivalent weighting per composite. Discarding the remnant composites has little impact on the statistical mean of the sample data, and thus all composites were used for variography and grade interpolation.

Figure 14.5 – Histogram for Mineralized Sample Length

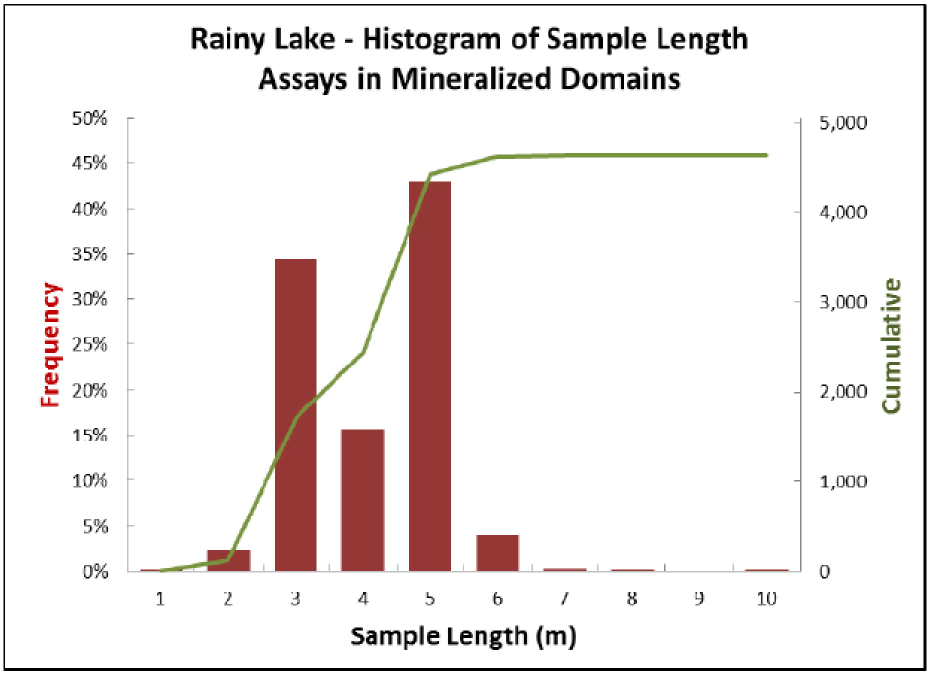


Table 14.3 – Specific Gravity Composites Statistics by Domain

Domain	Count	Min.	Max.	Mean	Std. Dev.	Variance
Raw Data (weighted by sample length)						
Fe (%)	3,635	3.59	45.18	29.40	5.49	30.14
SiO_2 (%)	3,635	20.90	87.00	44.98	5.42	29.42
Al_2O_3 (%)	3,512	0.01	14.80	0.38	1.02	1.03
P_2O_5 (%)	3,517	0.01	0.93	0.03	0.03	0.0009
MnO (%)	3,635	0.04	6.89	0.78	0.53	0.29
LOI (%)	3,635	-0.17	61.12	5.80	3.78	14.30
5-meter Composites						
Fe (%)	2,558	9.00	45.18	30.17	4.13	17.08
SiO_2 (%)	2,558	23.10	69.14	44.64	4.32	18.70
Al_2O_3 (%)	2,541	0.01	11.20	0.28	0.52	0.27
P_2O_5 (%)	2,524	0.01	0.38	0.03	0.02	0.0004
MnO (%)	2,558	0.14	4.66	0.75	0.45	0.21
LOI (%)	2,558	0.16	21.00	5.30	2.79	7.78

The iron (% Fe) composite histogram (Figure 14.6) shows a slight negative skew in the distribution owing a small amount of internal dilution included in the modelled domains. It is also evident that there are more than one population in the data set with means at 25 and 31 percent iron.

Figure 14.6 – Histogram for Mineralized Sample Length

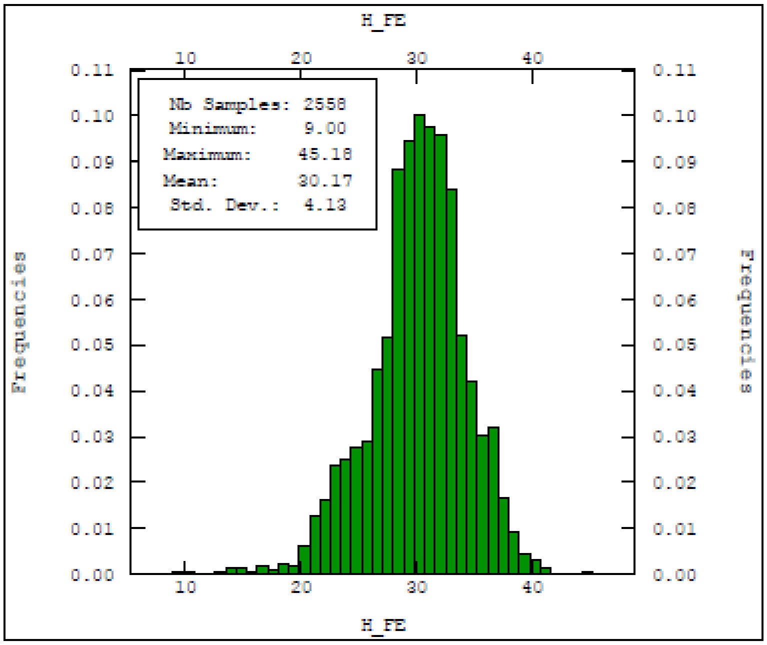


Figure 14.7 shows the iron histogram plots for each domain: JUIF, URC, PGC, LRC, and LRGC. Domain composite histograms indicate a near normal distribution with an average grade of between 30 and 35 percent iron and with a slight negative skew due to inclusion of a small amount of low grade material inside the modelled domains.

Table 14.4 shows the composite statistics by domain. As shown, iron is highest in the URC domain, averaging 33.5 percent iron and lowest in the LRGC domain, averaging 27.4 percent iron. Scatterplots show a linear relationship between iron and SiO_2 with a decrease in SiO_2 as iron content increases.

The histogram for LRGC clearly suggests more than one population (Figure 14.7), but due to the limited number of samples (315) the populations could not be subdivided and the domain was considered as a single domain.

Figure 14.7 – Iron Composite Histogram for Domains JUIF, URC, PGC, LRC, and LRGC

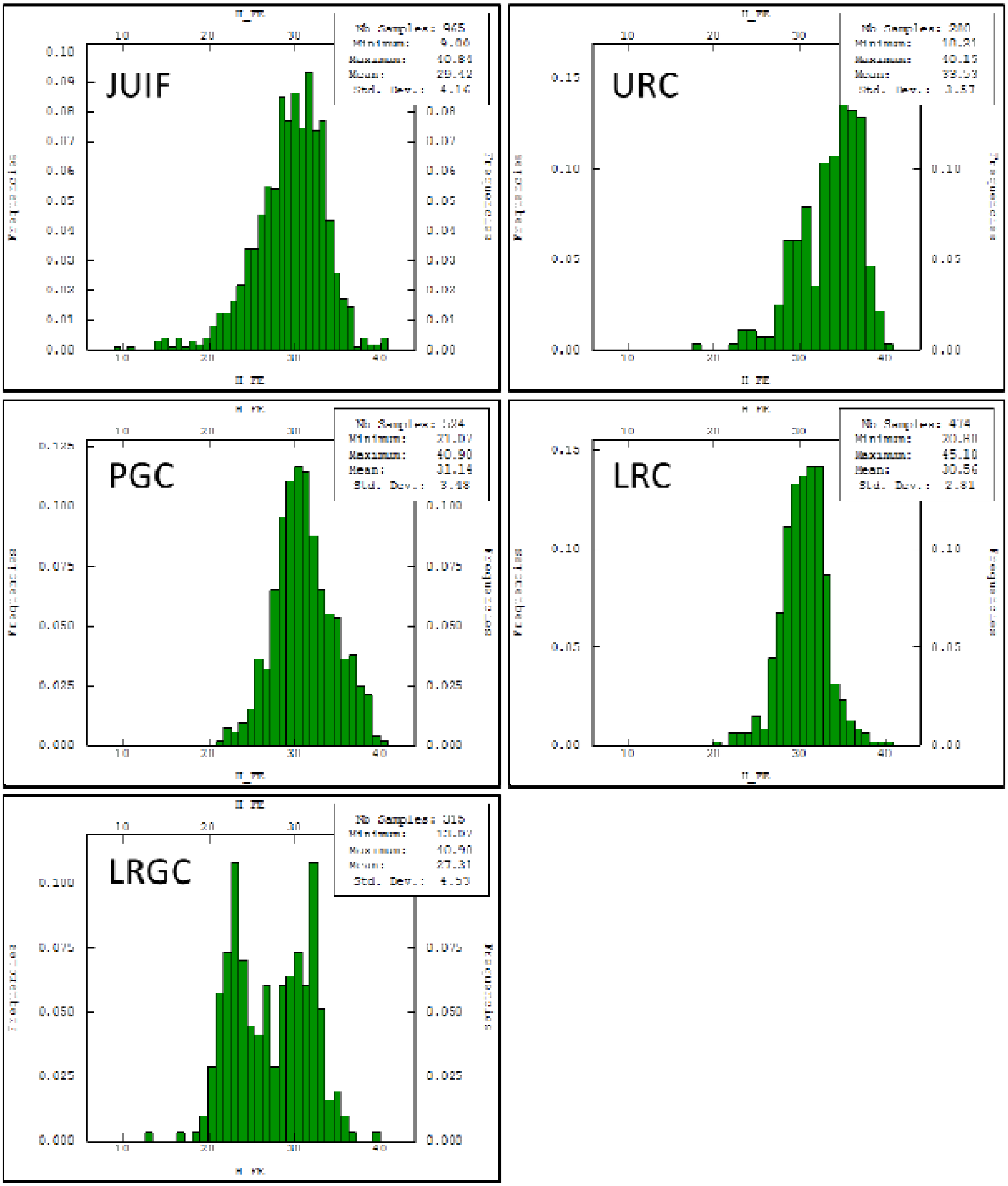


Table 14.4 – Composite Statistics by Domain

Domain	Stats	Fe (%)	Al2O3	SiO2	MnO	P_2O_5	LOI (%)
JUIF	Mean	29.46	0.50	45.05	0.92	0.03	5.79
	Std. Dev.	4.13	0.64	4.38	0.54	0.03	3.32
	Min	9.00	0.01	30.10	0.16	0.01	0.50
	Max	40.84	4.90	69.14	4.66	0.38	21.00
	Count	965	958	965	965	958	965
URC	Mean	33.48	0.15	40.51	0.91	0.02	5.18
	Std. Dev.	3.56	0.30	3.51	0.47	0.01	2.26
	Min	18.00	0.00	33.00	0.00	0.00	1.00
	Max	40.15	3.48	62.38	3.13	0.11	19.70
	Count	280	278	280	280	277	280
PGC	Mean	31.15	0.13	43.54	0.61	0.02	4.96
	Std. Dev.	3.49	0.12	3.90	0.25	0.01	2.04
	Min	21.00	0.00	31.00	0.00	0.00	0.00
	Max	40.90	1.26	56.60	1.72	0.07	15.17
	Count	524	522	524	524	515	524
LRC	Mean	30.54	0.14	45.67	0.54	0.02	3.97
	Std. Dev.	2.80	0.10	3.12	0.39	0.01	1.60
	Min	21.00	0.00	23.00	0.00	0.00	1.00
	Max	45.18	1.32	54.00	3.83	0.05	11.63
	Count	474	472	474	474	468	474
LRGC	Mean	27.44	0.19	47.18	0.65	0.02	6.35
	Std. Dev.	4.50	0.72	3.94	0.22	0.01	2.90
	Min	13.00	0.00	38.00	0.00	0.00	0.00
	Max	40.90	11.20	58.27	1.75	0.06	18.02
	Count	315	311	315	315	306	315

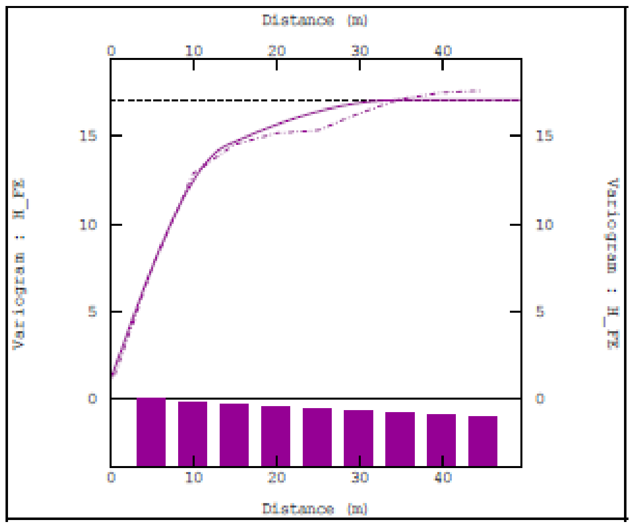
14.7 Geostatistical Analysis and Variography

For the geostatistical study, composites from all resource domains were combined into a single data set. Combining domains improved the variography due to the limited samples present within the individual domains. The LRGC domain was studied separately because it is statistically different (Table 14.4 and Figure 14.7).

Statistical analysis was completed with the ISATIS software. Directional experimental semivariograms were produced for iron, Al_2O_3, SiO_2, P_2O_5, MnO, LOI, and specific gravity. The semivariograms were produced using a 5-meter lag in the downhole direction to allow the nugget to be determined. Semivariograms, to define the directional ranges, were produced using a 500-meter lag and the nugget was fixed using the downhole variogram. Omni directional variograms were produced for specific gravity.

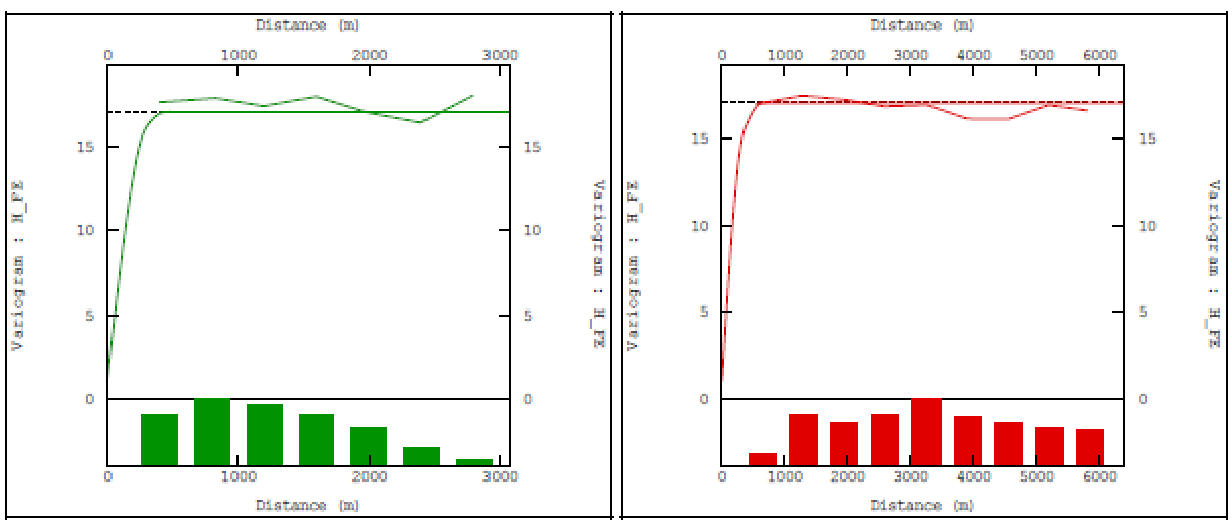
Figure 14.8 and Figure 14.9 show the modelled variograms produced for the combined domains. The number of sample pairs was checked in the variography process to ensure sufficient numbers were being used.

Figure 14.8 – Downhole Iron (%) Semi-variogram



(All domains)

Figure 14.9 – Iron Directional Semi-variogram for Along (Left) and Across (Right) Strike



(All domains)

All variograms were fitted with two spherical structures, but for particular directions (e.g., along strike) only one structure could be fitted. In general, variography shows very good continuity, with second structure ranges between 500 to 1,000 meters. The nugget and ranges are easily generated, providing an appropriate level of confidence for both the short scale variation and the longer range continuity. Similar variogram models were produced for all other variables.

The variograms were re-scaled for each domain using the proportions of the sills. Variography results and parameters are summarized in Table 14.5.

Table 14.5 – Summary of Variogram Parameters

Variable	Azimuth	Nugget Effect	Total Sill	Spherical Structure 1				Spherical Structure 2			
				Sill contrib.	Along Strike (m)	Along Width (m)	Across Strike (m)	Sill contrib.	Along Strike (m)	Along Width (m)	Across Strike (m)
Fe	150	1.08	17.05	9.98	350	300	11	5.99	650	450	33
SiO_2	150	0.72	19.19	10.26	350	300	9	8.21	550	400	33
Al_2O_3	150	0.03	0.40	0.19	700	300	18	0.18	900	450	100
P_2O_5	150	0.00003	0.0009	0.0002	250	300	10	0.0007	500	400	85
MnO	150	0.014	0.29	0.06	500	800	5	0.22	1,000	1,200	33
LOI	150	1.108	11.05	7.10	450	400	17	2.84	600	450	60

The estimation parameters were selected from analysis of the variogram models produced despite the poor resolution of the models because of the limited number of samples available for modelling and the data spacing (borehole spacing).

The results of the variography have been used to assign the appropriate weighting to the sample pairs utilized to estimate block model cells. The total ranges modelled were also considered in the selection of the optimum search parameters and the search ellipse radii dimensions used in the grade interpolation. Ideally, sample pairs that fall within the range of the variogram (where a strong covariance exists between the sample pairs) should be utilized if the data allows.

The results of variography suggest that ordinary kriging is an appropriate interpolation technique.

14.8 Estimation Parameters

To define the optimal parameters used in the grade interpolation, quantitative kriging neighbourhood analysis (QKNA, Vann et al., 2003) was also undertaken, to assist with the selection of search parameters to deliver estimates that are conditionally unbiased. Conditional “unbiasedness” is defined by David (1977) as "…on average, all blocks Z which are estimated to have a grade equal to Zo will have that grade." The criteria considered when evaluating a search area through QKNA, in order of priority, are (Vann et al 2003):

- The slope of regression of the true block grade on the estimated block grade;
- The weight of the mean for a simple kriging;
- The distribution of kriging weights, and proportion of negative weights; and
- The kriging variance.

QKNA provides a useful technique that uses mathematically sound tools to optimize a search area. It is an invaluable step in determining the correct search area for any estimation or simulation exercise.

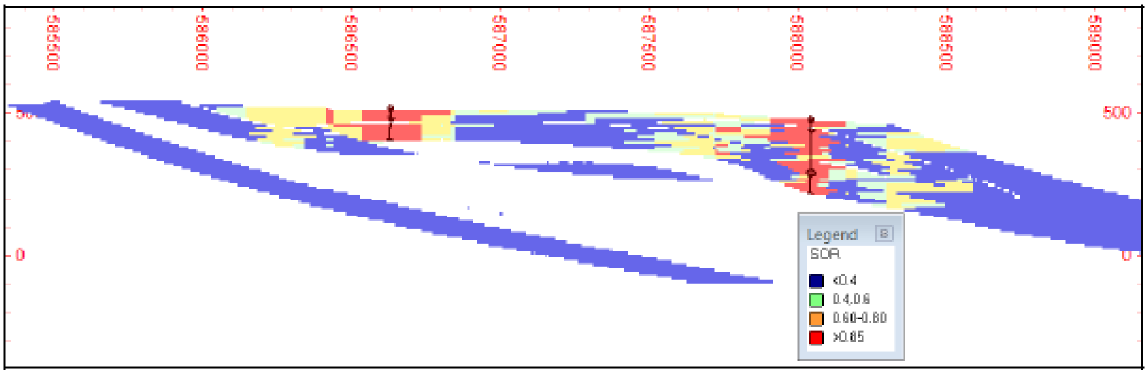
The scenarios were tested by running the estimation in CAE Studio 3 software on the specific domains modelled. The number of blocks filled in each neighbourhood run was checked to ensure that an adequate number of blocks were filled ensuring that meaningful results were generated.

The QKNA process was run to generate the slope of regression results using the chosen search parameters. From a base case, different parameters were tested independently and each run was analysed in order to determine the best result for each parameter.

The selected case QKNA run used a minimum of six composites, a maximum of 25 composites, and a maximum of four composites per borehole. Twenty-six percent of the blocks were filled during the first estimation pass and 34.3 percent during the second estimation pass.

Figure 14.10 illustrates the slope of regression values coloured by the splits in relation to the boreholes, whereby higher slope of regression values are seen around boreholes with a poor correlation between boreholes.

Figure 14.10 – Slope of Regression Distribution Around Well Informed Blocks, Looking Northwest



14.9 Block Model and Grade Estimation

A single block model was created using block sizes of 100 meters by 100 meters by 10 meters (X, Y, and Z, respectively). A rotation of 150 degrees (in azimuth) was applied around the vertical axis using the origin of the model as a base point. Given an average spacing of 400 meters between onsection drill collars and 500 meters between sections, a block size of 100 meters by 100 meters was deemed appropriate. Table 14.6 summarises the block model parameters.

Table 14.6 – Full Moon Iron Deposit Block Model Specifications

	Block Size (m)	Origin* (UTM)	No. Blocks	Subcells	Rotation Azimuth
X	100	587,200	65		
Y	100	6,148,400	120	Yes	150o
Z	10	-200	80		

* UTM Coordinates Nad83 datum, Zone 19

A grade for iron, Al_2O_3, SiO_2, P_2O_5, MnO, LOI, and specific gravity was estimated in each cell of the block model using ordinary kriging and the estimation parameters summarized in Table 14.7. All domains were estimated using dynamic anisotropy to assist the interpolation in areas of folding and of different strike and dip directions. Except specific gravity, all other variables were estimated with selective samples for each subdomain. Specific Gravity was estimated with selective samples for each main domain.

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Table 14.7 – Summary of Estimation Parameters

Variable	Axis 1* (m)	Axis 2* (m)	Axis 3* (m)	Min. Samples	Max. Samples	Max. Samples per borehole
Search Volume 1						
Fe	650	450	33	6	25	4
SiO_2	550	400	33	6	25	4
Al_2O_3	900	450	100	6	25	4
P_2O_5	500	400	85	6	25	4
MnO	1,000	1,200	33	6	25	4
LOI	600	450	60	6	25	4
Search Volume 2						
Fe	1,300	900	66	6	25	4
SiO_2	1,100	800	66	6	25	4
Al_2O_3	1,800	900	200	6	25	4
P_2O_5	1,000	800	170	6	25	4
MnO	2,000	2,400	66	6	25	4
LOI	1,200	900	120	6	25	4
Search Volume 3						
Fe	65,000	45,000	3,300	1	25	4
SiO_2	55,000	40,000	3,300	1	25	4
Al_2O_3	90,000	45,000	10,000	1	25	4
P_2O_5	50,000	40,000	8,500	1	25	4
MnO	100,000	120,000	3,300	1	25	4
LOI	60,000	45,000	6,000	1	25	4

The search ellipse dimensions and orientations were chosen in consideration of variography and QKNA results. The dip and rotation of the ellipse mirrors the overall dip and strike of the individual domains. Dynamic anisotropy was used to honour the geological structure and gentle along strike changes in strike orientation observed. Dynamic anisotropy uses angle data generated from the resource domain to assign dip and dip direction to every block in the model. The search ellipse is then rotated using these angles during estimation to honour the dip and dip direction of that block.

Block estimation was completed using three estimation runs. The first pass considered search volumes adjusted to the Fe variogram full ranges. For the second run, the initial search volumes were doubled. For the third estimation run, the search volumes were inflated to 100 times the initial search volumes and the minimum number of samples required was reduced to one. The third pass was used to make sure that all blocks inside the resource domains were estimated.

In each case the ellipse was validated in CAE Studio 3, prior to estimation to ensure that the correct dip, dip direction, and search radii were applied. Figure 14.11 shows ellipses for selected blocks in two vertical sections as a validation of the dynamic anisotropy process.

Figure 14.11 – Visual Validation of Search Ellipses, Looking Northwest

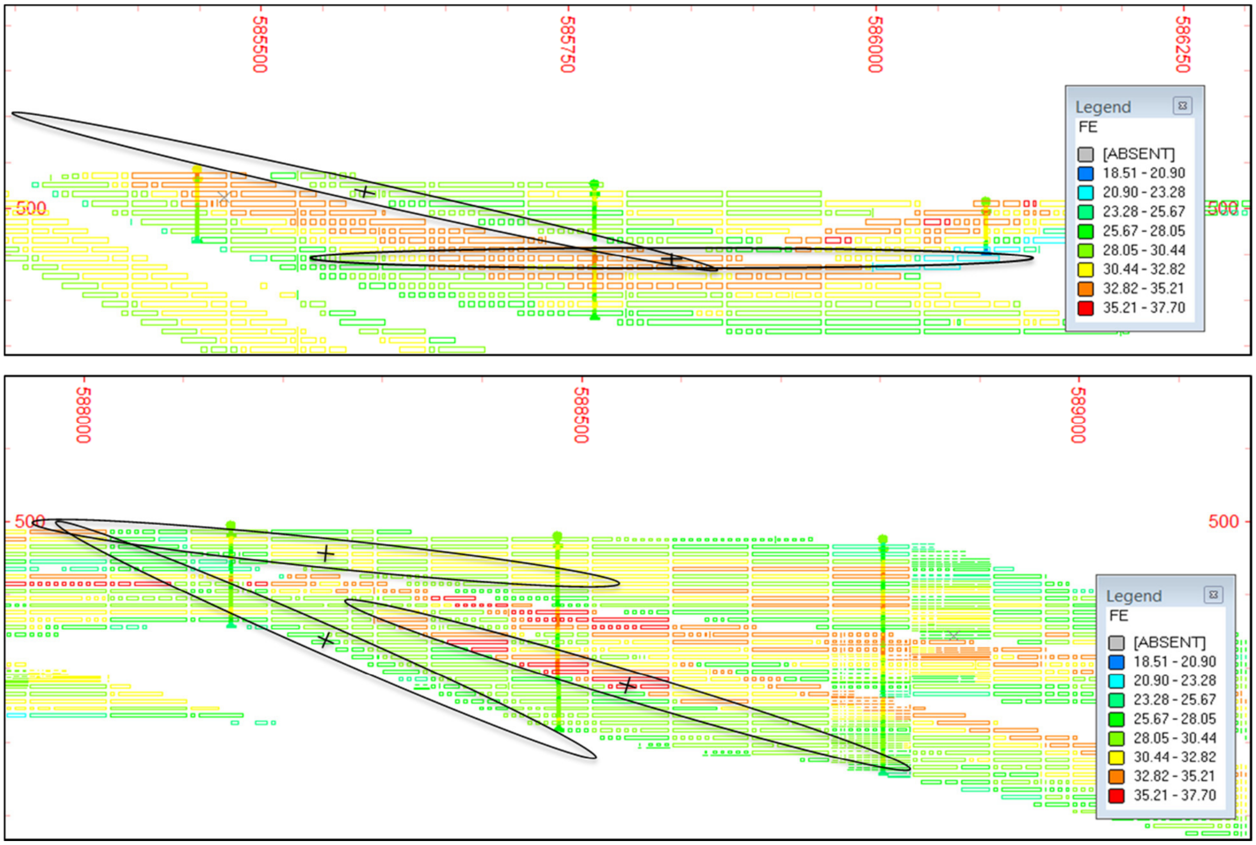


Table 14.8 shows the number of blocks filled during each estimation pass for all mineralized domains. As shown, the number of blocks filled is well balanced among the estimation passes, using search volumes adjusted to the Fe variograms full ranges derived from the geostatistical studies.

Table 14.8 – Summary of Estimation Parameters

Domain	Variable	Mass of Estimated Blocks (t)			Percentage Filled (%)		
		SVol 1	SVol 2	SVol 3	SVol 1	SVol 2	SVol3
JUIF	Fe	2,676	2,162	2,449	36.72	29.67	33.6
URC	Fe	354	735	678	20.01	41.6	38.38
PGC	Fe	905	1,446	1,182	25.62	40.93	33.45
LRC	Fe	816	1,320	1,791	20.77	33.61	45.62
LRGC	Fe	350	984	1,524	12.23	34.45	53.32

14.10 Model Validation and Sensitivity

The block model has been validated using the following techniques:

- Visual inspection of block grades in plan and section, and comparison with borehole grades;
- comparison of global mean block grades and sample grades within mineralized domains; and
- Comparison of local grades between samples and estimated blocks.

The global block means was compared with the sample means for iron, SiO_2, Al_2O_3, P_2O_5, MnO, LOI, and specific gravity. Table 14.9 shows the key results from the domains. Overall, SRK is confident that the global block model grades and input composite grades show a reasonable comparison. However, SRK does acknowledge that minor discrepancies do exist, particularly in Al_2O_3 where the actual grade is considered low.

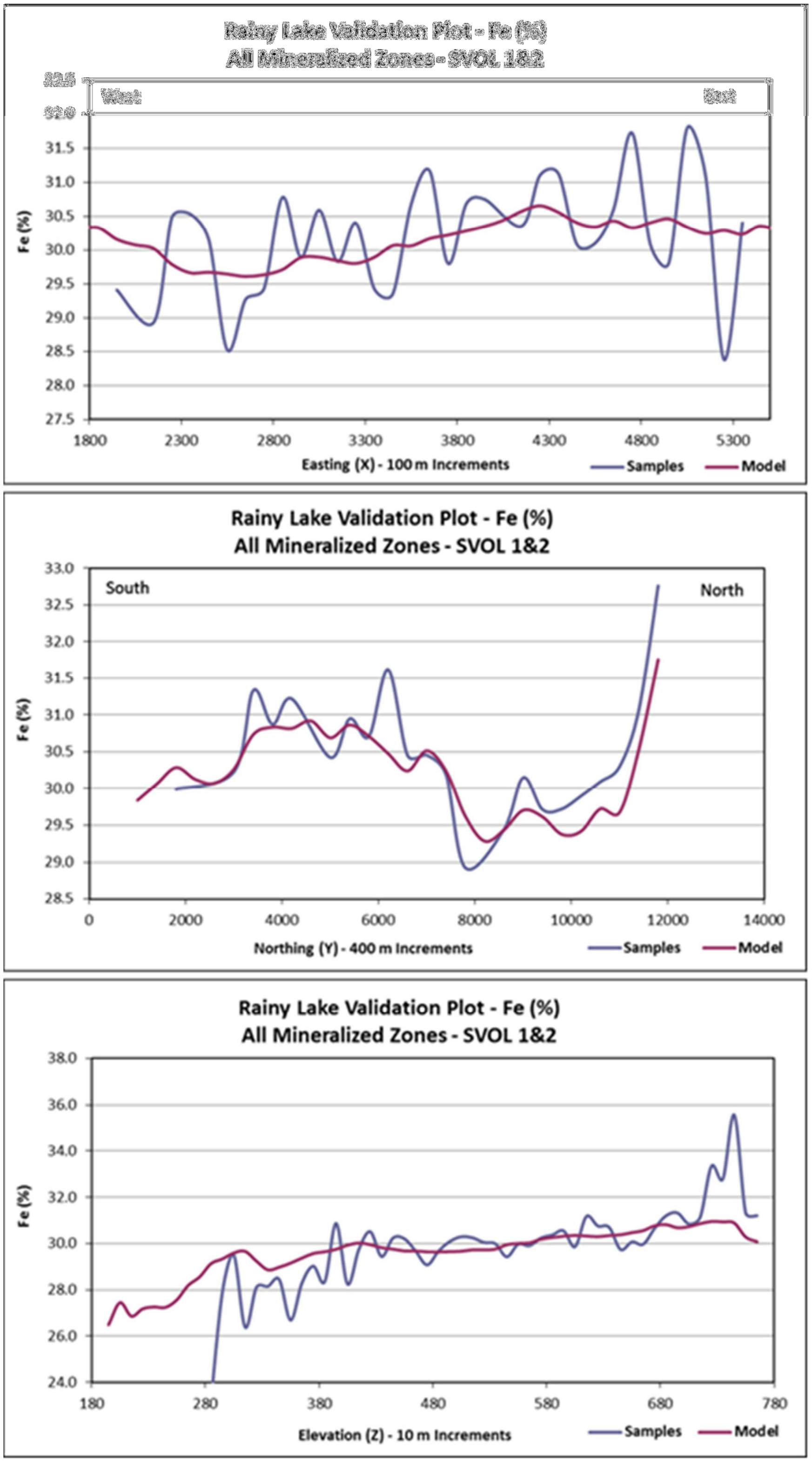
Table 14.9 – Comparison of Block and Sample Mean Grades

Domain	Variable	Composite Mean Grade	Block Mean Grade	Difference
JUIF	Fe	29.46	29.36	-0.10
	Al_2O_3	0.50	0.47	-0.03
	SiO_2	45.05	45.02	-0.03
	P_2O_5	0.03	0.04	0.01
	MnO	0.92	0.94	0.02
	LOI	5.79	5.96	0.17
	SG	3.20	3.21	0.01
URC	Fe	33.48	33.40	-0.08
	Al_2O_3	0.15	0.15	0.00
	SiO_2	40.51	40.56	0.05
	P_2O_5	0.02	0.03	0.01
	MnO	0.91	0.94	0.03
	LOI	5.18	5.22	0.04
	SG	3.33	3.32	-0.01
PGC	Fe	31.15	31.19	0.04
	Al_2O_3	0.13	0.14	0.01
	SiO_2	43.54	43.69	0.15
	P_2O_5	0.02	0.03	0.01
	MnO	0.61	0.60	-0.01
	LOI	4.96	4.81	-0.15
	SG	3.27	3.25	-0.02
LRC	Fe	30.54	30.51	-0.03
	Al_2O_3	0.14	0.14	0.00
	SiO_2	45.67	45.68	0.01
	P_2O_5	0.02	0.03	0.01
	MnO	0.54	0.52	-0.02
	LOI	3.97	3.98	0.01
	SG	3.24	3.22	-0.02
LRGC	Fe	27.44	27.20	-0.24
	Al_2O_3	0.19	0.19	0.00
	SiO_2	47.18	47.42	0.24
	P_2O_5	0.02	0.03	0.01
	MnO	0.65	0.65	0.00
	LOI	6.35	6.44	0.09
	SG	3.14	3.14	0.00

The local block means have been compared with the sample means for iron, SiO_2, Al_2O_3, P_2O_5, MnO, LOI, and specific gravity through sectional validation plots, or SWATH plots. Figure 14.12 shows the validation plots for iron comparing the averages of the composites iron grades and the iron block grades along the X, Y, and Z directions.

As expected, the estimated block values are smoothed around the composite values. Due to the relatively poor variography and limited samples, minor grade discrepancies do exists on a local scale, although overall, SRK is confident that the interpolated grades reflect the available input sample data and the estimate shows no sign of bias.

Figure 14.12 – Iron Validation Plots with Averages for Composites and Blocks in Stripes



14.11 Mineral Resource Classification

Block model quantities and grade estimates for the Full Moon iron deposit were classified according to the CIM Definition Standards on Mineral Resources and Mineral Reserves (November 2010) by Filipe Schmitz Beretta under the supervision of Mark Campodonic (MAusIMM, CP#225925) and Dr. Jean-Francois Couture, P.Geo. (OGQ#1106, APGO#0197).

Mineral resource classification is typically a subjective concept, and industry best practices suggest that resource classification should consider the confidence in the geological continuity of the modelled mineralization, the quality and quantity of exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating these concepts to delineate regular areas at a similar resource classification.

SRK is satisfied that the geological model for the Full Moon iron deposit honours the current geological information and knowledge. The location of the samples and the assaying data are sufficiently reliable to support resource evaluation and do not present a risk that should be taken into consideration for resource classification.

The mineral resource model is informed from core boreholes drilled at 400 to 600 meters spacing. The geological information is sufficiently dense to demonstrate reasonable continuity of the geological units containing the iron mineralization between sampling points and interpret its geometry with reasonable confidence.

The following criteria were considered:

- **Geological Continuity**: SRK considers that the confidence in geological continuity of the iron mineralization is good. Therefore, on the basis of geological continuity alone, Indicated or Inferred categories can be reported at the present drilling spacing (400 by 500 meters). The confidence in the geological continuity deteriorates near the cross fault and near the model boundaries. On this basis, an Inferred classification was assigned to blocks located no more than 500 meters away from the last informing boreholes in all directions. For those blocks located outside that perimeter, SRK considers that the confidence in the geology model is not sufficient to support a classification. Those blocks remain uncategorized.

- **Grade Continuity**: The variography study shows good continuity of grade with first and second structures ranges between 350 and 650 meters. On this basis, an Indicated classification can be assigned to blocks located within 500 to 650 meters of informing composites and an Inferred classification to a distance not exceeding 1 300 meters from informing composites.
- **Estimation Quality**: For the first estimation run, the search neighbourhoods have been set slightly larger than the average data spacing. The quality of the block estimates can be evaluated using the slope of regression in parallel with the search volume used. Good estimates are made with the first estimation run, with average slopes generally in excess of 0.6. Therefore the grade estimates of blocks estimated during the first estimation run with a slope of regression greater than 0.6 are sufficiently reliable to be assigned an Indicated classification. Conversely, other blocks estimated during the first run and those estimated during the second and third estimation pass were assigned an Inferred classification.

The following classification has been applied to the Full Moon block model:

- **Measured Mineral Resource**: Non reported (drill spacing of 200 by 250 meters is required to demonstrate geological and grade continuity, and to improve the variogram models);
- **Indicated Mineral Resource**: Contiguous volumes of mineralisation informed by boreholes spaced at 400 by 500 meters or less. Blocks estimated during the first estimation run with a slope of regression greater than or equal to 0.6;
- **Inferred Mineral Resource**: Contiguous volumes of mineralisation informed by boreholes spaced at 400 by 500 meters or less. Blocks were estimated using composites from at least 2 boreholes by any of the three estimation runs and are located not farther than 500 meters from the last boreholes in all directions and to a depth not exceeding 400 meters; and
- **Uncategorized**: All remaining blocks in the model.

The final block classification is summarized in Figure 14.13 and Figure 14.14. The northwest portion of the deposit was estimated using samples from other domains due to the lack of data. For this reason SRK considers that that portion of the deposit should remain unclassified until additional drilling information is acquired.

14.12 Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (November 2010) define a mineral resource as:

"[A] concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a mineral resource are known, estimated or interpreted from specific geological evidence and knowledge."

The "reasonable prospects for economic extraction" requirement generally implies that quantity and grade estimates meet certain economic thresholds and that mineral resources are reported at an appropriate cut-off grade taking into account extraction scenarios and processing recovery. SRK considers that the iron mineralization delineated by core drilling is amenable to open pit extraction. To assist with determining which portions of the iron mineralization modelled by SRK show "reasonable prospect for economic extraction" from an open pit, and to assist with selecting reasonable reporting assumptions, SRK used a pit optimizer to develop conceptual open pit shells using the following reasonable assumptions derived from similar projects. In absence of specific metallurgical data for each resource domain, SRK used average recovery information sourced from nearby similar taconite projects targeting the Sokoman Formation. The main optimization assumptions considered are:

- Overall slope angle 50 degrees;
- Overall mining costs of US$1.50 per tonne mined;
- Overall processing, shipping and G&A costs of $14.50 per tonne processed;
- Royalties costs of US$2.25 per tonne sold;
- Average head grade of 30 percent iron, plant recovery of 60 percent, product grade of 70 percent iron; and
- Selling prices varying between US$45 to US$300 per dry metric tonne of iron concentrate.

Figure 14.13 – Plan View Showing Mineral Resource Classification and Cross-Section Locations

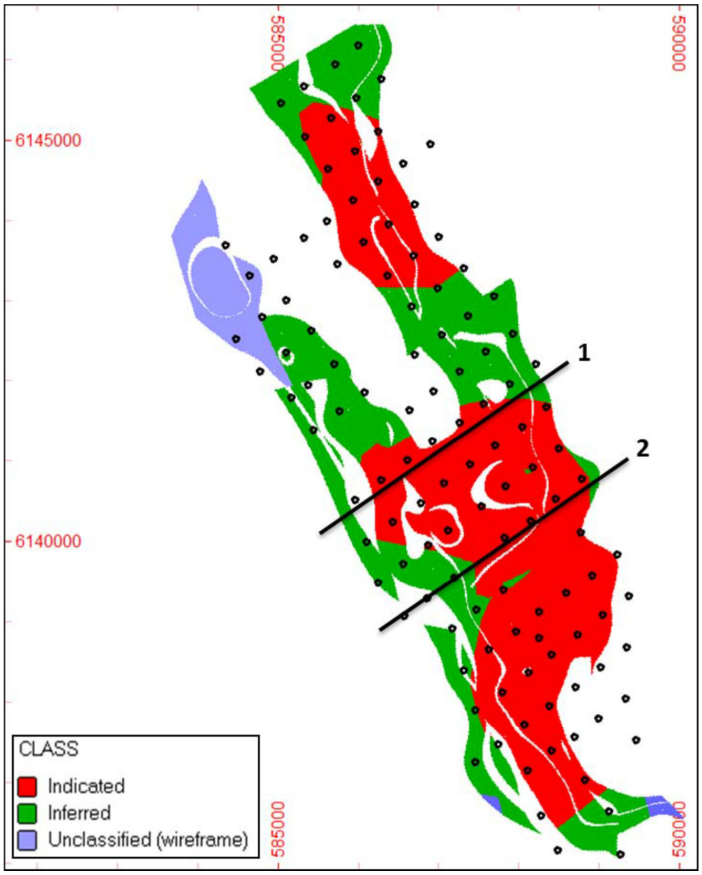
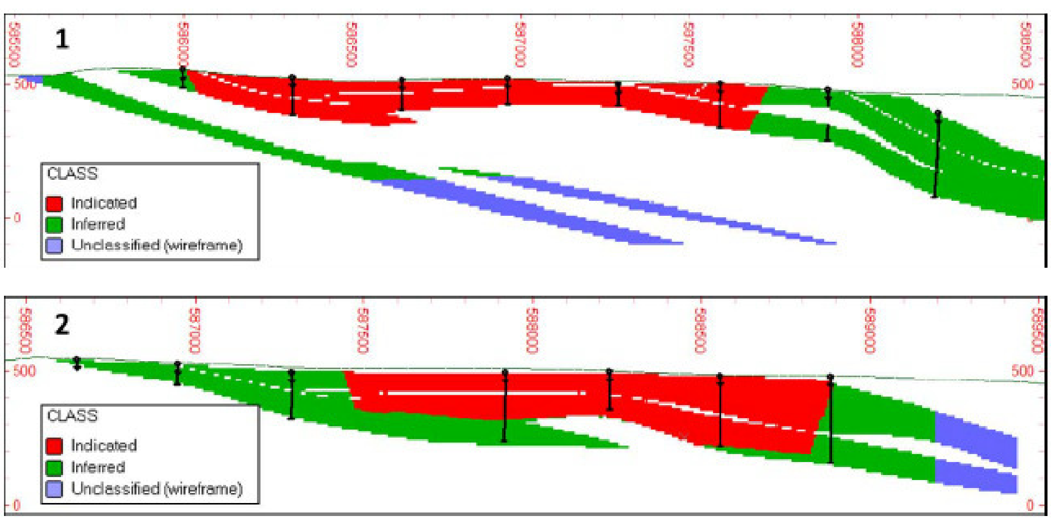


Figure 14.14 – Vertical Sections (1 and 2) Showing Block Classification with Conceptual Pit Shell



The pit optimization results are used solely for the purpose of testing the "reasonable prospects for economic extraction," and do not represent an attempt to estimate mineral reserves. There are no mineral reserves at the Full Moon iron deposit. The optimization results are used to assist with the preparation of a Mineral Resource Statement and to select and appropriate reporting assumptions. After review, SRK considers that the iron mineralization located within the conceptual open pit shell above a cut-off grade of 20 percent total iron satisfies the definition of a mineral resource and thus can be reported as a mineral resource.

Blocks located outside the conceptual pit envelope do not meet the "reasonable prospects for economic extraction" requirement, and therefore, cannot be reported as a mineral resource. Mineral resource reporting was completed in CAE Studio 3 using the conceptual pit envelope. Quantities and major element grade estimates for each resource domain are reported separately. Mineral resources were estimated in conformity with generally accepted CIM Estimation of Mineral Resource and Mineral Reserve Best Practices Guidelines. The mineral resources are not mineral reserves and do not have demonstrated economic viability.

The mineral resources discussed herein may be affected by subsequent assessments of mining, environmental, processing, permitting, taxation, socio-economic, political, and other factors. There is insufficient information available to assess the extent of which the mineral resources may be affected by these factors. The initial Mineral Resource Statement for the Full Moon iron deposit is presented in Table 14.10. The statement was prepared by Filipe Schmitz-Berretta under the supervision of Mr. Mark Campodonic (CP#225925) and Dr. Jean-Francois Couture, P.Geo. (OGQ#1106, APGO#0197). Mr. Campodonic and Dr. Couture are Qualified Persons pursuant to National Instrument 43-101 and independent from WCSLIM. The effective date of the Mineral Resource Statement is October 22, 2012.

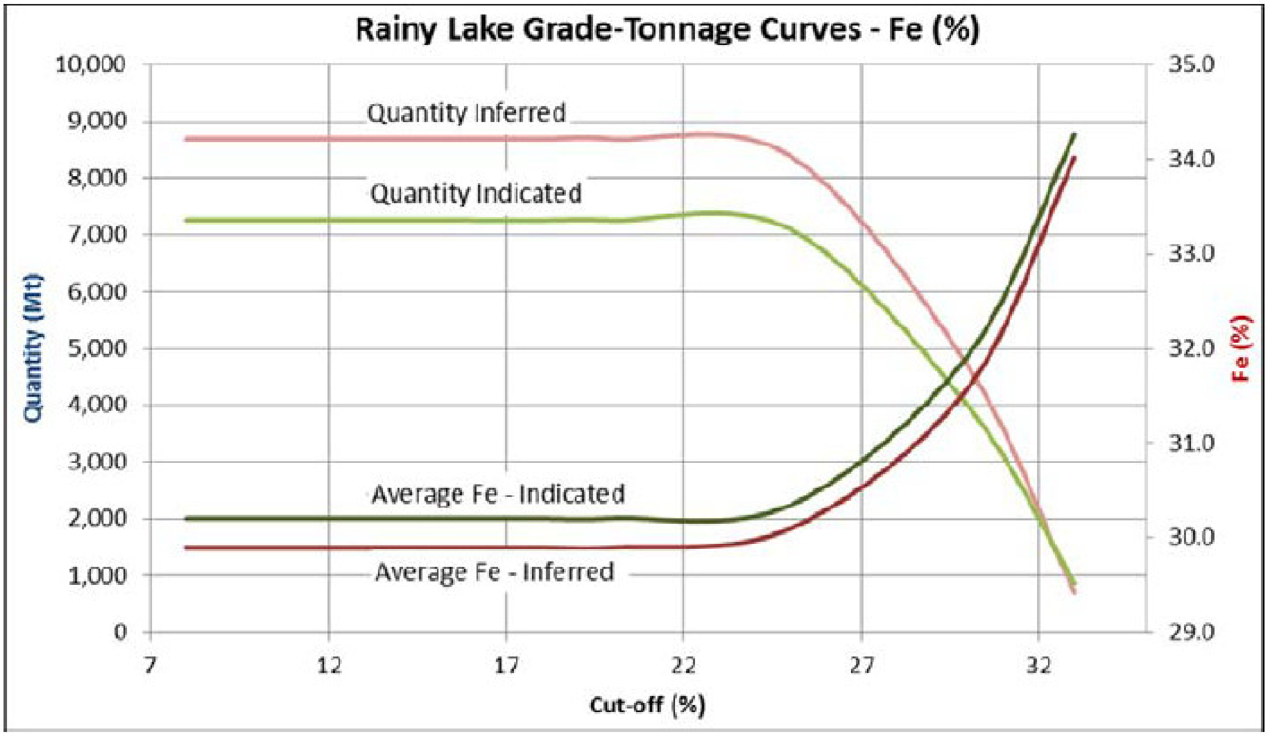
Table 14.10 – Mineral Resource Statement*, Full Moon Iron Deposit, Rainy Lake Property, Sunny Lake Project, Québec, SRK Consulting (Canada) Inc., October 22, 2012

Domain	Volume (Mm^3)	Quantity (Mt)	SG	Fe (%)	SiO_2 (%)	Al_2O_3 (%)	P_2O_5 (%)	P** (%)	MnO (%)	MN** (%)	LOI (%)
Indicated Mineral Resources											
JUIF	1,109.4	3,562.8	3.21	29.45	45.06	0.50	0.03	0.02	0.90	0.70	5.86
URC	235.4	777.1	3.30	33.51	40.31	0.12	0.02	0.01	0.96	0.75	5.37
PGC	399.6	1,314.8	3.29	31.30	43.31	0.12	0.02	0.01	0.61	0.47	5.01
LRC	309.2	997.0	3.22	30.58	45.71	0.14	0.02	0.01	0.52	0.40	4.01
LRGC	194.7	607.9	3.12	27.40	47.13	0.17	0.02	0.01	0.67	0.52	6.52
Total Indicated	**2,248.2**	**7,259.6**	**3.23**	**30.18**	**44.52**	**0.31**	**0.03**	**0.01**	**0.78**	**0.61**	**5.46**
Inferred Mineral Resources											
JUIF	683.0	2,185.2	3.20	29.17	45.14	0.48	0.03	0.02	0.97	0.75	5.99
URC	235.1	787.1	3.35	33.35	40.69	0.18	0.02	0.01	0.93	0.72	5.12
PGC	547.3	1,773.2	3.24	31.14	43.90	0.14	0.02	0.01	0.58	0.45	4.70
LRC	690.1	2,239.4	3.25	30.43	45.71	0.14	0.02	0.01	0.52	0.40	3.98
LRGC	543.5	1,708.6	3.14	27.22	47.38	0.21	0.02	0.01	0.65	0.51	6.44
Total Inferred	**2,699.0**	**8,693.5**	**3.22**	**29.86**	**45.10**	**0.24**	**0.02**	**0.01**	**0.71**	**0.55**	**5.23**

* Reported at a cut-off grade of 20 percent total iron inside a conceptual pit envelope optimized onsidering reasonable open pit mining, processing and selling technical parameters and costs benchmark against similar taconite iron projects and a selling price of US$110 per dry metric tonne of iron concentrate. All figures are rounded to reflect the relative accuracy of the estimates. Mineral resources are not mineral reserves and do not have a demonstrated economic viability.

** Converted from estimated oxide

Figure 14.15 – Grade-Tonnage Curve



14.13 Sensitivity Analysis

The mineral resources are sensitive to the selection of a reporting cut-off grade. To illustrate this sensitivity, resource model quantities and grade estimates are presented in Table 14.11 and summarized in a grade tonnage curve in Figure 14.15. The reader is cautioned that the figures presented in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of a cut-off grade. Figure 14.15 presents this sensitivity as a grade tonnage curve. The grade-tonnage curve shows a decreasing tonnage with an associated increasing iron grade from above a 23 percent iron cut-off.

Table 14.11 – Global Quantities and Grade Estimates* at Various Cut-off Grades

Domain	Volume (Mm^3)	Quantity (Mt)	SG	Fe (%)	SiO_2 (%)	Al_2O_3 (%)	P_2O_5 (%)	P** (%)	MnO (%)	MN** (%)	LOI (%)
Indicated											
16	2,249.10	7,262.10	3.23	30.20	44.50	0.31	0.03	0.01	0.78	0.61	5.46
18	2,248.9	7,261.6	3.23	30.20	44.50	0.31	0.03	0.01	0.78	0.61	5.46
20	**2,248.2**	**7,259.6**	**3.23**	**30.20**	**44.50**	**0.31**	**0.03**	**0.01**	**0.78**	**0.61**	**5.45**
25	2,197.0	7,099.9	3.23	30.34	44.40	0.31	0.03	0.01	0.78	0.61	5.38
30	1,222.5	3,988.7	3.26	31.93	43.32	0.27	0.03	0.01	0.75	0.58	4.62
33	263.7	873.2	3.31	34.26	40.22	0.14	0.02	0.01	0.80	0.62	4.66
Inferred											
16	2,699.0	8,693.5	3.22	29.89	45.07	0.24	0.02	0.01	0.71	0.55	5.22
18	2,699.0	8,693.5	3.22	29.89	45.07	0.24	0.02	0.01	0.71	0.55	5.22
20	**2,699.0**	**8,693.5**	**3.22**	**29.89**	**45.07**	**0.24**	**0.02**	**0.01**	**0.71**	**0.55**	**5.22**
25	2,600.0	8,390.1	3.23	30.10	44.91	0.24	0.02	0.01	0.71	0.55	5.13
30	1,433.1	4,685.8	3.27	31.59	43.94	0.21	0.02	0.01	0.66	0.51	4.37
33	210.5	704.0	3.34	34.02	40.53	0.16	0.02	0.01	0.81	0.62	4.57

*The reader is cautioned that the figures presented in this table should not be misconstrued as a Mineral Resource Statement. The reported quantities and grades are only presented as a sensitivity of the deposit model to the selection of cut-off grade.

15. Mineral Reserve Estimate

Since this report is a Preliminary Economic Assessment report, no Mineral Reserves have been estimated for the Full Moon deposit as per NI 43-101 regulations. In-pit Mineral Resources are described in Section 16.

16 Mining Methods

The mining methods and In-pit Mineral Resource estimate for the Full Moon Iron deposit were prepared by Jeffrey Cassoff, Eng., Lead Mining Engineer with Met-Chem Canada Inc. and Qualified Person. All of the work related to the mine design for the PEA was done using MineSight® Version 9.20. MineSight® is a commercially available software that has been used by Met-Chem for over 30 years.

16.1 Block Model

As was presented in Section 7 of this report, the Full Moon deposit is composed of several different lithological units. The deposit is covered by a layer of sand and gravel which is defined as overburden. Underneath the overburden lies the Menihek Shale which is a non-magnetic rock formation. The Menihek Shale only appears on the east side of the deposit. Below the overburden and Menihek Shale is the Sokoman Iron Formation which is composed of eight (8) units. The Mineral Resource estimate that was completed by SRK Consulting Inc. was limited to five (5) of the units since the iron content was very weak in the other three (3). The units are listed below and are identified as either resource or waste units:

- Lean Chert – Waste ;
- Jasper Upper Iron Formation – Mineral Resource ;
- Green Chert – Waste ;
- Upper Red Chert – Mineral Resource ;
- Pink Grey Chert – Mineral Resource ;
- Lower Red Chert – Mineral Resource ;
- Lower Red Green Chert – Mineral Resource ;
- Lower Iron Formation – Waste.

Below the Lower Iron Formation lies the Wishart Formation which is non-magnetic.

The mine design for the PEA is based on the Mineral Resource block model that was prepared by SRK and presented in Section 14 of this report. SRK provided Met-Chem with an Excel csv file for each of the five (5) lithological units that contain Mineral Resources (JUIF, LRC, LRGC, PGC and URC). Each file contains the coordinates for the blocks that are within lithological unit, the block density, the block classification (indicated or inferred) as well as the following grade items for each block; Fe, Al_2O_3, SiO_2, P_2O_5, MnO and LOI. The blocks are 100 m x 100 m x 10 m high and have a rotation of 150°. SRK also provided Met-Chem with wireframe solids that represent the extents of each lithological unit. Met-Chem created a mine planning block model by importing the five (5) Excel csv files to identify the resource blocks and used the wireframe solids to identify the non resource blocks.

The topographic surface that was used for the mine design was also supplied by SRK and was created from a high resolution LIDAR survey that was completed over the project area during the summer of 2012.

Since the block model does not provide the magnetite or the hematite content for each block nor does it provide the amount of recoverable iron, Soutex Inc. provided Met-Chem with the following formulas to correlate a weight recovery from the total iron. The formulas are based on the results of the metallurgical testwork that was presented in more detail in Section 13 of this report:

- JUIF: Weight Recovery % = 1.0411 x Total Fe% + 3.3655 ;
- PGC: Weight Recovery % = 1.3293 x Total Fe% + 0.8395 ;
- URC: Weight Recovery % = 0.9113 x Total Fe% + 8.463 ;
- LRC: Weight Recovery % = 1.4233 x Total Fe% - 0.9894 ;
- LRCG: Weight Recovery % = 1.77 x Total Fe% - 23.99.

In order to calculate the amount of concentrate that is produced from a given mineralized block in the model, the tonnage of the block is multiplied by its weight recovery. The correlation assumes that there is a magnetite and a hematite processing facility and that the grade of the concentrate produced will be 66% Fe.

16.2 Material Properties

The material properties for the different lithological units are presented below. These properties are important in estimating the In-pit Mineral Resources, the equipment fleet requirements as well as the dump and stockpile design capacities.

16.2.1 Density

In order to estimate the specific gravity of the iron formation, WCSLIM Iron used a standard weight in air / weight in water methodology on representative core samples which was presented in more detail in Section 14 of this report. Using the results from this testwork, SRK modelled a different density for each resource block in the model which is a function of the head grade. The average in-situ dry density of the Mineral Resources is 3.23 t/m^3. A density of 2.1 t/m^3 was used for the overburden, 3.0 t/m^3 for the Menihek Shale and 3.2 t/m^3 for the lower grade lithological units that are considered as waste rock (LC, GC and LIF).

16.2.2 Swell Factor

The swell factor reflects the increase in volume of material from its in-situ state to after it is blasted and loaded into the haul trucks. The swell factor is an important parameter that is used to determine the loading and hauling equipment requirements as well as the dump and stockpile designs. A swell factor of 45% was used for the PEA, which is a typical value used for open pit hard rock mines. Once the rock is placed in the waste dump, the swell factor is reduced to 30% due to compaction. A swell factor of 30% was used for the overburden, 15% following compaction.

16.2.3 Moisture Content

The moisture content reflects the amount of water that is present within the rock formation. It affects the estimation of haul truck requirements and must be considered during the payload calculations. The moisture content is also an important factor for the process water balance.

A moisture content of 2% was used for the PEA, which is typical for similar projects in the region.

16.3 Mining Method

The mining method selected for the Project is a conventional open pit, drill and blast, truck and shovel operation with 10 meter high benches.

Topsoil and overburden will be stripped and stockpiled for future reclamation use. The mineralization and waste rock will then be drilled, blasted and loaded into rigid frame haul trucks with hydraulic shovels. The mineralized material will be hauled to the primary crushers and the waste rock will be hauled to the waste rock pile.

To properly manage water infiltration into the pit, sumps will be established at the lowest points on the pit floor. Water collected in these sumps will be pumped to collection points at surface.

The mine will operate 365 days per year, 24 hours per day. The fleet requirements and manpower are based on this work schedule.

16.4 Pit Optimization

A pit optimization analysis was conducted to determine the cut-off grade and to what extent the deposit can be mined profitably. The pit optimization analysis was done using the MS-Economic Planner module of MineSight® Version 9.20. The optimizer uses the 3D Lerchs Grossman algorithm to determine the economic pit limits based on input of mining and processing costs and revenue per block. Since this study is at a PEA level, NI 43-101 guidelines allow Inferred Mineral Resources to be used in the optimization and mine plan.

Table 16.1 presents the parameters that were used for the pit optimization analysis. All figures are in Canadian Dollars. The cost and operating parameters that were used are preliminary estimates for developing the economic pit and should not be confused with the operating costs subsequently developed for the PEA and presented in Section 21. The sales price for concentrate of $110 /t FOB Sept-Îles was provided by WCSLIM and is discussed in more detail in Section 19 of this report.

Table 16.1 – Pit Optimization Parameters

Item	Units	Value
Mining Cost	$/t (mined)	2.50
Processing Cost	$/t (milled)	4.00
General and Administration Cost	$/t (milled)	2.00
Transportation Cost	$/t (conc.)	20.00
Sales Price (FOB Sept-Îles)	$/t (conc.)	110.00
Overall Pit Slope	Degrees	52
Discount Rate	%	8

The pit optimization showed that the all of the Mineral Resources are economic to mine and could be done with a stripping ratio of 0.4 to 1. In fact, the break-even selling price for the Full Moon deposit is calculated to be $48/t. of concentrate FOB Sept-Iles.

Mining all of the Mineral Resources at the planned production rate of 20 Mt of concentrate per year would result in a 290 year mine life. It was therefore decided by the Project team that the PEA would be limited to a 30 year mine life. The mine life was limited since a market study cannot be reliably conducted for the period of 290 years because the cash flows generated beyond 30 years have little impact on the internal rate of return ("IRR"), and payback period of a project.

In order to determine the best location for the 30 year open pit, Met-Chem used the following criteria:

- Maximize Fe grade (Weight Recovery) ;
- Minimize stripping ratio ;
- Minimize environmental disturbance ;
- Minimize haulage cost to the primary crusher.
- Indicated Mineral Resources are favoured over Inferred Mineral Resources.

Figure 16.1 and Figure 16.2 present plan views showing the weight recovery distribution of the Mineral Resources and the stripping ratio distribution respectively. In order to minimize the environmental disturbance, the limits of the 30 year open pit avoid as many wetlands, lakes and creeks where possible. A minimum offset of 100 m from Lac Rainy and Lac Eclipse to the limit of the pit was also considered.

With respect to minimizing the haulage cost, it was important to identify the location of the primary crushers early on during the PEA. This location which is discussed in more detail in Section 18 is at the south end of Lac Eclipse, at the western limit of the resource boundary. The primary crushers are connected to the plant site with two (2) conveyors that are approximately 1.2 and 1.7 km long.

Considering the above mentioned criteria, Met-Chem used pit optimization techniques to identify the best location for the 30 year open pit. In the optimization, the selling price was reduced, the mining cost was incrementally increased by $0.05/t for each 500 m in horizontal distance from the primary crushers and the mining cost was incrementally increased by $0.02/t for each 10 m in depth. The resulting pit shell whose outline is presented in Figure 16.3 provides a mine life of approximately 60 years at the planned production rate. As is shown on the same figure, the 30 year open pit focuses on the southern part of the optimized pit shell. The detailed design of the 30 year open pit for the PEA is presented in the following section of this report.

Figure 16.4 presents a typical section through the deposit showing the ultimate pit optimization shell, the 60 year open pit shell and the 30 year open pit design as well as the wireframe solids for the five (5) units that contain Mineral Resources.

Figure 16.1 – Weight Recovery Distribution

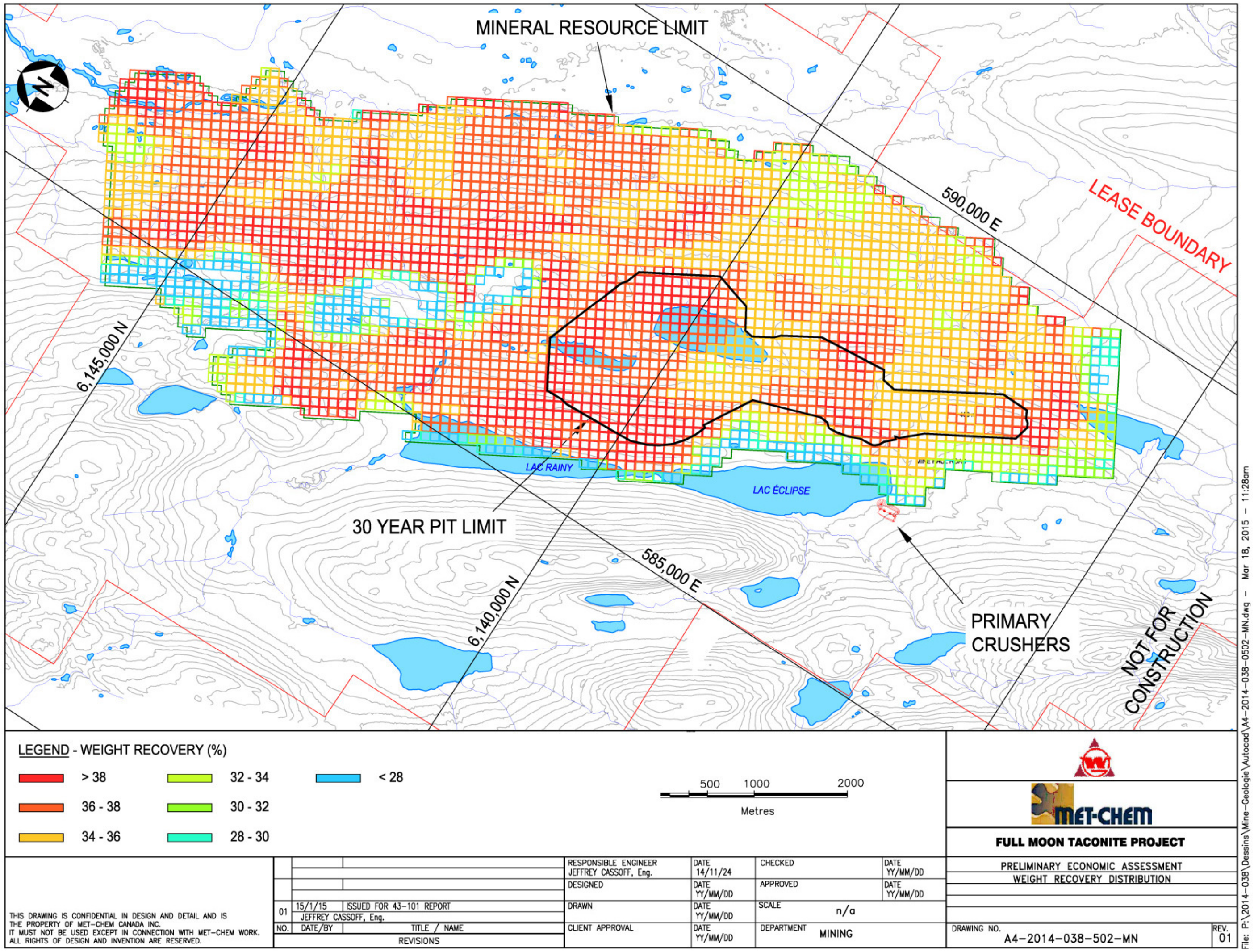


Figure 16.2 – Stripping Ratio Distribution

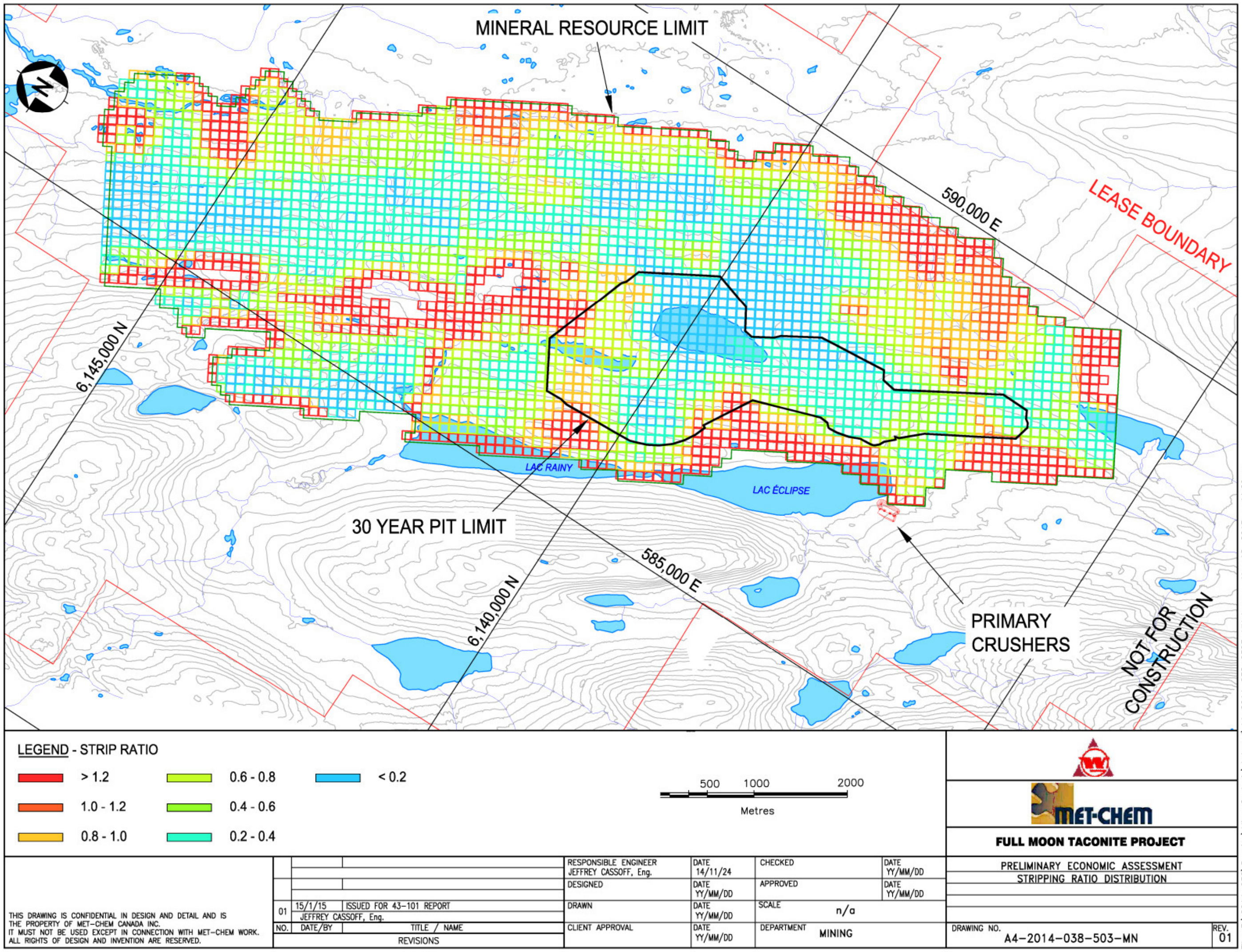


Figure 16.3 – Pit Optimization for 30 Year Open Pit

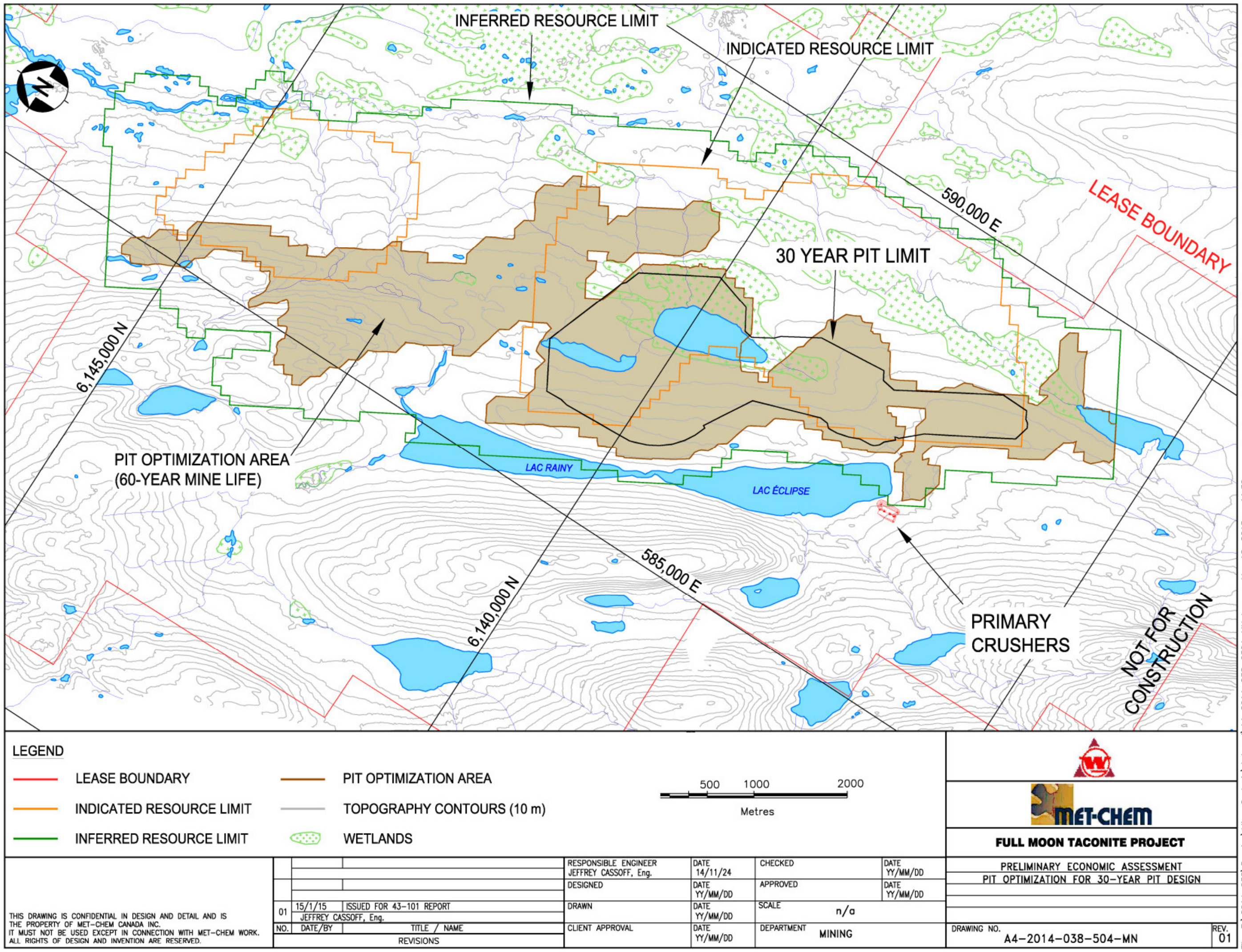
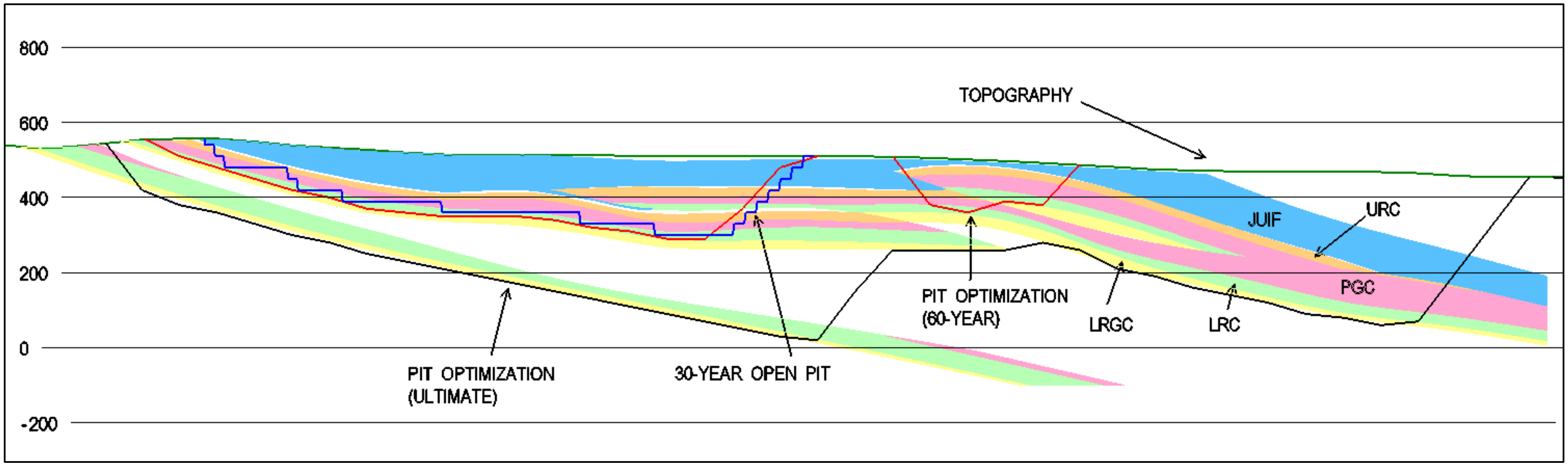


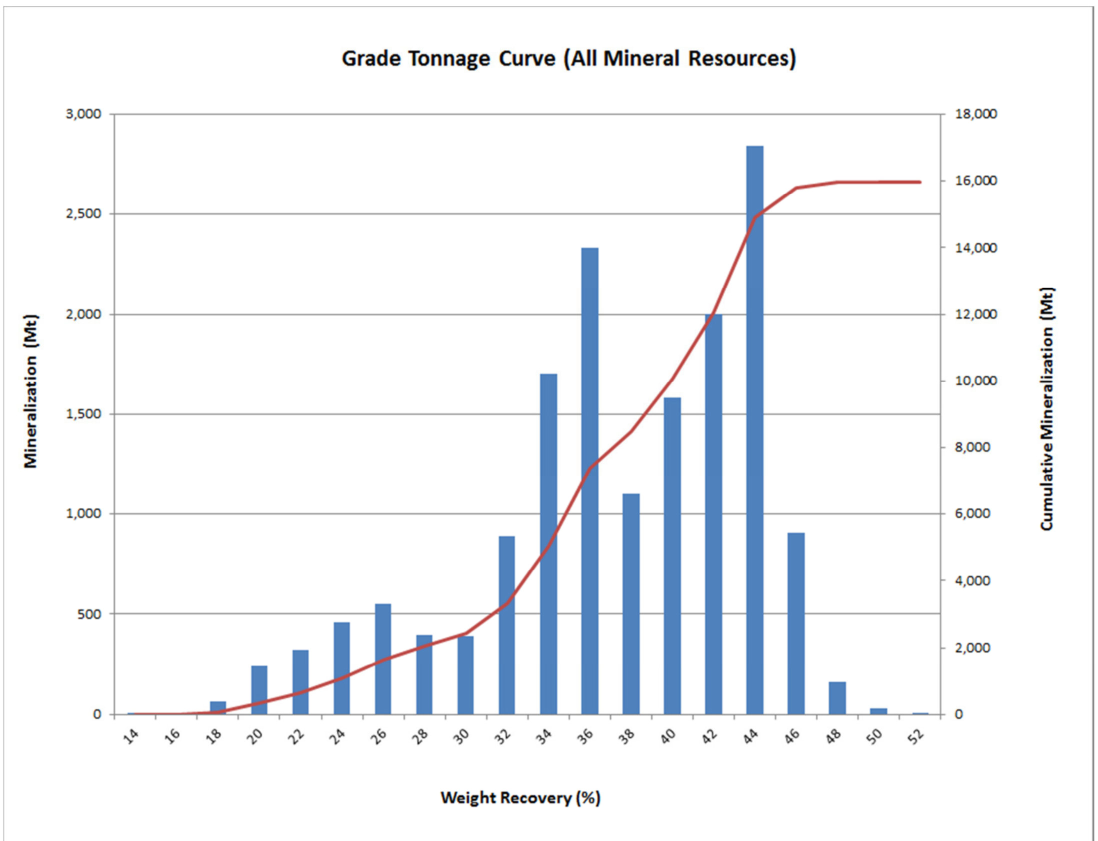
Figure 16.4 – Pit Optimization Results (Typical Section)



A cut-off grade is calculated for each deposit to determine whether the material being mined will generate a profit after paying for the mining, processing, transportation and G&A costs. Material that is mined below the cut off grade is either sent to the waste dump or stockpiled for future processing. Using the economic parameters presented in Table 16.1, the cut off grade for the PEA was calculated to be Weight Recovery > 10%. In order to account for a profit margin and to be in-line with the other projects in the region, it was decided to use a cut-off of Weight Recovery > 18%.

Figure 16.5 presents a histogram of the grades and tonnage of the Mineral Resources. The histogram shows that the Full Moon deposit contains virtually no tonnage below the cut-off grade.

Figure 16.5 – Grade Tonnage Curve



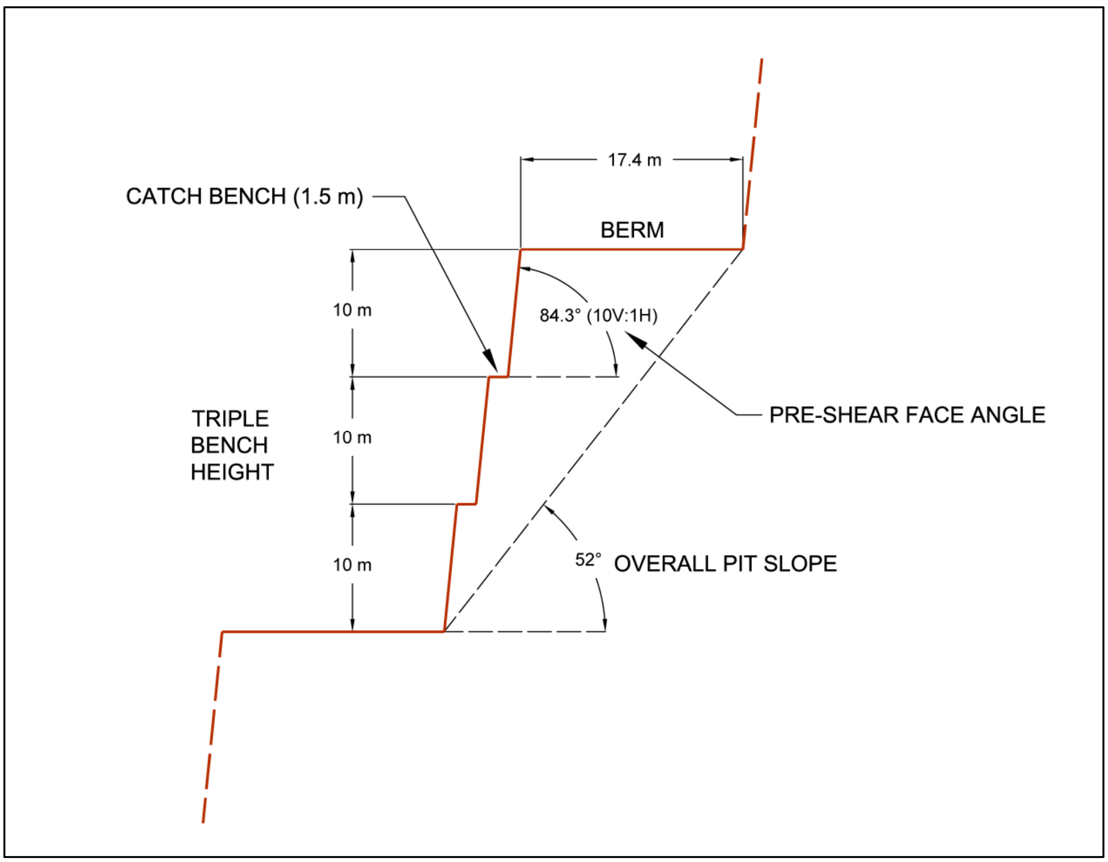
16.5 Mine Design

An open pit which provides a 30 year mine life at the planned production rate was designed using the optimized pit shell as a guideline. The pit design process includes smoothing the pit wall, adding ramps to access the pit bottom and ensuring that the pit can be mined using the selected equipment. The following section provides the parameters that were used for the detailed pit design and presents the results.

16.5.1 Geotechnical Pit Slope Parameters

An inter-ramp angle of 52° was used for the final pit walls. The final pit wall includes a 17.4 m berm for every three (3), 10 m high benches and considers a face angle of 84.3° assuming that pre shearing blasting techniques will be used. This design which is presented in Figure 16.6 is based on Met-Chem's internal database for similar deposits in the region. As was previously mentioned, an offset from Lac Rainy and Lac Eclipse of 100 m was used.

Figure 16.6 – Pit Wall Configuration



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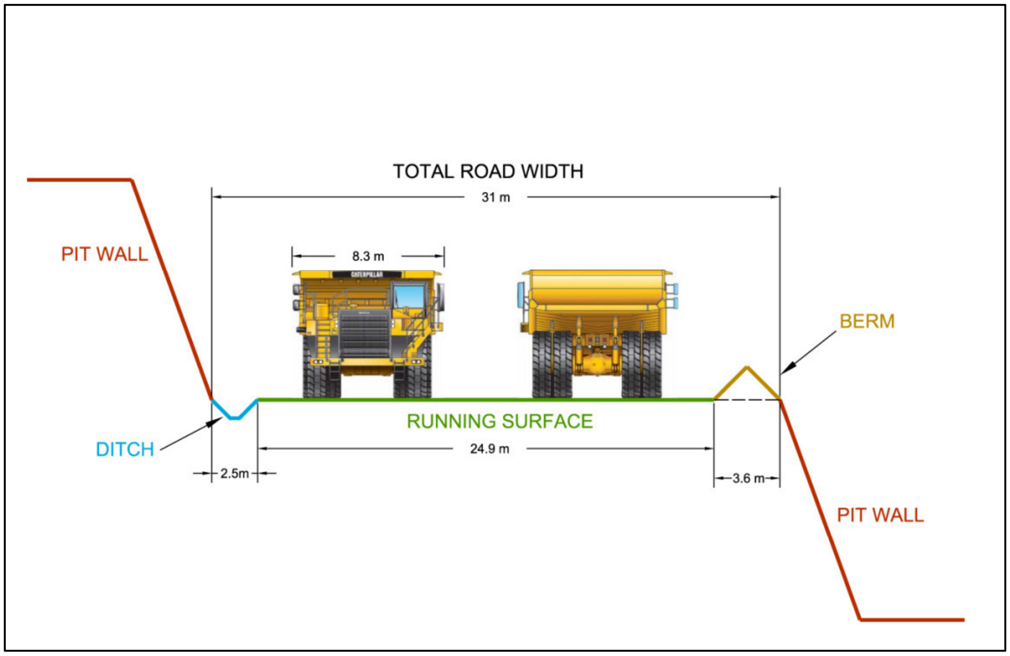
16.5.2 Haul Road Design

The ramps and haul roads were designed with an overall width of 31 m. For double lane traffic, industry practice indicates the running surface width to be a minimum of three (3) times the width of the largest truck. The overall width of a 227-tonne (250 tons) rigid frame haul truck is 8.3 m which results in a running surface of 25 m. The allowance for berms and ditches increases the overall haul road width to 31 m. A maximum ramp grade of 8% was used. Figure 16.7 presents a typical section of the in-pit ramp design.

Figure 16.7 – Ramp Design



16.5.3 Mining Dilution and Mining Recovery

In every mining operation, it is impossible to perfectly separate the ore and waste as a result of the large scale of the mining equipment and the use of drilling and blasting. The two (2) main sources of waste rock are the overburden and the GC unit which as was discussed in Section 16.1 of this report is considered as low grade mineralization. The overburden should be fairly straight forward to strip off the iron formation since it is clearly distinguishable and does not require drilling and blasting. As for the GC unit, since it is quite thin with an average thickness of 5 m within the pit area, Met-Chem has assumed that 50% of the GC unit will be processed and 50% will be sent to the waste dump. An Fe grade of 17% and a weight recovery of 4% were associated to the GC unit that will be blended into the pit tonnages and processed.

16.5.4 Pit Design Results

The 30 year open pit design for the Full Moon deposit is approximately 5,200 m long and 1,800 m wide at surface with a maximum pit depth of 180 m. The total surface area of this pit is roughly 500 ha and the overburden thickness averages 9 m.

There are two (2) access ramps for the pit. The access at the south end is 1 km from the primary crushers and enters the pit at the 540 m elevation. The ramp descends down the western wall of the pit and reaches the final bench which is at the 330 m elevation. The second access ramp which is at the north end of the deposit also enters the pit at the 540 m elevation.

The 30 year open pit includes 1,283 Mt of Indicated Mineral Resources at a Total Fe grade of 30.8% (Weight Recovery of 36.9%) and 327 Mt of Inferred Mineral Resources at a Total Fe grade of 30.7% (Weight Recovery of 37.7%). In order to access these Mineral Resources, 90 Mt of overburden, 9 Mt of Menihek Shale and 54 Mt of low grade mineralization must be mined. This total waste quantity of 153 Mt results in a stripping ratio of 0.1 to 1. Table 16.2 presents a summary of the Mineral Resources within the 30 year open pit for the Full Moon deposit and Table 16.3 presents the Mineral Resources by lithological unit. Figure 16.8 presents a plan view of the pit design.

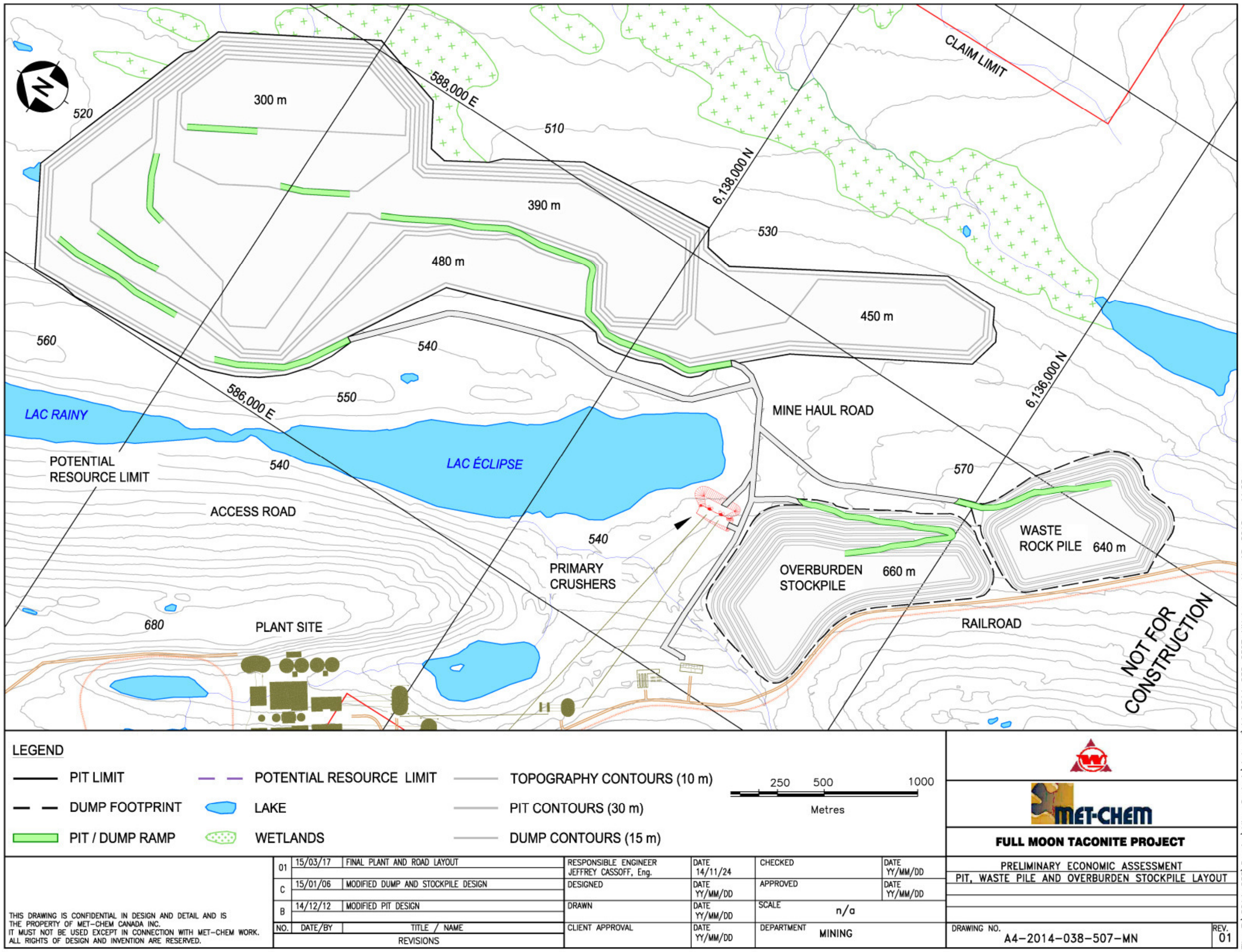
Table 16.2 – Mineral Resources within the 30 Year Open Pit

Description	Resources (Mt)	Total Fe (%)	Weight Recovery (%)
Indicated	1,283	30.8	36.9
Inferred	327	30.7	37.7

Table 16.3 – Mineral Resources within the 30 Year Open Pit (By Unit)

Description	Resources (Mt)	Total Fe (%)	Weight Recovery (%)
JUIF	884	30.4	35.0
URC	210	33.4	38.9
PGC	323	32.0	43.3
LGC	146	31.1	43.3
LRGC	5	28.0	25.6
GC (as dilution)	42	17.0	4.0
Total	**1,610**	**30.8**	**37.1**

Figure 16.8 – 30 Year Open Pit Design



16.5.5 Overburden Stockpile and Waste Rock Pile Design

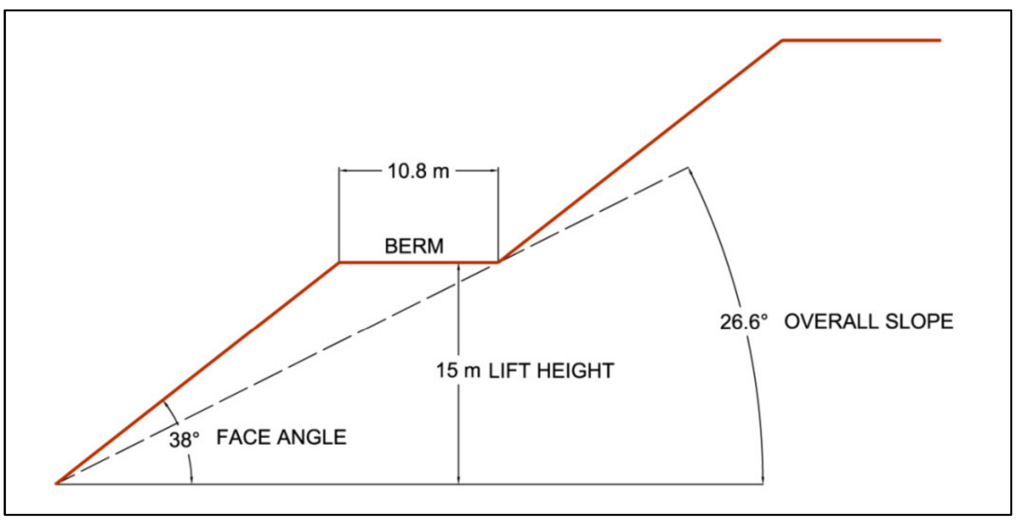
The overburden stockpile and waste rock pile were designed to contain the volumes that are expected to be mined during the first 30 years of the mine life. The two (2) piles are located inside the claims that are controlled by WISCO Century Sunny Lake Mines Limited, and avoid areas that have a potential for Mineral Resource expansion. The piles were designed with an overall slope of 26.6° (2H:1V), which is achieved by placing a 10.8 m wide berm for each 15 m in elevation. The face angle of each lift is 38° which is the angle of repose of the overburden and waste rock. It should be noted that although in pit dumping would shorten the haul distances and reduce the footprints, it was not possible since there is still a considerable amount of resource below the bottom of the 30 year open pit, as was shown in Figure 16.4.

The topsoil and overburden stockpile was designed to the south of the primary crushing facilities. The stockpile has a capacity of 50 Mm^3, a footprint area of 90 ha, a top elevation of 660 m and a maximum height of 100 m. Material that is placed in this stockpile will be used for future reclamation as part of the closure plan.

The waste rock pile was designed to the south of the overburden stockpile. The capacity of the waste rock piles is 26 Mm^3, the footprint area is 56 ha, the top elevation is 640 m and the maximum height is 70 m.

The general layout overburden stockpile and waste rock pile was presented on Figure 16.8 and Figure 16.9 shows a typical section.

Figure 16.9 – Overburden Stockpile and Waste Rock Pile Configuration



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16.6 Mine Planning

A 30 year production schedule (mine plan) was developed for the Project which targets the production of 20 Mt of iron concentrate per year. In order to account for start-up and commissioning, the production in Year 1 has been limited to 17 Mt (85% capacity). The mine plan was established annually for the first ten (10) years of production, followed by four (4), five (5) year periods.

A pre-production period of one (1) year has been included before the start of the operation. This period includes tree clearing, topsoil and overburden removal, mine haul road construction and the development of the pit for production. During pre production, 13.5 Mt of overburden is removed.

The southern part of the pit contains more overburden than the northern part, but the haul distances to the primary crushers and overburden stockpile are considerably shorter from the southern part of the pit. In order to determine the best starting location, Met-Chem carried out a high level trade-off study which showed that the shorter haul distances outweigh the benefits of loading and hauling more overburden up front.

The mine plan was therefore developed with the operations beginning in the southern part of the pit and development starting in the northern part in Year 3. During Years 4 and 5 mining progresses in both the north and south parts of the pit until the tonnages from the south area are depleted. In Year 6 the operations continue in the north part of the pit until Year 30.

Table 16.4 presents the mine production schedule. The table provides the tonnages of each unit that are mined in each period of the mine plan as well as the weight recovery and Total Fe%. The weight recovery throughout the mine plan averages 37.1% and varies from a high of 40.8% between Years 26 to 30 to a low of 35.2% from Years 16 to 20.

The total material mined averages 71 Mtpy and ranges from 50 Mtpy between Years 26 to 30 and reaches a peak of 78 Mtpy per year in Year 2. Figure 16.10 presents a chart showing the tonnages that will be mined each year as well as the weight recovery. The tonnages have been annualized for the five (5) year periods.

Figure 16.11 and Figure 16.12 present the status of the pit, waste rock pile and overburden stockpile at the end of Year 5 and Year 10.

Figure 16.10 – Mine Production Schedule

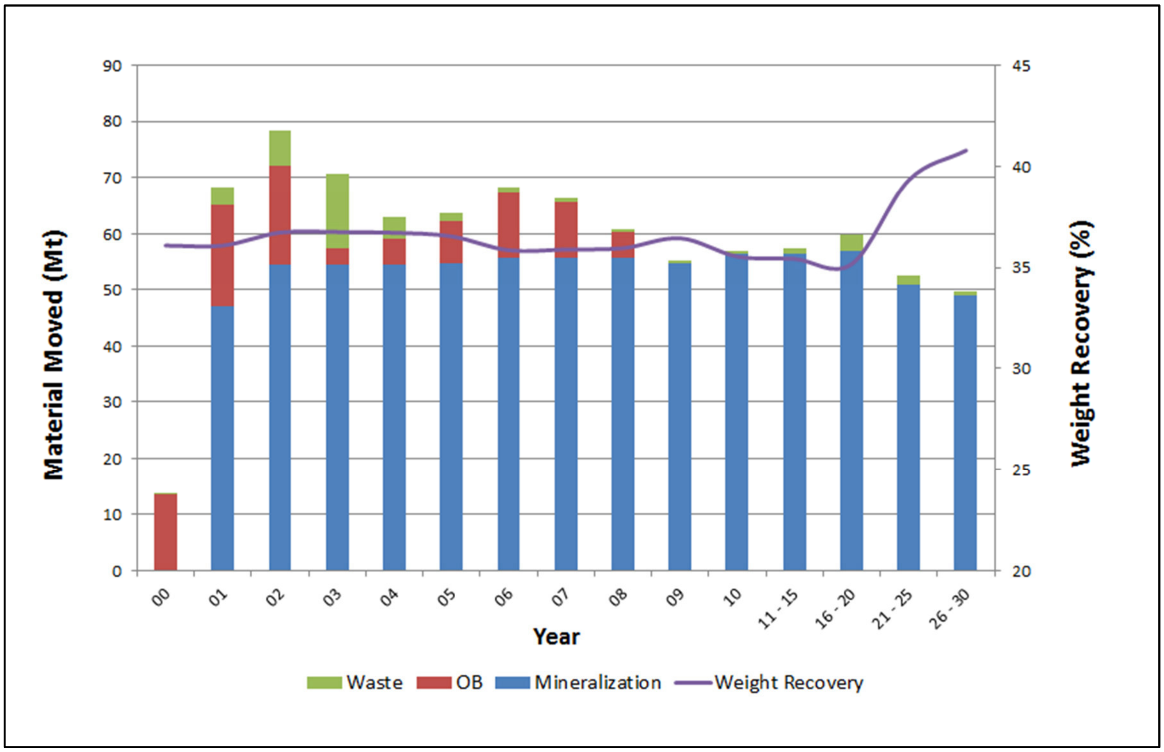


Table 16.4 – Mine Production Schedule

Description	Units	PRE PRO	Year 01	Year 02	Year 03	Year 04	Year 05	Year 06	Year 07	Year 08	Year 09	Year 10	Years 11 - 15	Years 16 - 20	Years 21 - 25	Years 26 - 30	Total
Concentrate	**Mt**	**0.0**	**17.0**	**20.0**	**20.0**	**20.0**	**20.0**	**20.0**	**20.0**	**20.0**	**20.0**	**20.0**	**100.0**	**100.0**	**100.0**	**100.0**	**597**
ROM to Plant	**Mt**	**0.0**	**47.1**	**54.4**	**54.5**	**54.5**	**54.7**	**55.7**	**55.8**	**55.7**	**54.8**	**56.3**	**282.3**	**284.4**	**254.3**	**245.1**	**1,610**
JUIF	Mt	0.0	32.1	32.7	39.0	27.3	35.0	46.9	49.2	49.4	48.2	50.8	251.6	165.3	39.0	16.9	**884**
URC	Mt	0.0	4.9	5.4	3.3	6.4	4.7	4.1	2.1	2.2	1.7	2.3	12.7	51.7	70.8	37.1	**210**
PGC	Mt	0.0	6.8	10.6	6.8	13.6	8.8	3.0	2.7	2.2	2.9	2.1	9.2	40.4	103.6	109.9	**323**
LRC	Mt	0.0	1.8	3.9	4.0	4.3	3.8	0.7	1.2	1.2	1.5	0.4	4.0	10.5	32.1	77.2	**146**
LRGC	Mt	0.0	0.1	0.1	0.4	1.0	1.0	0.0	0.1	0.1	0.0	0.1	0.2	1.2	0.1	1.1	**5**
GC (As Dilution)	Mt	0.0	1.4	1.7	1.1	2.0	1.5	0.9	0.5	0.6	0.4	0.6	4.5	15.2	8.7	2.9	**42**
Weight Recovery	**%**	**0.0**	**36.1**	**36.7**	**36.7**	**36.7**	**36.5**	**35.9**	**35.9**	**36.0**	**36.4**	**35.5**	**35.4**	**35.2**	**39.3**	**40.8**	**37.1**
Total Fe	**%**	**0.0**	**31.0**	**31.1**	**31.3**	**31.0**	**31.2**	**31.2**	**31.0**	**31.2**	**31.4**	**30.9**	**30.9**	**30.8**	**31.8**	**28.9**	**30.8**
Total Waste	**Mt**	**13.5**	**21.1**	**24.0**	**16.2**	**8.4**	**9.1**	**12.6**	**10.4**	**5.2**	**0.4**	**0.6**	**4.5**	**15.2**	**8.7**	**3.1**	**153**
Overburden	Mt	13.5	18.0	17.7	2.9	4.6	7.6	11.6	9.6	4.6	0.0	0.0	0.0	0.0	0.0	0.0	**90**
Menihek Shale	Mt	0.0	0.5	2.7	5.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.2	**9**
Wishart Shale	Mt																**0**
Low Grade	Mt	0.0	2.6	3.7	8.1	3.8	1.5	1.0	0.8	0.6	0.4	0.6	4.5	15.2	8.7	2.9	**54**
Total Material	**Mt**	**13.5**	**68.2**	**78.4**	**70.7**	**63.0**	**63.7**	**68.3**	**66.2**	**60.8**	**55.2**	**56.9**	**286.8**	**299.6**	**263.0**	**248.3**	**1,763**
Stripping Ratio		n/a	0.4	0.4	0.3	0.2	0.2	0.2	0.2	0.1	0.0	0.0	0.0	0.1	0.0	0.0	**0.1**

Figure 16.11 – End of Year 5

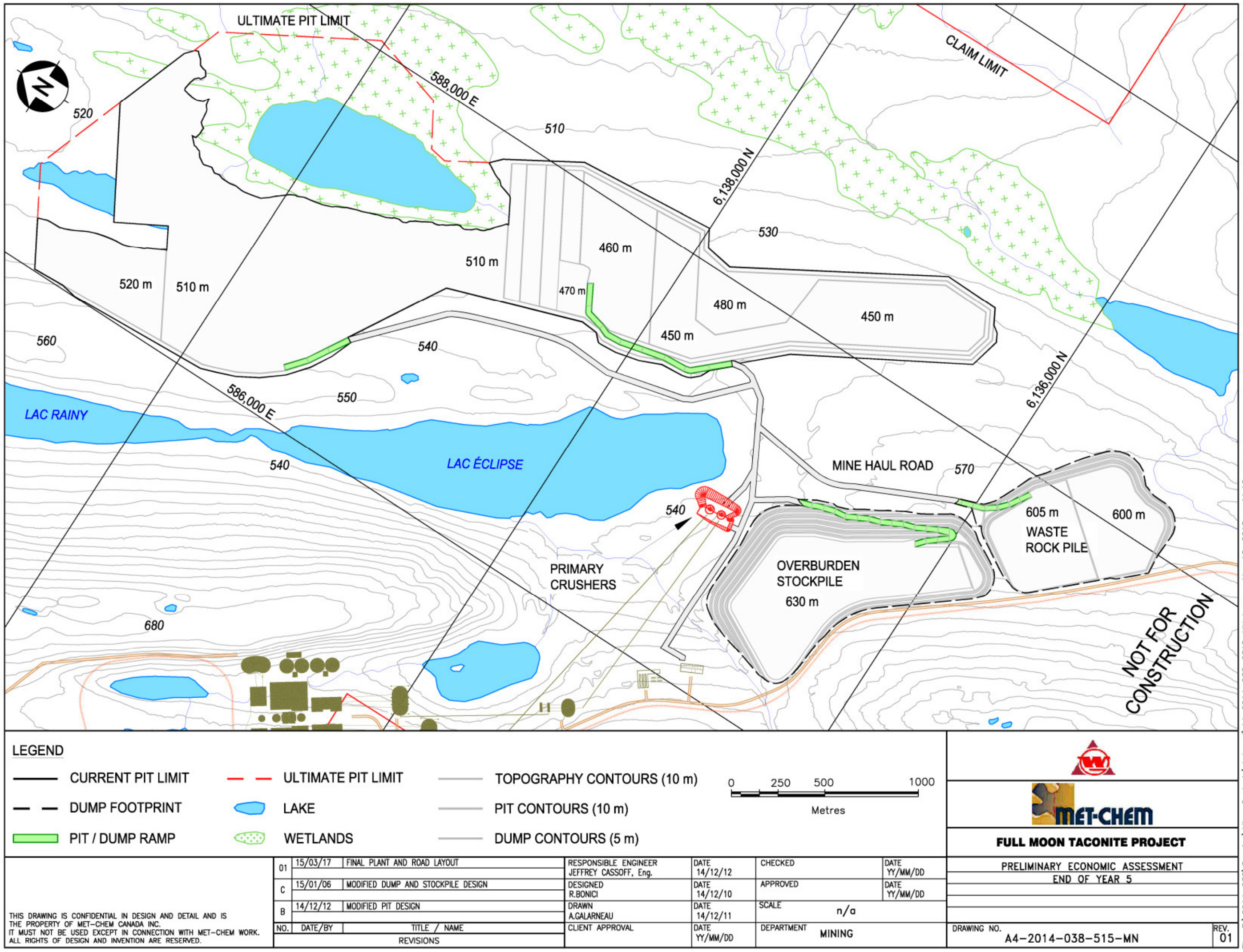
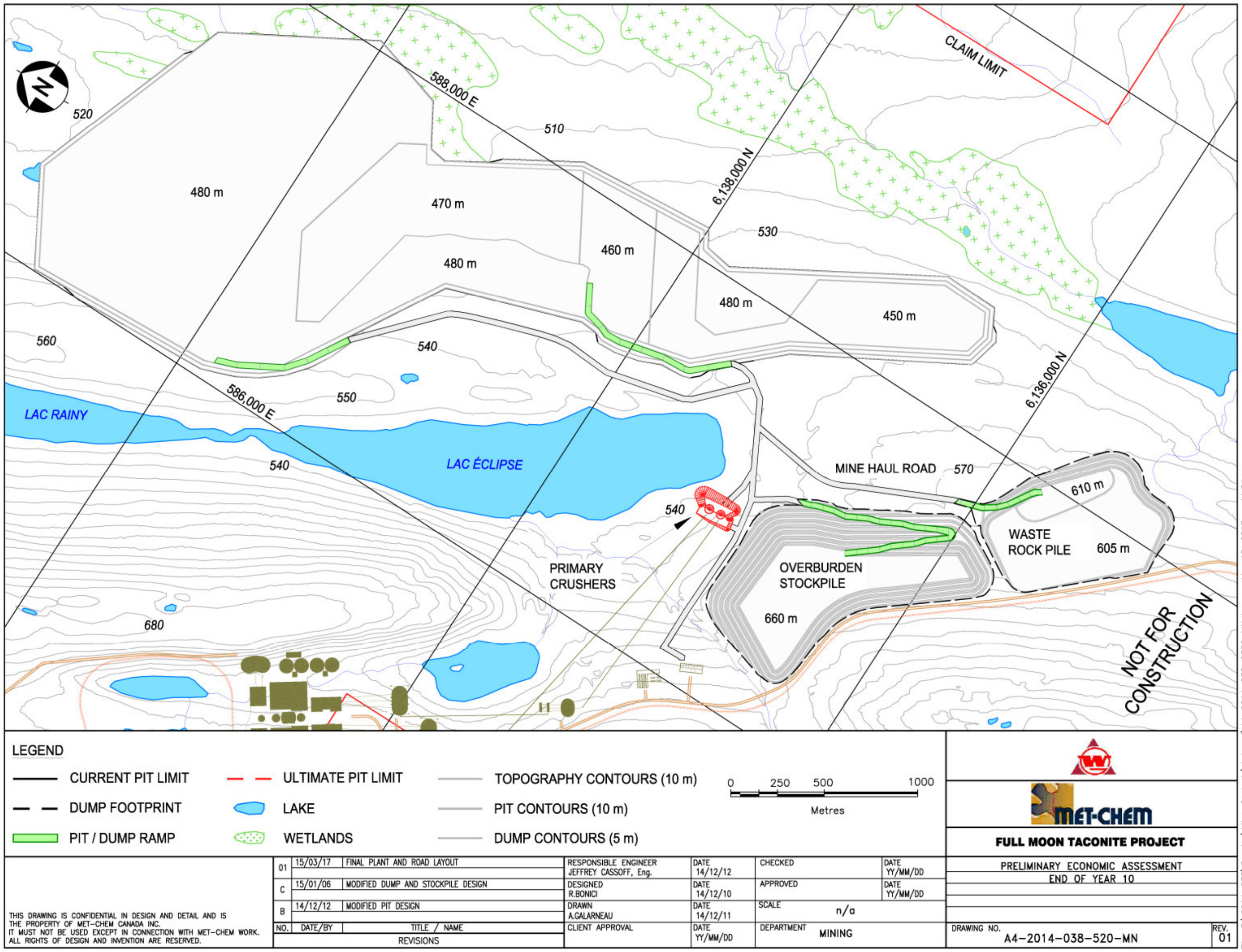


Figure 16.12 – End of Year 10



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16.7 Mine Equipment Fleet

The following section discusses equipment selection and fleet requirements in order to carry out the mine plan. The mine will be operated with an owner fleet which is presented in Table 16.5. The table presents the fleet requirements during peak production and identifies the Caterpillar model to give the reader an appreciation for the size of each machine.

Table 16.5 – Mine Equipment Fleet

Equipment	Typical Model	Description	Units
Major Equipment			
Haul Truck	CAT 793F	Payload – 227 tonne (250 tons)	20
Shovel	CAT 6060FS	Bucket – 26.5 m^3	3
Production Drill	CAT MD6640	311 mm hole (12 ¼")	3
Support Equipment			
Track Dozer	CAT D10T	450 kW (600 hp)	4
Road Grader	CAT 16M	250 kW (335 hp)	4
Wheel Dozer	CAT 844K	520 kW (700 hp)	2
Wheel Loader	CAT 994H	1,100 kW (1,475 hp)	2
Utility Excavator	CAT 336D	200 kW (270 hp)	2
Secondary Drill	CAT MD5150	152 mm hole (6 inch)	1
Cable Reeler	CAT 980K	274 kW (365 hp)	1
Water / Sand Truck	CAT 777	90,000 litres	3
Lighting Plant	Magnum MLT3080	6 kW (8 hp)	8
Service Equipment			
Fuel and Lube Truck	Peterbuilt 365	330 kW (440 hp)	2
Mechanic Truck	Peterbuilt 348	250 kW (335 hp)	3
Boom Truck	Peterbuilt 365	330 kW (440 hp)	2
Tire Handler	n/a	n/a	1
Mobile Crane	RTC8080	Capacity – 75 tonne	2
Lowboy	n/a	n/a	1
Transport Bus	Blue Bird	20 person	3
Pickup Truck	Ford F250	300 kW (400 hp)	20
Dewatering Pump	Godwin HL130M	220 kW (300 hp)	8

16.7.1 Haul Trucks

The haul truck selected for the Project is a rigid frame mining truck with a payload of 227 tonnes (250 tons). This size truck was selected since it matches well with the production requirements and results in a manageable fleet size. The following parameters were used to calculate the number of trucks required to carry out the mine plan.

These parameters result in 5,361 working hours per year for each truck as is presented in Table 16.6:

- Average Mechanical Availability – 85% ;
- Average Utilization – 90% (non-utilized time is accrued when the truck is not operating due to poor weather, blasting, shovel relocation and if no operator is available) ;
- Nominal Payload – 227 tonnes (160 m^3 heaped) ;
- Shift Schedule – Two (2), twelve (12) hour shifts per day, seven (7) days per week ;
- Operational Delays – 80 min/shift (this includes 15 minutes for shift change, 15 minutes for equipment inspection, 40 minutes for lunch and coffee breaks and 10 minutes for fuelling). Fuelling will be carried out once every two (2) shifts for 20 minutes ;
- Job Efficiency – 90% (54 min/h; this represents lost time due to queuing at the shovel and dump as well as interference on the haul road) ;
- Rolling Resistance – 3%.

Table 16.6 – Truck Hours

Description	Hours	Details
Total Hours	8,760	7 days per week, 24 hours per day, 52 weeks per year
Down Mechanically	1,314	15% of total hours
Available	7,446	Total hours minus hours down mechanically
Standby	745	10% of available hours (represents 90% utilization)
Operating	6,701	Available hours minus standby hours
Operating Delays	745	80 min/shift
Net Operating Hours	5,957	Operating hours minus operating delays
Working Hours	**5,361**	**90% of net operating hours (reflects job efficiency)**

Haul routes were generated for each period of the mine plan to calculate the truck requirements. These haul routes were imported in Talpac©, a commercially available truck simulation software package that Met-Chem has validated with mining operations. Talpac© calculated the travel time required for a

227 tonne haul truck to complete each route. Table 16.7 shows the various components of a truck's cycle time. The load time is calculated using a hydraulic shovel with a 26.5 m^3 (63-tonne) bucket as the loading unit. This size shovel which is discussed in the following section can load a 227 tonne haul truck in four (4) passes for waste rock, five (5) passes for mineralized material and six (6) passes for overburden.

Table 16.7 – Truck Cycle Time

Activity	Duration (Sec)
Spot @ Shovel	30
Load Time[1]	120
Travel Time	Calculated by Talpac©
Spot @ Dump	30
Dump Time	30

[1] Four (4) Passes @ 30 sec/pass.

Haul productivities (tonnes per work hour) were calculated for each haul route using the truck payload and cycle time. Table 16.8 shows the cycle time and productivity for the mineralization, overburden and waste haul routes in Year 7 as an example.

Table 16.8 – Truck Productivities (Year 7)

Material	Cycle Times (min)					Productivity	
	Travel	Spot	Load	Dump	Total	Loads/h	t/h
Mineralization	17.65	0.50	2.00	1.00	21.15	2.84	636
Overburden	26.73	0.50	3.00	1.00	31.23	1.92	381
Waste	28.73	0.50	2.50	1.00	32.73	1.83	422

Truck hour requirements were calculated by applying the tonnages hauled to the productivity for each haul route. A fleet of seven (7) trucks is required in pre-production, followed by 15 in Year 1, 18 in Year 2, and reaches a peak of 20 in Year 6.

16.7.2 Shovels

The main loading machine selected for the Project is an electric powered hydraulic shovel with a 26.5 m^3 bucket. This size shovel can handle the 10 m bench height and can load the 227 tonne haul trucks efficiently. The electric model of hydraulic shovel was chosen due to the availability of relatively low cost electric power. Hydraulic shovels were preferred over cable shovels since they can relocate much quicker which will assist with any blending that is required.

Using an 85% mechanical availability and 70% utilization, Met-Chem calculated that two (2) shovels are required in pre-production and a third shovel is added in Year 3. Each shovel will excavate roughly 20 Mtpy which is considered to be a reasonable assumption.

Two (2) large wheel loaders with 35 tonne payloads have been included in the fleet to support the shovels and carry out any run of mine stockpile rehandling that is required.

16.7.3 Drilling and Blasting

Production drilling will be carried out with electric powered rotary drills. Using the following parameters; 85% mechanical availability, 70% utilization and a penetration rate of 25 m/h, Met-Chem calculated that one (1) drill is required in pre production, two (2) during Year 1 and a third drill is added in Year 3. Table 16.9 presents the drilling and blasting parameters.

A secondary track drill has been included in the fleet for development work, establishing pre shear holes for wall stability control, blasting of oversized boulders and additional drilling that the main units will not be able to achieve.

Table 16.9 – Drill and Blast Parameters

Parameter	Units	Value
Bench Height	m	10
Blasthole Diameter	mm	311
Burden	m	8.5
Spacing	m	8.5
Subdrilling	m	1.0
Stemming	m	3.5
Explosives Density	g/cm^3	1.20
Powder Factor	kg/t	0.42

Blasting will be carried out under contract with an explosives supplier who will be responsible for the following services:

- Transportation and storage of explosive manufacturing products and blasting accessories to site ;
- Manufacturing of bulk emulsion explosives ;
- Loading and priming of blastholes.

The explosives supplier will have the following two (2) sites for his operations:

- Explosives Plant – this site includes the storage facility for raw materials, the offices and garages as well as the emulsion plant and pumper truck loading area;
- Explosive Magazines – this site includes the magazines to store the blasting caps, primers, detonation cord and packaged explosives.

The site selection has accounted for the required minimum distances as specified by the Canadian explosives regulations. Approvals and permits are required from the government regulating bodies prior to construction.

For the fabrication of the bulk emulsion, explosives suppliers have proposed transporting ammonium nitrate solution from Sept-Îles to site by rail. The explosives plant facilities will be composed of prebuilt modules that are easily transported and assembled.

In order to support the explosives supplier, the mine operator is required to build and maintain the access road to the two (2) sites and to supply electric power, communications and diesel fuel for the manufacturing of the emulsion as well as the operation of mobile equipment. The mine operator is also required to mobilize and house the contractor's workforce.

Based on the blasting parameters presented in Table 16.9, the amount of explosives required per year is approximately 25 million kg. During full production there will be roughly two (2) blasts per week each producing approximately 500,000 t of material.

The cost for explosives used for the PEA is $0.35/t, which is based on budgetary pricing provided by explosive suppliers. The pricing considers an "all in" down the hole service which means that the supplier will provide the emulsion and accessories as well as the bulk trucks and operators who will load the holes. Blast tie in and detonation will be performed by Full Moon blasting crews. There will be two (2) crews each composed of a blaster and a blaster's helper.

16.8 Mine Dewatering

Surface run-off, rainfall, snowmelt and groundwater will be accumulated in sumps that will be excavated in the pit floor and pumped to collection points at surface. The mining fleet includes eight (8) electric powered centrifugal pumps to account for mine dewatering.

16.9 Mine Manpower

The manpower requirements for the mine have been categorized into Mine Operations, Mine Maintenance and Mine Technical Services. The Mine Operations and Mine Maintenance staff will be comprised of four (4) crews in order to provide 24 h/d coverage. The blasting crew is an exception since they will work on the day shift only. The Mine Technical Services staff will work on the day shift only as well. The total mine manpower requirements during peak production is expected to reach 276 employees. Table 16.10 shows the mine manpower requirement during peak production.

Table 16.10 – Mine Manpower

Parameter	Personnel
Mine Operations	
Mine Manager	1
Mine Superintendent	1
Pit Foreman	12
Equipment Operator	172
Labourer	8
Dispatcher / Trainer	8
Blaster / Blaster Helper	4
Mine Maintenance	
Maintenance Superintendent	1
Maintenance Foreman	8
Maintenance Planner	4
Mechanic / Electrician / Welder	32
Attendant	8
Mine Technical Services	
Mine Technical Superintendent	1
Mining Engineer / Geologist	8
Grade Control Technician	4
Surveyor	4
Total Mine Workforce	**276**

17 Recovery Methods

Section 13 of this report discussed the metallurgical testwork and proposed a processing flowsheet with its expected recovery. In the following sections the design basis and criteria of the processing plant are presented together with the description of each of the processing sections. This information, with the general arrangement drawings, provides the basis for the capital and operating cost estimates.

17.1 Process Plant Design Basis and Design Criteria

Table 17.1 summarizes the general parameters upon which the beneficiation and the pellet plant's design have been based for the Full Moon Project.

17.1.1 Concentrator

The process plant is designed to produce 20.0 Mtpy of high silica content (4.5 %) concentrate over a 30 year mine life. The ROM is calculated based on a magnetite plant weight recovery of 27 % and a hematite plant weight recovery of 9.2 %. The selected lower hematite plant weight recovery compared to the one presented in Section 13 is due to the dewatering losses and the fact that the finisher tails, which are too dilute, are not processed by the hematite plant.

A design factor of 20 % is applied on nominal requirements to ensure that the process equipment has enough capacity to take care of the expected feed variation.

The production of low silica content (<1.5 %) concentrate leads to a weight recovery loss of 3 % and a production of 18.3 Mtpy of concentrate.

The process plant design is based on testwork performed to date (Section 13), knowledge acquired in the processing of magnetite-rich ores in the Iron Range in Northern USA and project developments in nearby properties.

17.1.2 Pellet Plant

The pellet plant is designed to produce 17.0 Mtpy of fired pellets in two (2) completely identical and independent processing lines. The production rate is based on induration machines designed to process magnetic concentrates. When fed by a blend of magnetite and hematite concentrates, the pellet plant production rate is expected to be lower or coke breeze addition may be required to maintain the production rate; this will have to be confirmed by further testwork.

The pellet plant processes the iron concentrate as received from the concentrator without any beneficiation plant to reduce impurity levels. There is no tailings stream at the pellet plant and no process water effluent is expected.

The pellet plant is designed to offer sufficient flexibility to produce many types of pellets from the low and high silica concentrates produced at the concentrator. The design pellet mix is:

- Direct Reduction Iron pellet with low silica and additives content;
- High Silica Flux pellet with high silica and additives content.

The pellet plant also has the capacity to produce all intermediate pellet types (Low or High Silica Acid ("LSA" or "HSA"), Low Silica Fluxed ("LSF"), etc.), but their productions are not described. All design documents show either a full production year of DR or a full production year of HSF pellets.

Table 17.1 – Design Criteria Summary

Parameters	Value	Units
Concentrator - General		
Operating Schedule		
Operating hours per day	24	h
Annual operating days	365	days/y
Equipment utilization - plant	90	%
Annual operating time - plant	7,884	h/y
Equipment utilization - crushing	65	%
Annual operating time - crushing	5,694	h/y
Design		
Design factor	20.0	%
Material Characteristics		
Plant Feed		
Iron (Fe) grade	30.8	%
Magnetite (Fe_3O_4) grade	27.0	%
Solids %	97.0	%
Plant Product		
High silica concentrate		
Silica (SiO_2) grade	4.5	%
Low silica concentrate		
Silica (SiO_2) grade	1.5	%
Plant Production - Nominal		
Crushing		
Run of mine (Dry)	55.2	Mtpy
Run of mine (Dry)	9,693	t/h
Concentrator		
Concentrator solids feed	55.2	Mtpy
Concentrator solids feed rate	7,001	t/h
Plant High Silica Concentrate		
Plant solids concentrate production	20.0	Mtpy
Plant solids concentrate production rate	2,537	t/h
Plant weight recovery	36.2	%
Magnetite plant weight recovery	27.0	%
Hematite plant weight recovery	9.2	%

Parameters	Value	Units
Plant Low Silica Concentrate		
Plant solids concentrate production	18.3	Mtpy
Plant solids concentrate production rate	2,325	t/h
Plant weight recovery	33.2	%
Magnetite plant weight recovery	24.7	%
Hematite plant weight recovery	8.5	%
Pellet Plant Operation		
General		
Number of lines	2	lines
Pellet plant availability	330	days/y
Total pellet production	17	Mtpy
Pellet plant HSF pellets		
Concentrate type required	High Silica	-
Concentrate production required	15.8	Mtpy
Concentrate required for pellet production	1,990	t/h
Pellet plant DR pellets		
Concentrate type required	Low Silica	-
Concentrate production required	16.3	Mtpy
Concentrate required for pellet production	2,052	t/h

17.2 Process Flowsheet

A simplified block flow diagram of the concentrator is presented in Figure 17.1 while Figure 17.2 depicts a simplified flowsheet of the pellet plant. The equipment list is based on the flowsheet diagrams and the equipment sizing is based on the mass balance.

Figure 17.1 – Concentrator Block Flow Diagram

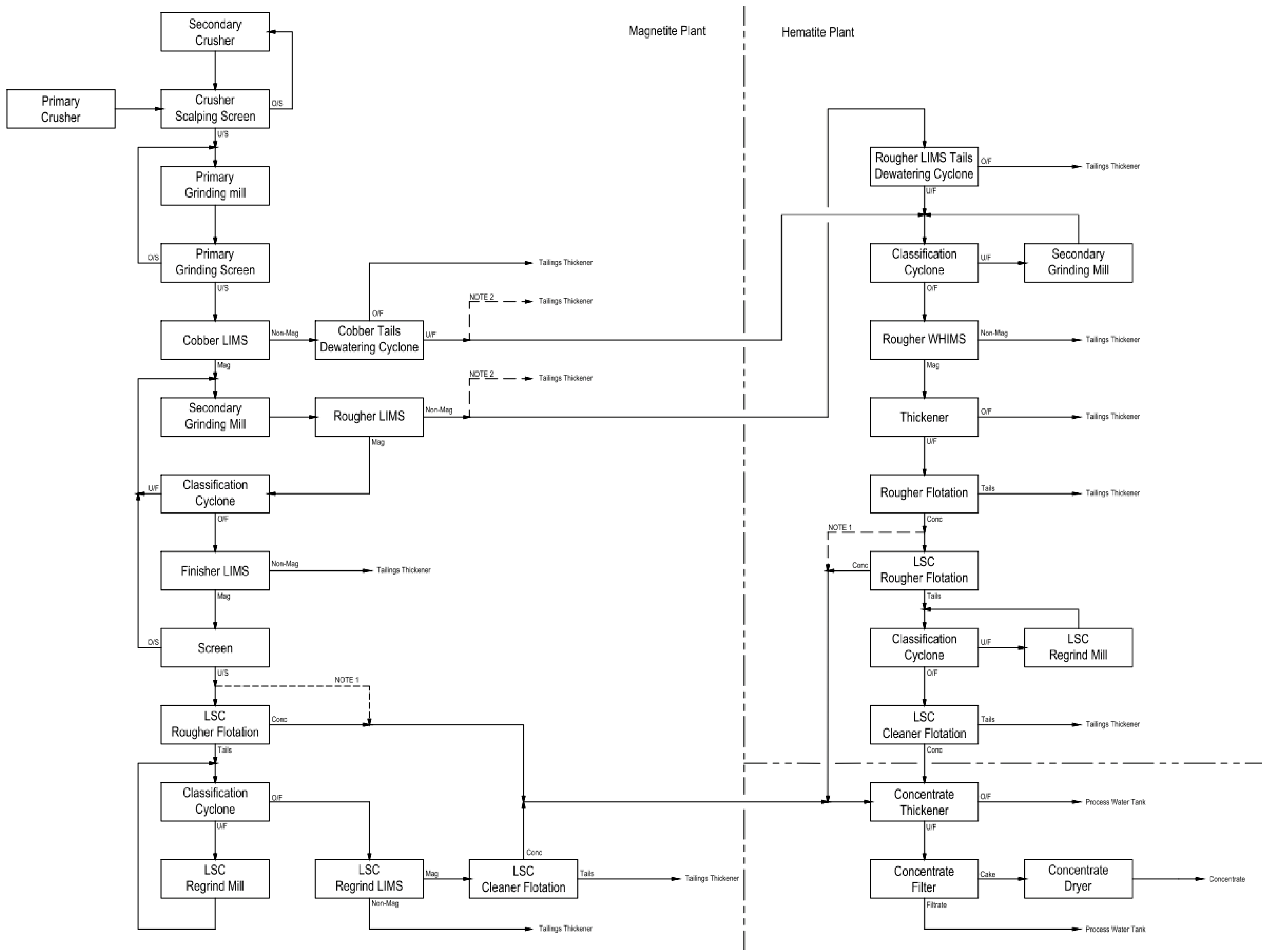
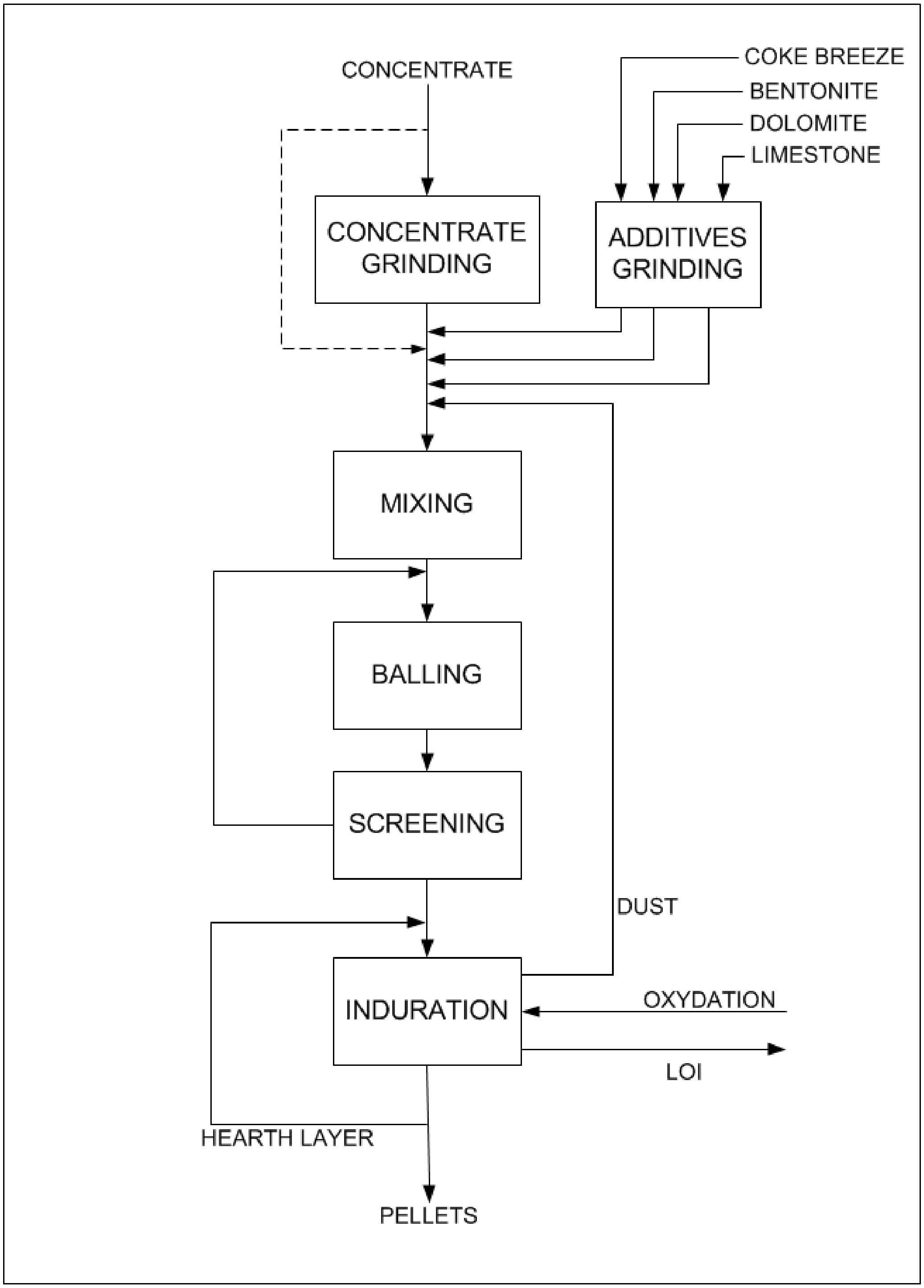


Figure 17.2 – Pellet Plant Simplified Flowsheet



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17.3 Mass and Water Balance

A mass balance summary for the concentrator is presented in Table 17.2 and the pellet plant mass balance summary is shown in Table 17.3.

Table 17.2 – Concentrator Mass Balance (Nominal) for Low Silica Concentrate Production

Stream Name	Solids Tonnage t/h	Slurry Tonnage t/h	% Solids w/w	Slurry Flowrate m^3/h	Solids SG	Slurry SG
Crushing and Stockpiling						
ROM	9,693	9,993	97	-	3.4	-
Crusher Scalping Screen U/S	9,693	9,993	97	-	3.4	-
Crusher Scalping Screen O/S	8,821	9,094	97	-	3.4	-
Primary Grinding						
Primary Grinding Mill Fresh Feed	7,001	7,217	97	-	3.4	-
Primary Grinding Mill Product	15,191	15,839	96	-	3.4	-
Primary Grinding Screen O/S	8,191	8,622	95	-	3.4	-
Primary Grinding Screen U/S	7,001	7,817	90	-	3.4	-
Magnetite Plant Regrinding & Magnetite Separation						
Cobber LIMS Feed	7,001	17,502	40	12,560	3.4	1.4
Cobber LIMS Concentrate	4,358	6,705	65	3,537	3.7	1.9
Cobber LIMS Tails	2,643	11,276	23	9,502	3.0	1.2
Total Ball Mill Feed	15,253	38,133	40	26,628	4.1	1.4
Rougher LIMS Concentrate	13,293	20,451	65	10,262	4.3	2.0
Rougher LIMS Tails	1,960	19,143	10	17,826	3.0	1.1
Magnetite Plant Classification Cyclone O/F	3,189	15,105	21	12,629	4.5	1.2
Cobber LIMS Tails Dewatering Cyclones U/F	2,497	5,549	45	3,873	3.0	1.4
Cobber LIMS Tails Dewatering Cyclones O/F	145	5,726	3	5,629	3.0	1.0
Rougher LIMS Tails Dewatering Cyclone U/F	1,637	5,456	30	4,357	3.0	1.3
Rougher LIMS Tails Dewatering Cyclone O/F	323	13,687	2	13,470	3.0	1.0
Magnetite Plant Finisher LIMS & Classification						
Finisher LIMS Concentrate	2,681	4,875	55	2,741	4.9	1.8
Finisher LIMS Tails	508	11,275	5	10,933	3.1	1.0
Magnetite Plant Screen U/S	1,890	3,825	49	2,316	5.0	1.7
Magnetite Plant Screen O/S	791	1,318	60	693	4.8	1.9
Magnetite Plant LSC Flotation						
Magnetite Plant LSC Rougher Flotation Sink	1,207	3,019	40	2,046	5.1	1.5
Magnetite Plant LSC Rougher Flotation Float	683	1,707	40	1,170	4.7	1.5
Magnetite Plant LSC Cyclone Feed	1,707	4,267	40	2,926	4.7	1.5
Magnetite Plant LSC Cyclone UF	1,024	1,463	70	658	4.7	2.2
Magnetite Plant LSC Cyclone OF	683	2,804	24	2,268	4.7	1.2
Magnetite Plant LSC LIMS	626	963	65	465	4.9	2.1

Stream Name	Solids Tonnage t/h	Slurry Tonnage t/h	% Solids w/w	Slurry Flowrate m³/h	Solids SG	Slurry SG
Magnetite Plant LSC LIMS Tails	57	1,910	3	1,871	3.2	1.0
Magnetite Plant LSC Cleaner Flotation Sink	522	1,306	40	886	5.1	1.5
Magnetite Plant LSC Cleaner Flotation Float	104	259	40	181	4.0	1.4
Magnetite Plant LSC Flotation	1,730	4,324	40	2,932	5.1	1.5
Hematite Plant Regrinding & Magnetic Separation						
Hematite Plant Classification Cyclone Feed	14,469	28,938	50	19,224	3.0	1.5
Hematite Plant Classification Cyclone U/F	10,335	14,764	70	7,826	3.0	1.9
Hematite Plant Classification Cyclone O/F	4,134	14,174	29	11,398	3.0	1.2
Rougher WHIMS Tails	2,199	11,898	18	10,483	2.8	1.1
Rougher WHIMS Concentrate	1,935	14,540	13	13,179	3.4	1.1
Hematite Plant Thickener U/F	1,853	3,707	50	2,397	3.4	1.5
Hematite Plant Thickener O/F	81	10,833	1	10,782	2.7	1.0
Hematite Plant RGH Flotation						
Hematite Plant Rougher Flotation Feed	1,853	4,633	40	3,324	3.4	1.4
Hematite Plant Rougher Flotation Cells Float	1,207	3,017	40	2,221	2.9	1.4
Hematite Plant Rougher Flotation Cells Sink	647	1,617	40	1,103	4.8	1.5
Hematite Plant LSC Flotation						
Hematite Plant LSC Rougher Flotation Sink	414	1,036	40	704	5.0	1.5
Hematite Plant LSC Rougher Flotation Float	232	581	40	399	4.6	1.5
Hematite Plant LSC Cyclone Feed	581	1,452	40	997	4.6	1.5
Hematite Plant LSC Cyclone UF	348	498	70	225	4.6	2.2
Hematite Plant LSC Cyclone OF	232	954	24	772	4.6	1.2
Hematite Plant LSC Cleaner Flotation Sink	181	826	22	682	4.9	1.2
Hematite Plant LSC Cleaner Flotation Tails	51	128	40	91	3.7	1.4
Hematite Plant LSC Flotation	595	1,861	32	1,386	5.0	1.3
Concentrate Thickening & Handling						
Final Concentrate Dewatering Thickener O/F	0	2,311	0	2,311	-	1.0
Final Concentrate Dewatering Thickener U/F	2,325	3,875	60	2,008	5.1	1.9
Dried Concentrate	2,325	2,372	98	-	-	-
Final Concentrate Drum Filter Filtrate	0	1,263	0	1,263	-	1.0
Dryer Vent	0	240	0	240	0.0	1.0
Tailings						
Tailings Thickener O/F	0	50,941	0	50,941	-	1.0
Tailings Thickener U/F	4,676	7,793	60	4,718	2.9	1.7

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Table 17.3 – Pellet Plant Mass Balance – High Silica Fluxed and Direct Reduction Pellets

Stream Description	HSF Pellet Solids t/h	DR Pellet Solids t/h
Concentrate Grinding		
Fresh Concentrate to HPGR	995	1,026
HPGR Discharge Recirculation	-	-
HPGR Discharge to Ground Concentrate Bin	995	1,026
Mixer		
Ground Concentrate to Mixer	995	1,026
Ground Bentonite to Mixer	6	6
Ground Limestone to Mixer	27	3
Ground Dolomite to Mixer	57	9
Ground Coke Breeze to Mixer	-	-
Washdown Thickener Underflow to Mixer	8	7
Mixer Discharge	1,093	1,053
Balling Disc Feed Hopper		
Mixer Discharge	1,093	1,053
Roller Decks Recirculation	366	353
Balling Disc Feed	1,459	1,405
Balling Disc		
Balling Disc Feed Hopper Discharge	1,459	1,405
Balling Disc Roller Deck Feed	1,459	1,405
Balling Disc Roller Deck Screen		
Balling Disc Roller Deck Feed	1,459	1,405
Balling Disc Roller Deck O/S and U/S	292	281
Balling Disc Roller Deck Product	1,168	1,124
Induration Machine Roller Deck Screen		
Balling Disc Roller Deck Product	1,168	1,124
Induration Machine Roller Deck U/S	74	72
Induration Machine Roller Deck Product	1,093	1,053
Induration Machine		
Induration Machine Roller Deck Product	1,093	1,053
Hearth Layer Bin Discharge	270	270
Magnetite Oxidation	33	36
Lost On Ignition	46	8
Dust to Washdown Thickener	8	7
Fired Pellets	1,343	1,343
Segregation Bin		
Fired Pellets	1,343	1,343
Pellets to Stockpile	1,073	1,073
Pellets to Hearth Layer Bin	270	270

17.4 Process Description

The following process description gives an overview of the beneficiation and pellet plants circuit based on the testwork and above presented design criteria. The process description is divided into the following sections:

- Dry processing (crushing, ore storage, HPGR and screening);
- Magnetite plant;
- Hematite plant;
- Concentrate dewatering and handling;
- Tailings thickening and water management;
- Services and reagents; and
- Pellet plant.

17.4.1 Dry Processing

17.4.1.1 Crushing and Stockpiling

a. Primary Crushing

There are two (2) crushing lines. Run-of-mine is hauled to the two (2) primary gyratory crushers (1,200 kW each) by trucks. Each primary crusher operates independently. Two (2) conveyors transport the -200 mm crushed mineralized material to the buffer stockpile prior to the secondary crushing operation.

Auxiliary equipment like dust collectors, pneumatic rock breakers, overhead cranes and monorails will support the operation and maintenance of the primary crushers and related equipment.

b. Secondary Crushing and Ore Storage

From the buffer stockpile after primary crushing, the mineralized material is reclaimed by apron feeders and is conveyed to the crusher scalping screens bin. The six (6) 10'x 20' double-deck crusher scalping screens, (one (1) for each secondary crusher) are fed from the bin from a respective chute. Oversize material from the screens is processed through a chute to the respective crusher. There will be six (6) secondary cone crushers (895 kW each). The secondary cone crusher product is conveyed back to the secondary crusher feed bin. The screens' undersize, -63 mm, is collected on a single conveyor and sent to the 85,000 t crushed stockpile.

Auxiliary equipment like dust collectors, overhead cranes and monorails will support the operation and maintenance of the secondary crushers, screens and related equipment.

17.4.1.2 Primary Grinding

The crushed mineralized material is reclaimed from the crushed stockpile by apron feeders onto the crushed stockpile conveyor. The latter feeds the primary grinding mills shuttle conveyor that takes the mineralized material to the primary grinding mills feed bins. There are six (6) HPGR mills, 2.2 m in diameter x 2 m wide, as primary grinding mills, each one feeding an independent concentrator production line in the magnetite plant. The discharge from each HPGR is conveyed to four (4) dedicated 8'x 24' double deck banana screens with 3 mm openings. The screening is wet and the undersize slurry from each set of four (4) screens is collected in a pumpbox, and pumped to a six (6)-way distributor ahead of the cobber magnetic separators. The oversize is collected on a conveyor and recirculated back to the primary grinding mills shuttle conveyor. Sending the oversize material back to the primary grinding mills' shuttle conveyor allows the oversize material to be redistributed evenly on all six (6) lines. It prevents the lines from each having a widely different circulating load and throughput.

17.4.2 Magnetite Plant

17.4.2.1 Cobber Magnetic Separation

The magnetite plant consists of six (6) independent process lines using three (3) steps of magnetic separation. The primary grinding mills screen undersize from each line is pumped to a six (6)-way distributor to feed the cobber LIMS. There will be a total of six (6) concurrent single 4'x10' cobber LIMS per processing line, whose role is to separate the magnetic iron from the non-magnetic iron and the gangue. The magnetic concentrate flows to the magnetite plant regrind mill and the non-magnetic outlet is pumped to the cobber tails dewatering cyclones. The cobber LIMS tails dewatering cyclones overflow is pumped to the tailings thickener and the underflow ("U/F") is pumped to the hematite plant classification cyclones. In a case where the hematite plant is not running, the underflow, having a 3 mm top size, will be pumped directly to the tailings thickener U/F pumpbox.

17.4.2.2 Regrind Circuit

a. Magnetite Plant Secondary Grinding Mills

There are twelve (12) magnetite plant regrind mills, two (2) for each process line. Each one is the entry point of a closed circuit formed with the rougher LIMS and the magnetite plant classification cyclones. The rougher magnetic separators role is to remove the non-magnetic iron and gangue material as soon as it is liberated and as a consequence to save on grinding energy.

The regrind mills are also part of and the entry point of a larger closed circuit that includes all equipment in the magnetite plant from the regrind mills to the magnetite plant screens aiming at ensuring all particles have reached the liberation size or smaller. The magnetite plant screens oversize material is returned to the magnetite plant regrind mills. The magnetite plant regrind mills are ball mills of 7.3 m in diameter, and 11.3 m in length with a 12,000 kW motor.

b. Magnetite Plant Rougher LIMS

The magnetite plant regrind mills product is pumped to the rougher LIMS distributors, which feed the rougher low intensity magnetic separators. There are 15 counter-rotation single 4’x10’ drum rougher LIMS per regrind mill. The magnetic concentrate from the rougher LIMS is pumped to the magnetite plant classification cyclones and the non-magnetic outlet is pumped to the rougher LIMS tails dewatering cyclones. The overflow from these cyclones is pumped to the tailings thickeners and the underflow is pumped to the hematite plant classification cyclones. In case the hematite plant is not running, the rougher tails will be pumped directly to the tailings thickener.

c. Magnetite Plant Classification Cyclones

At the magnetic plant classification cyclones, +45 µm particles at the cyclone underflow are recirculated back to the regrind mill while the -45 µm particles exit the cyclone overflow to be pumped to the magnetite plant finisher LIMS.

17.4.2.3 Finisher Magnetic Separation

There are ten (10) counter-current double 4’x10’ drum finisher LIMS per line. The finisher LIMS magnetic concentrate is pumped to the magnetite plant screen and the non-magnetic outlet is pumped to the tailings thickeners.

The finisher LIMS concentrate is screened at 45 µm and the oversize material is pumped back to the regrind mill. There will be 24 magnetite plant screens per line. Their role is to remove the coarse particles and improve the final concentrate grade. The magnetite plant screens are five (5) deck stack sizers™. Stack sizer™ screens were chosen because they are efficient fine cut-size screens designed to reduce the required footprint.

When High Silica Concentrate is produced, magnetite plant screen undersize is pumped directly to the concentrate thickeners.

17.4.2.4 LSC Flotation Circuit

To produce a Low Silica Concentrate, magnetite plant screen undersize is pumped to the magnetite plant flotation circuit. The LSC flotation circuit has four (4) independent processing lines.

Rougher flotation is performed in a single-pass open circuit. The feed from the magnetic concentration is first sent to conditioning tanks, where caustic starch is added as a depressant for magnetite. The amine collector is added in various stages along the rougher flotation cells. Each line comprises five (5) rougher cells. The rougher sink (iron concentrate) is pumped to the concentrate dewatering thickeners while the froth is pumped to the regrind circuit.

The froth from the rougher flotation cells is pumped to hydrocyclones for classification. For each line, the coarse fraction is reground in a 2,240 kW tower mill in closed circuit with the cyclones. The cyclone overflow at 100%-25 µm is fed to four (4) counter-current double 4'x10' drum LIMS in parallel for recovery of the magnetite and for density control. The magnetic concentrate is pumped to the cleaner flotation circuit. The non-magnetic material is pumped to the tailings thickener.

Each cleaner flotation line comprises five (5) cells. The cleaner concentrate (sink) is pumped with the rougher concentrate to the concentrate dewatering thickener while the cleaner froth is pumped with the non-magnetic tails to the tailings thickener.

17.4.3 Hematite Plant

17.4.3.1 Regrind Circuit

As with the magnetite plant, the hematite plant consists of six (6) independent process lines.

The dewatered magnetic cobber and rougher tailings are reclaimed in the hematite plant regrind mill pump box, from where they are pumped to the hematite plant classification cyclones. The hematite plant classification cyclones play a role similar to the ones in the magnetite plant. They function in closed circuit with the hematite plant regrind mills and send the particles of liberated size from the overflow to the rougher WHIMS and the oversize particles in the underflow to the hematite regrind mill.

There is one (1) hematite plant regrind mill for each line. Each one is 7.9 m in diameter x 12.3 m in length with a 16,000 kW motor that reduces the particles size down to 45 µm. The regrind mills product is pumped back to the hematite plant classification cyclones.

17.4.3.2 Rougher WHIMS and Flotation Circuit

The rougher wet high intensity magnetic separators are fed by gravity by the hematite plant classification cyclones overflow and they recover hematite from gangue material in the hematite classification cyclones overflow. There are four (4) rougher WHIMS SLon-3000 per line. The rougher WHIMS magnetic concentrate is pumped to the hematite plant desliming thickeners and the non-magnetic outlet is pumped to the tailings thickeners.

The desliming thickeners are used to dewater the dilute magnetite concentrate from the rougher WHIMS but also to remove the very fine particles generated that could be detrimental to the downstream flotation. There are six (6) 22 m diameter desliming thickeners, one for each line. The desliming thickeners underflow is pumped to the flotation circuit while the overflow is pumped to the tailings thickeners.

Rougher flotation is performed in a single-pass open circuit. The feed from the desliming thickeners is first sent to conditioning tanks, where caustic starch is added as a depressant for hematite. The amine and phosphoric acid collectors are added in various stages along the rougher flotation cells. Each line comprises eight (8) rougher cells. The rougher sink (iron concentrate) is pumped to the hematite LSC flotation circuit while the froth is pumped to the tailings thickeners.

When High Silica Concentrate is produced, rougher flotation concentrate is pumped directly to the concentrate thickeners.

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17.4.3.3 LSC Flotation Circuit

To produce a Low Silica Concentrate, rougher flotation concentrate is pumped to the hematite LSC flotation circuit. The LSC flotation circuit has two (2) independent processing lines.

LSC rougher flotation is performed in a single-pass open circuit. The feed from hematite rougher flotation is first sent to conditioning tanks, where caustic starch is added as a depressant for hematite. The amine and phosphoric acid collectors are added in various stages along the LSC rougher flotation cells. Each line comprises five (5) rougher cells. The rougher sink (iron concentrate) is pumped to the concentrate dewatering thickeners while the froth is pumped to the regrind circuit.

The froth from LSC rougher flotation cells is pumped to hydrocyclones for classification. For each line, the coarse fraction is reground in a 2,240 kW tower mill in closed circuit with the cyclones. The cyclone overflow at a 100% -25 µm is fed to the cleaner flotation circuit.

Each cleaner flotation line comprises five (5) cells. The cleaner concentrate (sink) is pumped with the rougher concentrate to the concentrate dewatering thickener while the cleaner froth is pumped to the tailings thickener.

17.4.4 Concentrate Thickening and Handling

In a case where the LSC is produced, the concentrate thickeners are fed with the rougher and cleaner concentrates of the magnetite and hematite LSC flotation circuits. In a case where the HSC is produced, the concentrate thickeners are fed with the magnetite plant screen undersize and the hematite rougher flotation concentrate. There are two (2) 30 m diameter magnetite concentrate thickeners and two (2) 18 m diameter hematite concentrate thickeners. Thickeners underflow, at 60 % solids, is pumped to the concentrate drum filters while overflow is pumped to the process water tank. Flocculant requirement to secure clear water at the overflow will be confirmed by future testwork.

In total 24 drum filters are used to reduce the concentrate moisture of the concentrate thickener underflow down to 11 %. Dewatered concentrate is conveyed to the concentrate dryers. There are eight (8) low attrition flash dryers required to reduce the final concentrate moisture below 1 %. Dried concentrate is conveyed to the concentrate loading area.

17.4.5 Tailings Thickener and Water System

The tailings thickeners receive the pumped finisher tails, dewatering equipment overflows and flotation tails. Underflow from the tailings thickener and the cobber tails dewatering cyclones are pumped to the tailings pond. There are four (4) 80 m diameter dewatering thickeners.

Water is returned from the tailings pond to the process water tank to be reused in the concentrator. There is one process water tank and one fresh water tank at the concentrator. Water returning from the tailings pond is sufficient for all process needs. Fresh water is used only for service water, gland seal water and reagents preparation.

17.4.6 Services and Reagents

The concentrator uses the following reagents:

- Silica collector (amine collector);
- Carbonate collector (phosphoric acid collector);
- Depressant (starch);
- Frother;
- pH modifier (caustic);
- Flocculant.

The silica and carbonate collectors and the caustic are expected to be delivered by trucks in liquid form. Each collector system comprises a storage tank and a distribution pump.

Depressant is delivered in bags and dumped into a feed hopper and conveyed using a screw feeder to an agitated mixing tank. Fresh water is added to the mixing tank. The prepared depressant is transferred to a distribution tank and distributed to the flotation circuits by metering pumps.

Frother is expected to be delivered in 1 m^3 bulk containers. Dosing pumps control the frother distribution to the flotation circuits.

Flocculant is used in the tailings thickener. The flocculant is delivered in bags and dumped into a feed hopper and conveyed using a screw feeder to an agitated mixing tank. Fresh water is added to the mixing tank. The diluted flocculant is transferred to a distribution tank and is distributed to the thickener feed slurries by metering pumps.

Each reagent storage and handling area has a sump pump, safety showers and eye wash stations.

17.4.7 Pellet Plant

17.4.7.1 Concentrate Reception

Iron concentrate produced by the concentrator is transported by train to a car-dumper in the Sept-île shipping yard from where concentrate is conveyed to the concentrate stockpiles in the pellet plant yard. There are two (2) concentrate stockpiles, one (1) for the low silica concentrate and one (1) for the high silica concentrate.

The description below is for one (1) pellet plant line. There are two (2) identical processing lines in the pellet plant.

17.4.7.2 Concentrate Grinding

The concentrate is reclaimed from the concentrate stockpiles to feed the Coarse Concentrate Silo. Concentrate is reground in the concentrate grinding mill, which is a High Pressure Grinding Rolls mill, in order to control the concentrate blaine in the appropriate range for balling. The HPGR has two (2) motors of 1,330 kW each. There is no provision for recirculation. When allowed by the coarse concentrate blaine index, a portion of the concentrate may by-pass the grinding mill and directly feed the ground concentrate bin. The ground concentrate feeds the ground concentrate bin. The concentrate bins are designed to account for unscheduled short maintenance in the reclaiming-grinding area.

17.4.7.3 Additive Storage and Grinding

Limestone and dolomite will be used as flux agents, either alone or in combination to meet the required pellet quality. Additives (dolomite and limestone) are conveyed from the port and are stored in separated stockpiles in the pellet plant yard. They are reclaimed separately with front-end loaders and delivered to the additive dump hopper and then conveyed to the coarse limestone bin or the coarse dolomite bin.

The additives are withdrawn from the coarse limestone and/or the coarse dolomite bins into the additive grinding mill by a weight feeder in the proportions required by the pellet type. The additives are dry ground together in a 4.6 m diameter by 6.4 m length ball mill having a maximum power of 2,240 kW. The ground additives that have passed through the mill are lifted to the cyclones separator where coarse material is returned to the mill. The on-size material is recovered in collectors and conveyed into the ground additives bin.

17.4.7.4 Bentonite Storage and Grinding

Bentonite is conveyed from the port and is stored in a covered area to avoid contact with water before its final use in process. A loader is used to reclaim the unground bentonite and deliver it to the bentonite dump hopper from where it is conveyed to the coarse bentonite bin.

The coarse bentonite is ground in a Raymond mill having a maximum power of 500 kW and a grinding ring of 1,850 mm diameter. The bentonite grinding mill is provided with a hot gas generator to achieve drying during the grinding operation. The bentonite is more efficient as its moisture level decreases. Material which has passed through the mill is lifted in a stream of air and classified. Coarse material is returned to the mill and on-size material is recovered in collectors and discharged into the ground bentonite bin.

17.4.7.5 Coke Breeze Storage and Grinding

Coke breeze is optional and may be added when concentrate from the hematite plant decreases the overall concentrate Fe_3O_4 content. Coke breeze helps to decrease the fuel consumption at the induration machine and to maintain the Heat Magnetite Equivalent ("HME") when the concentrate Fe_3O_4 grade varies.

Coke breeze is conveyed from the port and is stored in the coke breeze stockpile in the pellet plant yard. A front-end loader reclaims the coke from this stockpile and delivers it in the coke dump hopper from where it is conveyed to the coarse coke breeze bin. Coke is ground in a 2.4 m x 4.3 m (D x L) coke grinding ball mill with a maximum power of 298 kW equipped with screens. Coarse material is returned to the mill and on-size material is sent to the ground coke bin.

17.4.7.6 Mixing

Concentrate, bentonite, additives and coke breeze are weighed and added to the high intensity mixer feed chute in the proportions required by the pellet type. There are two (2) mixers per indurating line and each mixer is sized to handle 100 % of the nominal capacity for breakdowns and scheduled maintenance. The mixers can be fed simultaneously or individually. The total volume per mixer is 30 m3 and the installed motor drive power is 1,200 kW.

Process water is added to the mixer for proper balling behavior and Sodium Hydroxide (NaOH) may be introduced in the mixer to activate the bentonite. Washdown thickener underflow is also fed to the mixers. The material is intensively mixed to obtain a homogenous balling material.

The mixed material is conveyed to the balling area and rejected green balls from the balling area are added to the conveyor downstream to the mixers.

17.4.7.7 Balling

A conventional arrangement for the balling discs, single roller deck screens, fine and coarse green balls return conveyor is proposed. Balling discs are sized to handle 100 % of the furnace capacity when one (1) disc is out.

The mixed material is distributed into the balling disc feed hoppers from an overhead conveyor equipped with automated V-Shape ploughs. The horizontal plough conveyor will be extended over the last balling disc feed hopper and will discharge onto the recycling conveyor.

The mixed material is continuously withdrawn from the balling disc feed hopper by a weight conveyor belt and is fed to the 7.5 meter diameter balling disc. The inclination of the balling disc is automatically adjustable from 40 to 60°. The installed power is 255 kW and the capacity is 150 tph per balling disc. There are 13 balling disks per line. The balling discs discharge onto the single roller decks, one (1) per balling disk, which remove undersized and oversized green balls which are recycled to the balling disc feed hoppers. The on-size screened green balls are discharged onto the main product conveyor and transferred on a reciprocating conveyor that spread them evenly onto the Induration machine double roller deck, one (1) per indurating line, to remove any green balls broken in transit. Undersized green balls are recycled to the balling disc feed hoppers. On-size green balls feed the indurating machine.

17.4.7.8 Induration and Hearth Layer

A straight travelling grate is the proposed technology for induration of the concentrate.

A machine having a grate size of 816 m^2, being the largest proven machine capacity for the application, is selected and should produce 8.5 Mtpy of pellets with a magnetite only concentrate. A slightly lower capacity is expected for a hematite/magnetite mix.

The induration furnace is a standard design equipped with updraft and downdraft green pellet drying, preheating and firing zones, and fired pellet cooling zones. Variable speed fans will provide the air addition control and a heart layer system made of recycled fired pellet allows pallets and refractories to be protected.

Dribbles are recovered on the dribble conveyor and discharged on the fired pellets conveyor.

17.4.7.9 Fired Pellets Handling

Fired pellets are conveyed and discharged into the heart layer segregation bin. Coarse pellets are conveyed into the heart layer bin to feed the induration machine. Other pellets are conveyed outside the pellet plant onto product piles where the reclaiming system allows their retrieval for expedition. When DR grade pellets are being made, they are sprayed with a fine dolomite coating mixture prior to being stocked. At least one (1) pellet stockpile is required per pellet type.

17.4.7.10 Gas and Dust Cleaning

A gas cleaning and dust removal system will be provided by electrostatic precipitators ("ESP") for the indurating machine and dust collectors for the rest of the plant (grinding area, segregation bin, etc…). Dust retrieved by the dust cleaning system is directly reintroduced into the process when possible or is sent to the washdown thickener to be reintroduced at the mixer feed.

17.4.7.11 Process Water System

Most of the plant areas are considered dry areas. However, spillage and periodical clean-ups will require wash water. The washed material will be collected in a 20 meter diameter thickener to be reintroduced into the process at the mixer feed. There is one (1) washdown thickener per line. The thickener overflow is directed to the process water tank. Excess process water, if any, is treated and returned to the environment.

18 Project Infrastructure

18.1 General Arrangement

The Full Moon Property is located 88 km (by the projected road) northwest of the town of Schefferville, Québec. The waste and overburden dumps, the crushing plant as well as the buildings, such as concentrator, offices and workshops, are located west of the planned open pit. Drainage ditches will be constructed around the open pit and dumps to direct water runoff to settling ponds to avoid contamination. The mineralized material will be hauled by the mine haul trucks to the 2 gyratory crushers about 2 km from the concentrator. A haulage road will be constructed between the mine and the crushers. All crushed material will be sent via two conveyors (1.69 km and 1.24 km) to the two cone crushing and screening plants, stockpiled, and, subsequently reclaimed and transported to the concentrator via a short conveying system.

The annually produced 20 Mt of iron concentrate (10 Mt per line of the concentrator) will be conveyed to two 60,000 tonne storage silos or to a combined emergency stockpile. The stored iron concentrate will be loaded in train cars and transported by rail via the newly constructed railway loop. This railway loop will tie-in at an existing railway system for further transport. An accommodation camp will be built about 1.5 km from the concentrator. A 450 km long, new 315 KV power line will be built starting at the LG4/Tilley substation.

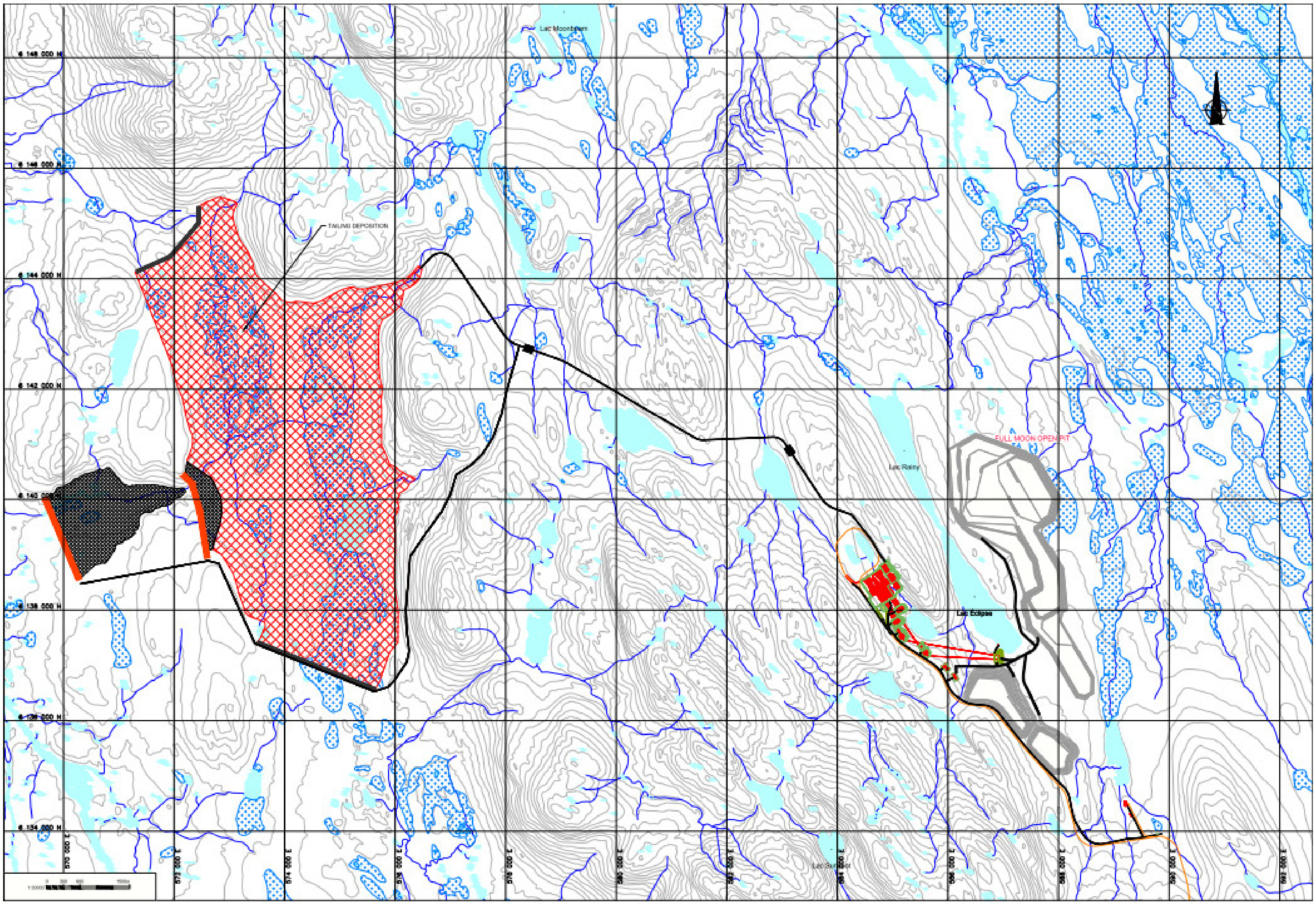
Four options were analyzed, namely:

- Option 1: High Silica Concentrate without pelletizing plant;
- Option 2: Low Silica Concentrate without pelletizing plant;
- Option 3: High Silica Concentrate with a pelletizing plant; and
- Option 4: Low Silica Concentrate with a pelletizing plant.

Figure 18.1 shows the general location map with the planned project development.

The concentrate will be transported via the new, 91 km long railway line, first to Schefferville and subsequently, on the existing TSH and QNSL railroads from Schefferville to Sept-Iles, where the ore cars (gondolas) will be transferred to a new multi-user terminal. From the multi-user terminal, the iron concentrate could be sent to two (2) pellet plants via conveying system or to the port facilities to be loaded directly into vessels.

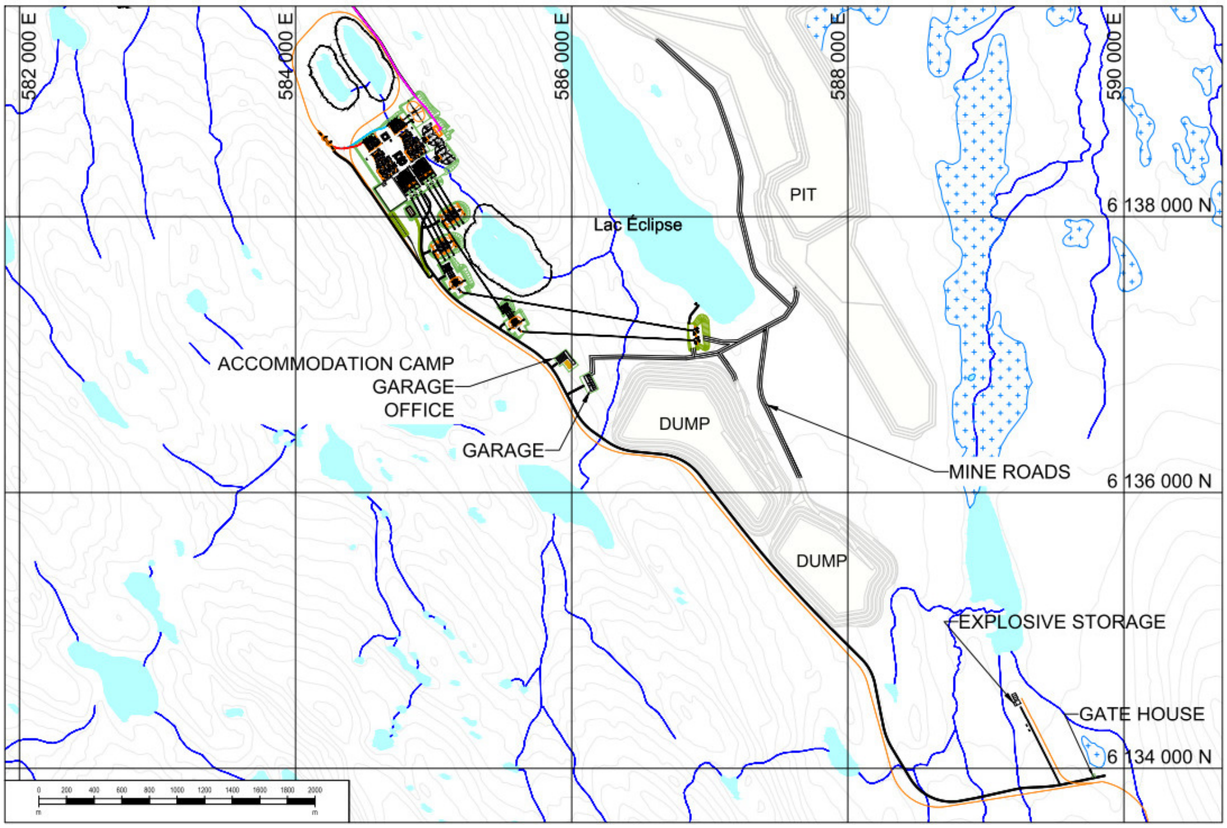
Figure 18.1 – General Location Map



18.2 Mine Area

The open pit, waste/overburden dump, mine workshop, mine road, the drainage system and the explosives storage are illustrated on Figure 18.2.

Figure 18.2 – Mine Site Installations and Infrastructures



18.2.1 Haulage Roads and Site Roads

The haulage roads from the mine area are designed with a width of 31 m, the same width as the roads in the open pits, to accept 227 t (240 ton) rigid frame haul trucks. All roads were designed to minimize the cut and fill and respecting a maximum grade of 8%. The earth excavation will be used to backfill the lower points on the road alignment. The rock excavation will be used, without any further crushing, for the sub-base for a thickness of 1,000 mm. Finally, the base of the road will have a thickness of 400 mm and will be made of waste rock from the mine. The roads in the mine area include:

- Haulage roads to the crushers;
- Haulage road from the pit to the waste dumps and the overburden dumps; and

- The access road to the explosives storage (shown in Figure 18.2). This road will be designed with a width of 15 m because the explosives trucks and other vehicles are much smaller than the haulage trucks.

18.2.2 Mine Equipment Workshop

The workshop for mining equipment maintenance will be a steel structure building unit. The building will be 130 metres long by 45 metres wide accommodating eight (8) bays. Both ends of the bays will have a large door for the 227 t trucks.

18.2.3 Fuel Storage and Filling Station

The main storage facility for the diesel fuel for the mine equipment will be two galvanized steel of 1 million liters tanks installed and secured on a pad. The two fuel tanks will be interconnected and will direct the flow to the equipment fueling station, consisting of two mine truck fueling pumps.

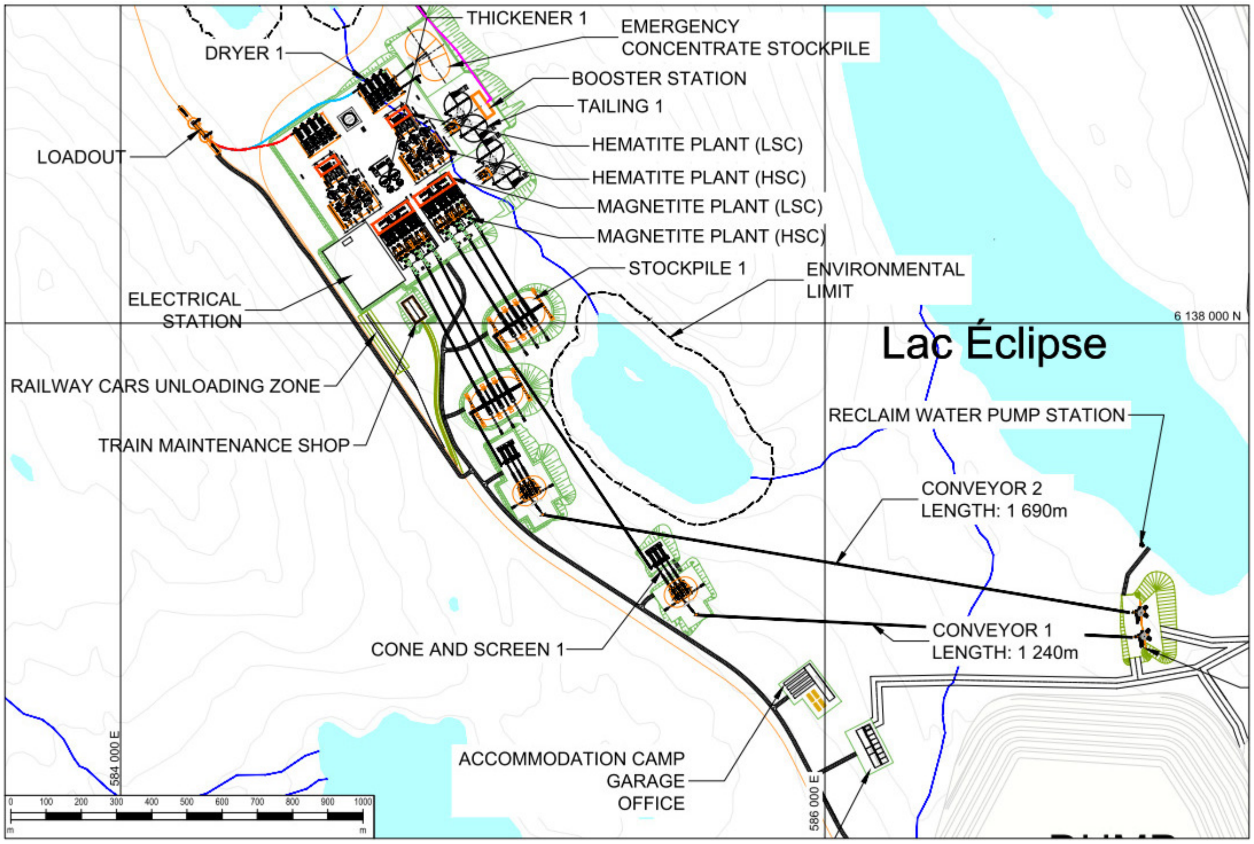
18.2.4 Explosives Preparation and Storage

The explosives preparation and storage facilities will be constructed in a remote area, about 6 km south of the concentrator (shown on Figure 18.2) and at least 500 m from the access road to the site. A magazine for the accessories will be located approximately 150 m further along the access road to the explosive preparation building. It will be designed to the specifications and requirements of the explosives supplier. A dedicated access road will serve the explosives storage area.

18.3 Concentrator Area

The concentrator infrastructures are illustrated on Figure 18.3.

Figure 18.3 – Concentrator Site Installations and Infrastructures



18.3.1 Crusher Plant

The mineralized material coming from the pit will be crushed in two gyratory crushers, each with a capacity of 5,816 tonnes per hour. Maximum size of the crushed material will be 200 mm. There will be four truck dump "stations", one on each side of the crusher dump hopper.

18.3.2 Crushed Material Screen and Stockpile

The mineralized material crushed by the gyratory crusher will be conveyed to the two cone crushing and screening installations. This material is reclaimed, screened and crushed as required before conveyed to the two covered A-frame stockpiles.

18.3.3 Concentrators

A reclaiming apron feeder system reclaims the stockpiled material and transports it to the two magnetite concentrator lines. The rejected material from the magnetite lines will be sent to the two hematite concentrator lines. The final combined product (magnetite and hematite) forms concentrate cakes that will be dried, conveyed and stored in the loading station. One emergency concentrate stockpile is planned also. The final rejects of the hematite plant will be partially dewatered and pumped to the tailings pond.

HSC and LSC processing capability

As has been shown in Section 17 and in Section 18.1, the study analyzed two types of concentrate namely High Silica Concentrate, Option 1 and Option 3, and Low Silica Concentrate, Option 2 and Option 4. The infrastructures around the concentrators will be basically the same and the design of the concentrators allows the addition of the processing equipment required to reduce the silica content of the concentrate and the production of the LSC.

18.3.4 Security Gate House

A security gate house will be installed on the main access road, just before the access road to the explosive plant, about 6 km southeast of the concentrator. The guard will authorize the entry of visitors to the concentrator and mine site.

18.3.5 Administration Building and Accommodation Camp

The accommodation camp will be built at the beginning of the construction period, to accommodate the construction labor workforce. The accommodation camp will include housing to accommodate 300 workers, a cafeteria large enough to accommodate all shift workers and supervisors, a meal preparation section, including all required cooking appliances and utilities, food preparation and refrigerators. It will also include an entertainment/recreation room, and a medical clinic facility for first aid to serve the camp and operations.

The administrative building will be constructed beside the accommodation camp. The administration building sections will house the offices for the project managers and other supervisory personnel as well as the concentrator supervisors, secretary, accounting, human resources, safety and first aid personnel. A section of the building will be reserved for the mine related operations such as offices for managers and department supervisors, surveyors, engineering and mine planning personnel, as well as secretary personnel.

18.3.6 Site Roads

The access road from the guardhouse to the concentrator and to the accommodation camp is designed to minimize the cut and fill required; the road is 15 m wide and the maximum grade of the road is 7%. The total length of the site road system is approximately 100 km and accounts for the access road from Schefferville and all the roads on site. The earth excavation will be used to backfill the lower points on the road alignment. The rock excavation will be used without any further crushing for the sub-base for a thickness of 1,000 mm. The final base of the road will have a thickness of 400 mm and will be made of crushed stone (MG-20).

18.3.7 Site Drainage and Settling Ponds

A storm drainage system will be excavated that will exploit the natural drainage around the pits, roads, infrastructures and pads with a network of open ditches and culverts that will connect with one or more settling ponds.

Ditches and culverts will be designed for a 1 in 100 year recurrence event and will be checked for peak intensity flows. Sedimentation ponds will also be designed for a 1 in 100 year recurrence event.

18.3.8 Services

Electrical power will be supplied to the project from a 315kV-35kV substation to be built near the concentrator and will be connected to a new 315 kV power line located approximately 450 km from the concentrator.

A 35 kV transmission line network will distribute the power needed to the substations and PEB of different areas, such as the mine site, accommodation camp, the concentrator and other facilities. The mine site will be powered by a 7.2 kV transmission line from a substation 35kV-7.2kV that will provide all the power for the electric power shovels and the electric production drills.

A pump house will be constructed at a small lake close to the concentrator. Water will be pumped to a water treatment facility located inside the concentrator. Potable water will also be pumped to the accommodation camp. The pumping and distribution system will include a potable water reserve tank.

Electrical connection and controls of all potable water equipment will be connected to the plant emergency power supply.

Central organic waste collection and on site composting equipment will be provided and inorganic waste will be disposed into an incinerator.

18.3.9 Communications

Telecommunications and radio systems will be provided to enable communication between individuals working in the different areas, as well as provide computer and internet services in all offices, control rooms etc.

18.4 Tailings Management Facilities Area

The tailings pond is located in a natural valley to the west of the concentrator, at a distance of about 10 km. Containment of the process solids will be made by the natural terrain on the east and west sides while a maximum high 37-meter tailings dam would be required on the north and south sides at mine closure. Impervious dikes required for the sedimentation and polishing pond will reach a maximum height of 44 m. Final elevation of the process solids (463 m) will be 162 m lower than the process plant (625 m). The tailings area is shown on Figure 8.1.

18.4.1 Dikes and dams

Three types of dikes will be used in this project: impervious dikes build in one phase, impervious dikes build in multiple phases, and, peripheral dikes raised with tailings.

Impervious dikes built in one phase: for impervious dikes with more than fifteen meters in height are required to be built in one phase. A dike design with a center till core and rockfill shoulders was considered. Otherwise, for impervious dikes with less than fifteen meters in height, a rockfill dike using a geomembrane as impervious layer was considered.

Impervious dikes built in multiple phases: for impervious dikes with more than fifteen meters in height and that could be built in multiple phases. A dike design using rockfill and an inclined till core as impervious layer was considered. Otherwise, for impervious dikes with less than fifteen meters in height, a rockfill dike using a geomembrane as impervious layer was considered.

Peripheral dikes raised with rockfill or/and tailings (tailings dikes): this design consists in an impervious starter dike build using rockfill and a geomembrane as impervious layer which is raised using tailings or/and rockfill. In comparison with other methods, raising dykes with tailings is very economical. However, for this to be achievable the liquefaction susceptibility of the tailings must be evaluated and the silty tailings need to be drained to allow its manipulation and transportation to the construction site. This can be achieved either by separating the tailings from the water using a cyclone or by stacking the deposited tailings using machinery.

The general design criterion for impervious dikes with a till core is to have a five meter freeboard to minimize frost penetration in the till core. For other types of dikes, a design criterion of five meter freeboard was also retained due to the topographic map precision available at this stage of the project.

18.4.2 Control structure

A water control structure must be built between the sedimentation and polishing ponds to maintain a certain volume of water in the impoundment area. This ensures an adequate primary sedimentation of the suspended solids and reduces the need for further water treatment.

This control structure will need to be raised along with the dike in which it will be built, as required based on the impoundment fill rate.

18.4.3 Emergency spillway

Emergency spillways need to be designed to evacuate a 24-hour or 6-hour probable maximum rainfall that occurs when all the water ponds are at their maximal capacity.

18.4.4 Pumping station and treatment plant

A water treatment plant is proposed to be constructed and should allow excess water from the impoundment area to be treated to meet the criteria of the “Ministère de Développement Durable, Environnement et Parcs” before being released into the environment. These infrastructures are necessary to ensure that water levels are adjusted for spring melting and for fall accumulation.

The preliminary analysis of these structures was not part of the present mandate, and is therefore not included in the cost estimate presented in this report.

18.5 Rail Infrastructures

18.5.1 Railroad

The main railroad starting from Schefferville, going to the site and finishing by a loop have a total length of 91.5 km. The rail is a single line with one siding every 30 km with the capacity to receive a train of at least 240 railcars of the gondola type. The sidings allow the train crossing on the single line.

There are also three spur lines that are connected to the main line:

- A spur line with a length of 600 m, going to the explosive preparation and storage;
- A spur line for material unloading with a length of 440 m;
- The spur line going to the rail maintenance workshop with a length of 505 m.

18.5.2 Rail Maintenance Workshop

The workshop for rail equipment maintenance will be a steel structure building unit. The building will be 75 metres long by 40 metres wide.

18.6 Port and Terminal Area

The iron ore concentrate will be transported by train via an existing railroad to the port of Sept-Iles. The port of Sept-Iles planes to develop a multi-user terminal. This terminal will be used to unload the train and stockpile the material before sending the material to the port via a conveyor system to load ships.

18.7 Pellet Plant Area

18.7.1 Pellet plant

There will be a pellet plants, with two lines, each with a capacity of 8.5 Mtpy.

The project was developed to keep the possibility to sell the concentrate directly on the market or use part of the concentrate for the production of pellets. Depending on the concentrate properties (HSC or LSC), the produced pellets will vary. The HSC will be producing the HSF pellets (Option 3), while the LSC will produce the DR pellets (Option 4).

18.7.2 Security Gate House

A security gate house will be installed on the main access road. The guard will authorize the entree of visitors to the pellet plant site.

18.7.3 Administration Building

The administrative building will be constructed beside the pellet plant. The administration building will house the offices for the project managers and other supervisory personnel as well as the plant supervisors, secretary, accounting, human resources, safety and first aid personnel.

18.7.4 Site Drainage and Settling Ponds

A storm drainage system will be excavated that will exploit the natural drainage around roads, infrastructures and pads with a network of open ditches and culverts that will connect with one or more settling ponds.

Ditches and culverts will be designed for a 1 in 100 year recurrence event and will be checked for peak intensity flows. Sedimentation ponds will also be designed for a 1 in 100 year recurrence event.

18.7.5 Services

Electrical power will be supplied to the pellet plant site from a 161kV-35kV substation to be built near the pellet plant and will be connected to a new 161 kV Hydro-Québec power line located approximately 3 km from the plant.

A 35 kV transmission line network will distribute the power to the substations and PEB of the different areas including offices, the pellet plant and other facilities.

A pump house will be constructed on the shore of Saint-Marguerite River near the pellet plant site. Water will be pumped to a water treatment facility located inside the plant.

18.7.6 Communications

Telecommunications and radio systems will be provided to enable communication between individuals working in the different areas, as well as provide computer and internet services in all offices, control rooms etc.

19 Market Studies and Contracts

19.1 Iron Ore Market Overview

The developing world, and in particular Asia, will be the growth engine for the next decade. The developed world demand outlook is more moderate and so the majority of the growth in materials demand is expected to come from developing world consumption, supported by the continued urbanization of the major developing economies, including China and India.

The large increase in developing world demand for metals (in particular China) has replaced much of the demand from the industrialized world. The world's manufacturing industries have continued to move from high cost developed countries to low labour cost countries and a significant portion of their production has been exported back to the developed world. But the increase in consumption by the developing countries has begun to increase labour costs, which has resulted in some labour intensive manufacturing beginning to return to the developed world.

Developing world end-use metal consumption per capita will slowly catch up to the developed world levels. This is because consumption in the developing world is focused on metal intensive products such as fridges, furniture and pots and pans, compared to services which dominate industrialized world spending.

The quality of construction in the developing world is improving, resulting in a larger portion of developing world steel used in construction being galvanized (increasing demand for zinc) or upgraded to stainless steel (increasing demand for nickel). This results in higher quality, at a higher cost.

As the Chinese economy matures and shifts away from capital-intensive growth of the last two decades, metals demand growth is expected to moderate.

The price of iron ore has declined by nearly 50% in 2014 as mining companies including Rio Tinto Group and BHP Billiton Ltd., expanded production in Australia, resulting in an oversupply of iron ore. It is expected that more of China's higher cost iron ore supply will exit the market, as the lower cost Australian supply continues to flood the market. The Australian Bureau of Resources and Energy Economics estimated that "global trade in iron ore increased by 10% in 2014 to 1.35 billion tonnes, driven by a 24% increase in Australian exports and a 10% increase in Brazilian exports. China's imports are estimated to have increased by 118 million tonnes as steel mills continued to switch from domestics to cheaper foreign sources of iron ore."

As noted in Australia's Resources and Energy Quarterly, December 2014 – "2015 world trade in iron ore is forecast to increase by 2.8% to 1.4 billion tonnes, supported by a 7% increase in Australian and Brazilian exports. However, this increase is forecast to be partially offset by a reduction in exports from high cost producers."

Australia & New Zealand Banking Group Ltd. recently said in a report "that any recovery in the price of iron ore will be driven by supply cuts, including high-cost mines in China, where almost the entire industry is loss-making at current prices now. They further noted that prices are set to remain weak in 2015, but appear to be "oversold" and there is potential for a relief rally in the second half of 2015.

China - Iron Ore Imports

China is estimated to have imported a record 938 million tonnes of iron ore in 2014, up 14% on 2013. However, this increase led to record high levels of port stocks which peaked at 106 million tonnes in June and only declined marginally in the six months to December.

Australia's Resources and Energy Quarterly forecast that "China's 2015 iron ore imports will increase by a further 3.7% and total 973 million tonnes, supported by increased demand for seaborne ore. Low steel industry profitability is expected to eventually push mills to source the cheapest iron ore available and switch increasingly to low cost imports."

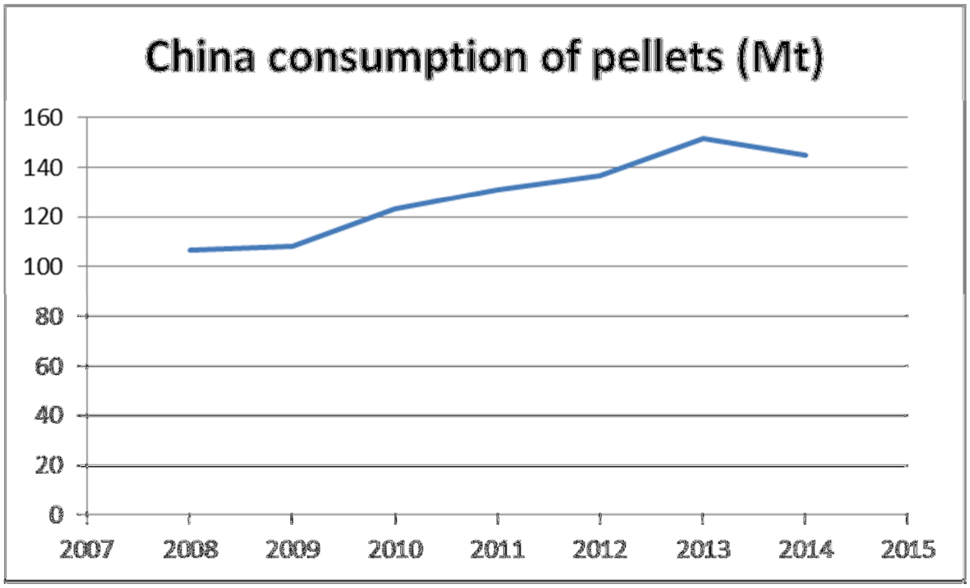
Australian Iron Ore Exports

In 2014 Australia's exports of iron ore are estimated to have increased by 24% to 718 million tonnes. This growth was driven by an expansion of production and infrastructure capacity in the Pilbara region.

Following the slump in prices from June to October 2012, prices remained above US$120/t CFR for 62% Fe content China fines with a sharp down turn in the fall of 2012, reaching US$99 CFR for 62% Fe content fines and then a restocking phase pushed prices towards US$135/t in 2013. As new capacity came on-stream, the industry's price started gradually to drop and by the end of 2014 it reached a historical bottom of US$65 CFR for 62% Fe content fines.

The concentrates and pellets premiums on the higher grade products (with lower silica content) has averaged US$13.71 and US$36.08 respectively through the last several years from 2008 until December 31, 2014. The demand and the consumption of pellets in China is gradually increasing over the last seven years (Figure 19.1). From 106.8 Mt in 2008 to 151.4 Mt in 2013. Consumption in 2014 was slightly reduced likely impacted by supply/demand and price variation of 62% fines.

Figure 19.1 – China consumption of pellets (Mt)



Source : CRU Iron Ore Market

19.2 Market Opportunities and Strategy

Located approximately 600 km north of Sept-Iles, Quebec the Scheffervile area of the Labrador Trough is a prolific DSO iron ore district initially exploited by the Iron Ore Company of Canada in the 1950s through the early 1980s. IOC made investments in to the town of Schefferville and also built the QNS&L rail line, connecting Schefferville to Ross Junction and Sept-Iles, and indirectly investing in the town's utilities and airport. Investment was also made in Sept-Iles for the port to transport iron ore product to clients. Falling iron ore prices in the 1980s and demand for alternative iron ore products resulted in closing of IOC's DSO Schefferville operations in 1982.

Since 2005, a resurgence in iron ore demand has renewed interest in the Schefferville area, attracting large investment in exploration and development. Development in the area is facilitated by the existing rail line, IOC port in Sept-Iles as well as the future multi-user port currently under construction 35 km beyond Sept-Iles.

19.2.1 European Market

European iron ore demand is expected to rise from 144.8 Mt in 2014 to a peak of 154.1 Mt in 2017. The European iron ore market is dominated by a few large operators in Sweden and Norway. Sweden's LKAB is currently Europe's largest iron ore producer at a rate of ~28 Mt annually products.

19.2.2 China Market

Development in China has been the driving force for iron price for the past twenty years as the urbanization of its large population has required an immense steel consumption. The Chinese demand has accounted for more than half of the world iron consumption. Chinese iron smelters lower requirements for high value products (compared to European smelters) has resulted in a limited demand for pellet and lump as such products demand premia on prices.

19.3 Iron Ore Pricing for Project Financial Evaluation

For the PEA Study, the long term benchmark iron ore price base case is forecasted at US$95/DMT CFR China for 62% Fe sinter fines. This price forecast is based on the average consensus iron ore price forecast from seven analysts reports shown in Table 19.1. This price has been used as the basis for the economic analysis presented in Chapter 22. A sensitivity analysis at various prices above and below the aforementioned base price was also performed as part of the economic evaluation of the Project.

Table 19.1 – Analyst long term price forecast ($US/DMT, 62%Fe sinter fines CFR China)

Compagny	Date	2014E	2015E	2016E	2017E	2018E	LT
RBC	09/Nov/14	$111.50	$105.00	$100.00	$100.00	$90.00	$80.00
BMO	29/Sep/14	$106.00	$95.00	$105.00	$100.00	$115.00	$109.00
CS	24/Sep/14	$100.00	$89.00	$87.00	$90.00		$90.00
Canaccord	2/Dec/14	$96.80	$70.00	$77.50	$85.00		$85.00
Metal Expert Consulting	31/July/14	$104.00	$105.00	$110.00			$120.00
Scotia Bank	6/Oct/14	$99.00	$88.00	$85.00	$80.00	$85.00	$100.00
Goldman Sachs	6/Aug/14	$106.00	$80.00	$82.00	$82.00		$80.00
Average (Consensus)		**$103.33**	**$90.29**	**$92.36**	**$91.17**	**$96.67**	**$94.86**(1)

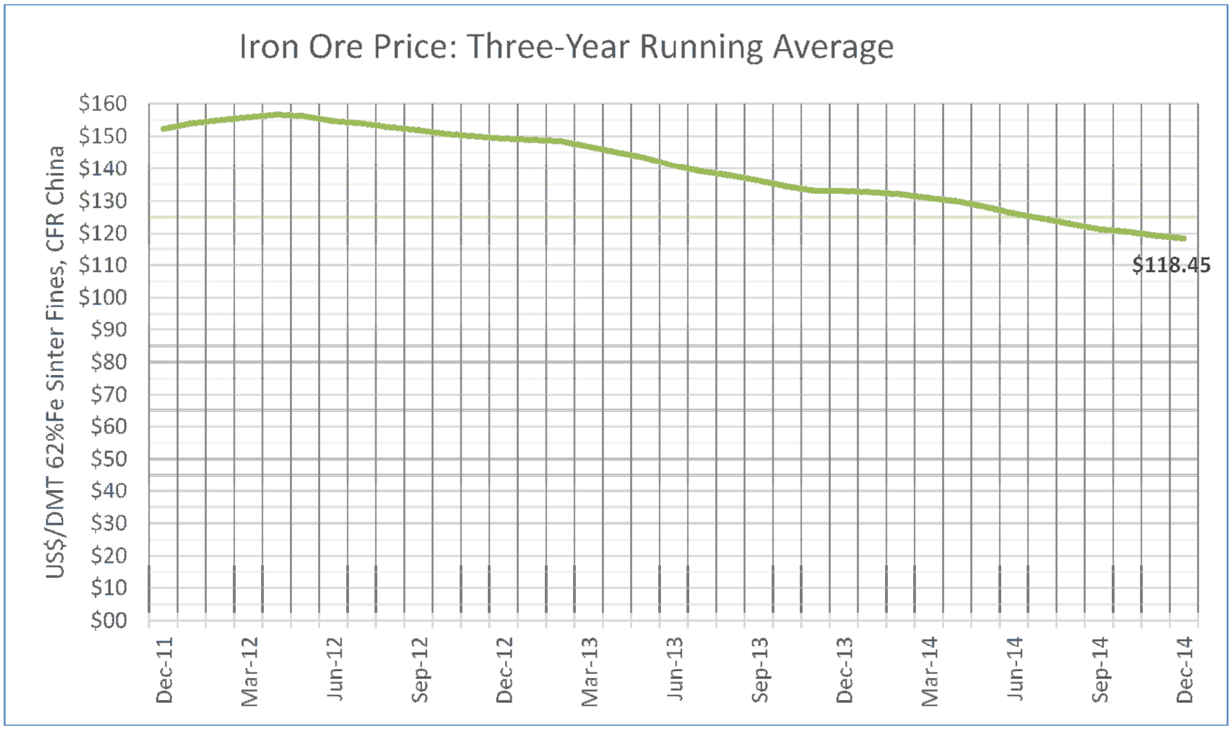
(1) Rounded to US$95 for financial evaluation purposes

Analyst price forecasts for 62%Fe DMT CFR China were obtained during the period July 2014 through December 2014, when WCSL collected research on forward looking price forecasts for use as the base case benchmark price in the economic analysis. A cut- off date for data was established as of December 31, 2014.

CAUTION: *Readers are cautioned that the period for collection of "forward looking information" related to forecasts for iron or selling price was July through December 2014 and the effective date of the PEA Study NI 43-101 Technical Report is March 2, 2015. During the first two months of 2015, the benchmark price for 62%Fe per DMT sinter fines CFR China has seen significant volatility and has occasionally reached levels below US$60 per DMT.*

In its analysis, WCSL also collected historical data to track the 3-year running average selling price for 62%Fe DMT CFR China and this information is shown graphically in Figure 19-2. As of December 2014, the three-year running average iron ore price for 62%Fe sinter fines CFR China was US$118.45/DMT. This data is presented for information purpose only and was not used for determining the projected long term price used for the economic analysis.

Figure 19.2 – Iron Ore Price Based on Three-Year Running Average
(Source: Metals Bulletin Iron Ore Monthly Index)



For calculation, the moisture content is 6% and exchange rate CAN$0.80=US$1.00.

The rest of the products are premiums or deductions from the basic CFR Sinter Fines Fe 62%. To develop a long term price base and deductions and premia, a six year average historical daily price base was used. Days were selected where the CFR Sinter Fines Fe 62% of $95 and allocating premiums or deductions based on that same average price. Subsequently a six year average and regression analysis was prepared. The Table 19.2 explained the calculation of premiums and deductions for each type or products and the Table 19.3 show the price used in the economic analysis.

Table 19.2 – Project Long Term Prices with Premium or Deductions

Type of Analysis	DR Pellet	Low Silica Concentrate	HSF Pellet	High Silica Concentrate
MBIO Index 62% CFR US$DMT	US$ 95.00	US$ 95.00	US$ 95.00	US$ 95.00
Premium for 66%	US$ 6.13	US$ 6.13	US$ 6.13	US$ 6.13
Deduction for silica	-	-	(US$ 6.00)	(US$ 6.00)
Premium for Pellet	US$ 22.00	-	US$ 23.00	-
Other consciderations	US$ 16.87	US$ 16.87	US$ 16.87	US$ 16.87
Selling Price	**US$ 140.00**	**US$ 118.00**	**US$ 135.00**	**US$ 112.00**

Table 19.3 – Products Seling Prices

Product	CFR Price $US/DMT	Shipping Cost $US/DMT	FOB Price $US/DMT	FOB Price $CAD/DMT
Low Silica Product				
DR Pellet	140.00	15.00	125.00	156.25
Low Silica Concentrate	118.00	15.00	103.00	128.75
High Silica Product				
HSF Pellet	135.00	15.00	120.00	150.00
High Silica Concentrate	112.00	15.00	97.00	121.25

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20 Environmental Studies, Permitting and Social Community Impact

The Project is subject to numerous laws and regulations. The most significant laws, directives and regulations amongst the legislation and government directives to be considered and respected are presented hereinafter.

The project description used in this section considered the option that would include the pelitizing plant near the multi-user terminal in Sept-Îles.

20.1 Legal Framework

This section presents the key aspects of the two regulatory frameworks (provincial and federal) as they pertain to the Project. The Project has components located in two of the four territories applicable in the Province of Québec and therefore is subject to two specific environmental assessment regimes in force for each of these territories.

- The Kativik regime: corresponds to a territory north of the 55th parallel, subject to the Kativik environmental regime for project authorization under the James Bay and Northern Québec Agreement ("JBNQA") (Section 23) and under Chapter II of the Quebec Environment Quality Act ("EQA");
- The Southern Québec regime: corresponds to the Southern Québec territory, subject to the environmental regime for project authorization under Section 31 of the EQA.

Application of the Canadian Environmental Assessment Act, 2012 (CEAA 2012) is independent of the four Québec territories. Application of the CEAA 2012 to mining projects is determined by the regulations under CEAA 2012, in particular the Regulations Designating Physical Activities which specify the "designated projects" that may require an EA by the federal agency. Application of CEAA 2012 to other components of a mining project, aside from the mine, depends on other triggers specified by the regulation and requires a case-by-case screening of the features of a given project against the triggers specified by the federal regulation. It is not anticipated that any of the four main project components will require a Panel Review under the CEAA 2012. Table 20-1 presents the applicable EA regimes for each of the four main Project components.

Each of the three above-mentioned environmental regimes for project authorization has their own specific procedures and requirements for the scope and execution, when applicable, of EAs, and each also has specific mechanisms for the evaluation of EAs and for the public hearings.

Table 20.1 – Environmental Assessment Regimes Applicable to the Project

Project Component	Applicable EA Regime
Mine Site and Related Infrastructure	JBNQA – Section 23 CEAA 2012
Transmission Power Line from the Brisay Dam	This component is considered to be under Hydro Québec responsibility
Railway to Schefferville	JBNQA – Section 23 EQA CEAA 2012
Pelletizing Plant	EQA CEAA

20.1.1 Provincial Legislation (Québec)

20.1.1.1 Environment Quality Act (CQLR C. Q-2)

ENVIRONMENTAL IMPACT ASSESSMENT ("EIA")

The Project includes three main components (the power line from Brisay is being considered under Hydro-Québec responsibility):

- The mine site and related infrastructure;
- The railway to Schefferville;
- The pelletizing plant in Sept-Îles.

There has been no discussion yet with provincial authorities regarding a potential strategy on how to harmonize the procedures of the two environmental regimes. It could be decided to prepare two separate applications for environmental authorization, one the mine and the railway and a second for the pellet plant.

Despite the fact that there are two distinct procedures under different regimes, they all follow the same general five steps. In the Southern regime, the Ministère du Développement durable, de l'Environnement et de la Lutte contre les changements climatiques ("MDDELCC") is in charge of the procedure and relies on the Bureau d'audience publique du Québec ("BAPE") for public hearings. In the Northern Québec regimes, various committees are in charge of the EIA and review, and consultation, while the MDDELCC generally acts as the Administrator of the procedure and takes the final decision. The five general steps are as follows:

1. **Project Submission:** A notice of intent and preliminary information is provided to the Administrator. This information includes the purpose, nature and scope of the project, as well as possible variants in terms of location or layout.

2. **Evaluation:** If the project is automatically subject to the assessment process, the responsible committee or the MDDELCC prepares a directive specifying the scope of the impact study that the proponent must undertake. This directive is sent to the Administrator, who forwards it to the proponent, with or without modifications. This project-specific directive shall also comply with Directive 019 on the mining industry which includes the guidelines for all mining projects. In certain cases, and before making a decision, the committee may request additional information if it considers the information submitted to be incomplete.

3. **Preparation of the Impact Study:** The proponent conducts the impact study, which must take into account the directive issued by the committee or MDDELCC. Once the impact assessment report has been submitted, the committee or MDDELCC reserves the right to ask questions regarding aspects of the project, or to ask for additional information. Therefore, the proponent may have to answer one or more series of questions and comments.

4. **Review and Public Participation:** The proponent submits the impact study to the Administrator, who forwards it to the committee or MDDELCC for analysis. Usually, in Northern Québec, the committee reviews the project at its next scheduled meeting. If the committee or MDDELCC considers that more information is required, it may formulate questions that are transmitted to the Administrator. The proponent may have to answer one or more series of questions and comments. The committee or MDDELCC must receive answers to these questions before continuing with its review. Moreover, the Native administration and the public have the opportunity to make representations to the committee, which may hold public consultations if it deems necessary. Under the Southern Québec, the BAPE is responsible for public consultation.

5. Decision and Authorization: The committee or MDDELCC decides whether the project will be authorized or refused, and if appropriate, specifies conditions for its implementation. The Administrator takes into consideration the decision of the committee in determining whether to approve the project and issue a certificate of authorization or a decree, depending on the EIA regime. This authorization does not exempt the proponent from obtaining authorization(s) that may be required by any law or regulation, including in relation to the EQA.

At this stage, a construction certificate of authorization can be requested. This certificate begins the process of requesting the various certificates of authorization needed for the different aspects of the project (i.e.: certificates of authorization for infrastructure, for the mine's operation).

Mine Site and Related Infrastructure

The mine infrastructure, including the open pit, the ore concentrator and tailings management facility are all located north of the 55th parallel and therefore subject to the Kativik environmental regime under the Section 23 of the JBNQA and under Chapter II of the EQA. For such project, Sections 187 to 204 of the EQA requires any person or group to follow the Environmental and Social Impact Assessment and Review procedure before undertaking a project targeted by Schedule A of the Act. Schedule A, paragraph (a), stipulates that all mining development, including the addition to, alterations or modifications of existing mining development are subject to the provincial procedure. The Project should therefore be the subject of an environmental and social impact assessment.

The Regulation respecting the environmental and social impact assessment and review procedure applicable to the territory of James Bay and Northern Québec (CQLR c. Q-2, r. 25), provides details on the information that should be included within the impact assessment statement ("EIS"). The Kativik environmental quality commission ("KEQC") act as the committee and as such is in charge of both the assessment and review of the EIS and the public consultation. The KEQC decides whether the project will be authorized or refused, and if appropriate, specifies conditions for its implementation. The Administrator takes into consideration the decision of the KEQC in determining whether to approve the project and issue a certificate of authorization.

Railway to Schefferville

The proposed railway between the mine site and Schefferville existing railway is encompassed within two EIA regimes: the Kativik environmental regime as described above and the Southern Québec regime. The Southern regime is described below.

According to Schedule A of the EQA, paragraph (q), all railroad projects are subject to the assessment and review procedure contemplated in sections 153 to 167 and 187 to 204. Therefore, the railway from the mine site to Schefferville requires a certificate of authorization under the Kativik environmental regime.

The Regulation respecting the environmental and social impact assessment and review procedure stipulates in Division 2, Section 2 (h) the construction of more than 2 km of railway is subject to the environmental impact assessment and review procedure provided for in Division IV.1 of the EQA and must be the subject of a certificate of authorization issued by the Government in accordance with Section 31.5 of the EQA.

Pelletizing Plant

The pelletizing plant and connected infrastructure such as the power line and substation are located in the Sept-Îles area and are therefore entirely under the Southern Québec regime. The Regulation respecting the environmental and social impact assessment and review procedure stipulates in Division 2, Section 2 (n.9), that the construction of a metal products processing plant that has an annual production capacity of 20,000 metric tons or more is subject to the environmental impact assessment and review procedure provided for in Division IV.1 of the EQA and must be the subject of a certificate of authorization issued by the Government in accordance with Section 31.5 of the EQA.

CERTIFICATES OF AUTHORIZATION UNDER SECTION 22 OF THE ENVIRONMENT QUALITY ACT

The implementation of the Project will require obtaining numerous certificates of authorization under Section 22. Section 22 stipulates that: "No one may erect or alter a structure, undertake to operate an industry, carry on an activity or use an industrial process or increase the production of any goods or services if it seems likely that this will result in an emission, deposit, issuance or discharge of contaminants into the environment or a change in the quality of the environment, unless he first obtains from the Minister a certificate of authorization. However, no one may erect or alter any structure, carry out any works or projects, undertake to operate any industry, carry on any activity or use any industrial process or increase the production of any goods or services in a constant or intermittent watercourse, a lake, pond, marsh, swamp or bog, unless he first obtains a certificate of authorization from the Minister".

CERTIFICATE UNDER SECTION 128.7 OF THE ACT RESPECTING THE CONSERVATION AND DEVELOPMENT OF WILDLIFE

Before carrying out any activity in aquatic, wetland and riparian environments in Québec, an authorization under Section 128.7 of the Act respecting the conservation and development of wildlife (CQLR c.C-61.1) may be required.

CERTIFICATE UNDER SECTION 32 OF THE ENVIRONMENT QUALITY ACT

Section 32 of the EQA stipulates that no one can establish a water supply intake and water purification devices nor carry out sewer work or install wastewater treatment devices before having submitted the plans and specifications to MDDELCC and having received their authorization.

CERTIFICATE OF AUTHORIZATION UNDER SECTION 48 OF THE ENVIRONMENT QUALITY ACT

Section 48 of the EQA specifies the requirement to obtain an authorization before installing or placing any apparatus or piece of equipment aimed at preventing, reducing or preventing the release of contaminants into the air.

20.1.1.2 Mining Act (CQLR C. M-13.1)

Many aspects of the Mining Act were modified with its redrafting, adopted in December 2013. Thus, a chapter has been added which includes provisions specific to Aboriginal communities. It requires holders of claims to advise within 60 days the relevant municipality and property owner that the mining lease has been obtained and to inform the municipality and property owner at least 30 days before the beginning of any work. The law also requires these claim holders to provide an annual report regarding the work carried out to the Ministère de l'énergie et des ressources naturelles ("MERN").

It is mandatory to declare the discovery of any mineral substance containing 0.1 % or more of uranium octaoxide, within 90 days of the discovery.

The Mining Act requires that a mining redevelopment and restoration plan, for which a certificate of authorization set forth in the Environment Quality Act was issued, as well as an economic scoping and marketing study regarding processing in Québec, be submitted to the Minister before a mining lease can be granted.

Moreover, it also requires the prior holding of public consultations before a mining lease can be issued for a metal mine with a production capacity of less than 2,000 metric tonnes per day.

The Mining Act allows the government, at the moment of concluding a mining lease and for reasonable cause, to require that the Québec's economic spinoffs, of the authorized (by the lease) ore mining, be maximized.

It requires the holder of the lease to put together and maintain a monitoring committee to promote local community involvement throughout the project.

The Mining Act requires that holders of mining rights provide information regarding the amount and value of extracted ore, the duties paid under the Mining Tax Act (CQLR I-0.4) and all contributions paid to the Minister.

20.1.1.3 Depollution Attestation

The depollution attestation, renewable every five years, establishes the environmental conditions under which the industrial establishment can operate. It should be noted that the proponent shall submit depollution attestation request within at most one year after having begun operating its mining site.

20.1.1.4 Other Provincial Laws, Regulations and Guidelines

In addition to the previously-mentioned laws, the Project must comply with the following:

- Forest Act (CQLR c. F-4.1);
- Watercourses Act (CQLR c. R-13, r. 1);
- Dam Safety Act (CQLR c. S-3.1.01, r. 1);
- Petroleum Products Act (CQLR c. P-30.01, r. 1.);
- Act respecting threatened or vulnerable species (CQLR c. E-12.01);
- Act respecting the conservation and development of wildlife (CQLR c. C-61.1);
- Cultural Heritage Act (CQLR c. P-9.002);

- Act Respecting Occupational Health and Safety (CQLR c. S-2.1);
- Sustainable Development Act (CQLR c. D-8.1.1);
- Dam Safety Regulation (CQLR S-3.1.01, r. 1);
- Transportation of Dangerous Substances Regulation (CQLR c. Q-2 r. 32.);
- Groundwater Catchment Regulation (CQLR c. Q-2, r. 6);
- Regulation respecting pits and quarries (CQLR c. Q-2, r. 7.);
- Regulation respecting wildlife habitats (CQLR c. C-61.1, r. 18);
- Regulation respecting occupational health and safety in mines (CQLR c. S-2.1, r. 14);
- Clean Air Regulation (CQLR c. Q-2, r. 4.1);
- Guidelines 019 on the mining industry (2012);
- Guidelines for specification of culverts installation at stream crossing;
- Guidelines for Preparing a Mining Site Rehabilitation Plan and General Mining Site Rehabilitation Requirements (1997),;
- Soil Protection and Contaminated Sites Rehabilitation Policy;
- Québec Water Quality Criteria (2013) for the protection of surface water.

These directives, laws or regulations may require that the Project proponent obtain one or more specific Certificates of Authorization.

20.1.2 Federal Legislation (Canada)

20.1.2.1 Canadian Environmental Assessment Act

The Canadian Environmental Assessment Act, 2012 (CEAA 2012) and its regulations establish the legislative basis for the federal practice of EA in most regions of Canada. The CEAA 2012 applies to projects described in the Regulations Designating Physical Activities. A project may also be designated by the Minister of the Environment if he or she is of the opinion that the carrying out of the project may cause adverse environmental effects, or that public concerns related to those effects warrant the designation.

Under CEAA 2012, an EA focuses on potential adverse environmental effects that are within federal jurisdiction, including:

- Fish and fish habitat;
- Other aquatic species;
- Migratory birds;
- Federal lands;
- Effects that cross provincial or international boundaries;
- Effects that impact on Aboriginal peoples;
- Changes to the environment that are directly linked to federal decisions about a project.

The EA will consider a comprehensive set of factors that include cumulative effects, mitigation measures and comments received from the public.

In order to determine whether such a federal EA is required, the proponent shall provide the Canadian Environmental Assessment ("CEA") Agency with a project description if the latter is targeted by the regulation. The CEA Agency, once it has received a complete project description, will have 45 days to determine whether an EA is necessary. This decision will be based on the likelihood of environmental effects that are within areas of federal jurisdiction. This 45-day timeframe includes a 20-day period during which the public will be invited to comment.

According to Section 16(a) of the Regulations Designating Physical Activities, the Project is subject to a federal EA as it involves the construction, operation (and, eventually, the decommissioning and closure) of a new metal mine with a production capacity of over 3,000 t/day, under Section 16(b) it includes a metal mill with an ore input capacity of 4,000 t/day or more. In addition, the construction of a new railway line that requires a total of 32 km or more of new right of way is also subject to federal assessment according to Section 25(a).

The Canada-Québec Agreement on Environmental Assessment Cooperation has been signed on August 2010. The agreement promotes a better coordination of the two environmental assessment processes (federal and provincial) in order to reduce overall delays. Through this agreement, information is allowed to be exchanged between the two levels of government and a joint review panel may be used to conduct hearings, if necessary.

20.1.2.2 Other Federal Laws and Regulations

Given the Project's elements, various permits from federal authorities will be required under the following laws and regulations:

- Explosives Act (RSC. 1985, c. E-17);
- Fisheries Act (RSC. 1985, c. F-14);
- Navigation Protection Act (SC 1985, c. N 22);
- Migratory Birds Convention Act (SC 1994, c. 22);
- Species at Risk Act (SC 2002, c. 29);
- Metal Mining Effluent Regulations ("MMER")

EXPLOSIVES ACT (RSC. 1985, C. E-17)

This law governs the manufacture, testing, sale, storage and importation of explosives in Canada. Under Section 7(1)(a), any disposal or storage of explosives requires approval.

FISHERIES ACT (RSC. 1985, C. F-14)

On June 29, 2012, amendments to the Fisheries Act received Royal Assent. The changes will focus the Act on protecting the productivity of recreational, commercial and Aboriginal fisheries. The Government is now focusing protection rules on real and significant threats to the fisheries and the habitat that supports them, while setting clear standards and guidelines for routine projects.

According to the Department of Fisheries and Oceans of Canada ("DFO"), the new Fisheries Protection Program contains a new prohibition that manages threats to fish that are part of or support commercial, recreational or Aboriginal fisheries with the goal of ensuring their productivity and ongoing sustainability. The new prohibition is also supported by definitions of commercial, recreational and Aboriginal fisheries in the Act, as well as a definition of "serious harm to fish", which is the death of fish or any permanent alteration to, or destruction of, fish habitat. Thus, according to modified Section 35(1), "No person shall carry on any work, undertaking or activity that results in serious harm to fish that are part of a commercial, recreational or Aboriginal fishery, or to fish that support such a fishery". Further, Section 36(3) stipulates that: “No person shall deposit or permit the deposit of a deleterious substance of any type in water frequented by fish or in any place under any conditions where the deleterious substance or any other deleterious substance that results from the deposit of the deleterious substance may enter any such water”. However, metal mine may obtain authorization to deposit deleterious substance into waters under the Metal Mining Effluent Regulations and it requires the inscription of the body of water to Schedule 2 of the Regulations.

According to Section 37(1) of the Fisheries Act, “If a person carries on or proposes to carry on any work, undertaking or activity that results or is likely to result in serious harm to fish that are part of a commercial, recreational or Aboriginal fishery, or to fish that support such a fishery, or in the deposit of a deleterious substance in water frequented by fish or in any place under any conditions where that deleterious substance or any other deleterious substance that results from the deposit of that deleterious substance may enter any such waters, the person shall, on the request of the Minister — or without request in the manner and circumstances prescribed by regulations made under paragraph (3)(a) — provide the Minister with any plans, specifications, studies, procedures, schedules, analyses, samples, evaluations and other information relating to the work, undertaking or activity, or to the water, place or fish habitat that is or is likely to be affected by the work, undertaking or activity, that will enable the Minister to determine:

a) whether the work, undertaking or activity results or is likely to result in any serious harm to fish that are part of a commercial, recreational or Aboriginal fishery, or to fish that support such a fishery, that constitutes or would constitute an offence under subsection 40(1) and what measures, if any, would prevent that result or mitigate its effects; or;

b) whether there is or is likely to be a deposit of a deleterious substance by reason of the work, undertaking or activity that constitutes or would constitute an offence under subsection 40(2) and what measures, if any, would prevent that deposit or mitigate its effects.”

The Project does include activities and infrastructures likely to cause such damage to fish and/or which require the discharge of substances considered to be deleterious into waters where fish live. Indeed, with no other information than what is currently available, the mining waste rock shall be considered deleterious while the tailings will be dumped into bodies of water likely to contain fish. WISCO will therefore need to comply with the requirements of Section 37(1).

NAVIGATION PROTECTION ACT (SC 1985, C. N 22)

Section 3 of the Act stipulates that: "It is prohibited to construct, place, alter, repair, rebuild, remove or decommission a work in, on, over, under, through or across any navigable water that is listed in the Schedule except in accordance with this Act or any other federal Act". In the event that such a work was constructed, an authorization request would be submitted to Transport Canada. According to preliminary information, the Project is not subject to this Act.

MIGRATORY BIRDS CONVENTION ACT (SC 1994, C. 22)

Canada hosts approximately 450 species of native birds, the majority of which are protected under the Migratory Birds Convention Act, 1994, and are collectively referred to as "migratory birds". Environment Canada is responsible for implementing the Migratory Birds Convention Act, which provides for the protection of migratory birds through the Migratory Birds Regulations and the Migratory Birds Sanctuary Regulations.

SPECIES AT RISK ACT (SC 2002, C. 29)

The purposes of the Species at Risk Act ("SARA") are to prevent wildlife species in Canada from disappearing, to provide for the recovery of wildlife species that are extirpated (no longer exist in the wild in Canada), endangered, or threatened as a result of human activity, and to manage species of special concern to prevent them from becoming endangered or threatened. Once a species is listed under the SARA registry, it becomes illegal to kill, harass, capture or harm it in any way. SARA can apply if a species at risk is found at any time throughout the year within a mining lease.

METAL MINING EFFLUENT REGULATIONS

The Metal Mining Effluent Regulations was registered on June 6, 2002 and is under the Fisheries Act. It applies to all Canadian metal mines (except placer mines) that exceed an effluent flow rate of 50 cubic metres per day and deposit effluent into fisheries waters at any time after the regulations were registered. The MMER also considers any seepage and surface drainage water discharged from the site as being effluents. Each mining effluent must be discharged from an identifiable final discharge point. The MMER prescribes limits for arsenic, copper, cyanide, lead, nickel, zinc, total suspended solids ("TSS"), radium-226, and pH in mine effluent. Mines subject to the MMER are also required to conduct Environmental Effects Monitoring ("EEM") programs in accordance with prescribed criteria. The objective of EEM is to evaluate the effects of mining effluent on the receiving aquatic environment, specifically with regard to effects on fish, fish habitat, and the use of fisheries resources. The owner or operator must thus monitor effluent quality and flow at least once a week. The Regulations also include provisions for reducing metal sampling frequency to once per quarter on certain conditions. Sampling will go back to being once a week if these conditions are no longer met. Monthly acute lethality tests must be conducted on each discharged effluent, using the standardized 96-hour testing method on rainbow trout and conducting monitoring tests on Daphnia magna. Moreover, the MMER also includes a requirement that effluent be non-acutely lethal to rainbow trout. However, there is no requirement that the effluent be non-acutely lethal to Daphnia magna.

The MMER includes provisions (regulatory amendment) allowing the use of a natural water body frequented by fish for mine waste disposal. Thus, to be able to list the water body on Schedule 2 of the MMER, the proponent must conduct an assessment of alternatives for mine waste disposal in order to demonstrate that the selected mine waste disposal site ("MWDS") is the most environmentally, technologically and socio-economically sensible solution. The proponent must also develop a compensation plan to compensate fish habitat loss stemming from the use of the water body. The effluents discharged from the MWDS must meet the Regulations' discharge limits and other requirements.

WISCO plans on using bodies of water that could be home to different varieties of fish as a MWDS. It is recommended to proceed with fish and fish habitat surveys in order to determine where fish habitat occurs. If the surveys confirm the bodies of water or streams are fish habitat, WISCO must comply with the requirements of the MMER. The targeted water bodies must be listed on Schedule 2 of the Regulations and thus this requires an assessment of alternatives for tailings and mining waste rock disposal.

20.1.3 Environmental Permitting Schedule

The environmental permitting schedules, both provincial and federal, are presented in Tables 20.2 and 20.3. Steps that are shadowed in gray represent a statutory analysis delay. The timelines for issuing EA guidelines vary according to the regulatory agency. Usually, the Government of Québec is expected to issue generic guidelines within one month after the submission of the Notice of Intent but receipt of the guidelines for the components assessed under the JBNQA regime could take more time (several months). The over provincial EA takes in general around 800 days to be completed.

Given the 365-day timeline for completion of EAs under CEAA 2012, the guidelines from the CEA Agency should be received within 60 days of the decision to require an EA (which would follow a regulatory 45-day period for the review of the Project Description) following receipt of a satisfactory Project Description. The mine site project component may require additional delays (8 to 12 months) in the event the MMER is triggered.

Table 20.4 provides a list of required or potentially required permits and authorizations for each of the three main Project components.

Table 20.2 – General Provincial Environmental Assessment Schedule

Steps	Duration (days [a])
Notice of intent	15
Reception of the directive	22
Realization and submission of the impact assessment, including baseline studies	315
Compliance analysis and transmission of questions (first series)	56
Review and submission of answers to questions (first series)	60
Compliance analysis and transmission of questions (second series)	45
Review and submission of answers to questions (second series)	30
Receivability notice	30
Mandate for public information and consultation	15
Public information and consultation period	45
Public hearings 1 and 2 (with a pause of 21 days between each)[b]	22
Submission of the BAPE's report	53
Submission of the environmental analysis report	30
Issuing of the government decree / certificate of authorization	60
Total	**798**

Notes: a) Calendar days

b) There is no determined period of time for information and consultation under JBNQA.

Table 20.3 – General Federal Environmental Assessment Schedule

Steps	Duration (days)
Project notice	15
Post the notice of commencement on the Registry Internet Site	1
Comments on Project notice	10
Revised Project notice	5
Public and Aboriginal comment period in regards to the EIA guidelines	30
Final EIA guidelines produced and submitted to the proponent	30
Environmental impact assessment and submission	275
Analysis of concordance	20
Preparation of complementary information	48
Public and Aboriginal comment period in regards to the EIA summary	30
Federal examination and transmission of comments (first series)	49
Present the revised EIA or additional information	52
Revised EIA or additional information examination by the Agency and transmission of comments (second series)	20
Present the revised EIA or additional information	28
Revised EIA or additional information examination by the Agency	20
Preparation of the draft environmental assessment report	50
Public and aboriginal comment period in regards to the draft EAR	30
Present the final EAR to the Environment Minister	60
Minister recommendation	60
Total EA Process	**769**
Registration to the Schedule 2 of MMER [b]	365
Total with Registration to Schedule 2[c]	**1,134**

Notes: a) Calendar days
b) Administrative procedures for the deposit of deleterious substances into fish habitat.
c) Some steps overlaps which explain the overall duration is shorter than the sum of all steps.

Table 20.4 – List of Required or Potentially Required Permits and Authorizations for Each of the Four Main Project Components

Activities or Infrastructure	Permits – Certificate of Authorization (CoA) - Authorizations	Laws and Regulations	Government Authority	Timeframe
Provincial Government				
Mine Site and Related Infrastructure				
Mine development project (EA procedure)	CoA	Section 23 of the JBNQA	MDDELCC and KEQC	24-28 months
Construction of secondary roads	CoA	Section 22 of the EQA	MDDELCC	3 months
Clearing	Clearing permit for mining activities	Forest Act	MFFP	3 months
Work affecting fish habitats or wildlife habitats	CoA	Section 128.7 of the Act respecting the conservation and development of wildlife	MFFP	3 months
Location of piles and management facilities	Authorization	Section 241 of the Mining Act	MRN	3 months
Extraction or exploitation of surface mineral substances, borrow pits	Non-exclusive lease	Section 140 of the Mining Act	MRN	3 months
Construction of dykes	CoA	Section 5 of the Dam Safety Act	*Centre d'expertise hydrique du Québec* - MDDELCC	3 months
Setting up of infrastructure and equipment: Permanent or temporary roads Accommodation camp Rail loop and rail spur 161 kV – 35 kV substation Construction of dykes Mining operations (pits, piles, management facilities) Silo Drainage Water treatment plant, sedimentation ponds Storage sites Crusher, concentrator, conveyors	Multiple CoA	Section 22 of the EQA	MDDELCC	3 months for each request
Use of high-risk petroleum products	Permits	Section 120 of the Safety Code	*Régie du bâtiment du Québec*	1 month
Water intake, drinking water, wastewater, ducts	CoA	Section 32 of the EQA	MDDELCC	3 months
Storage, transport of explosives	Permits	Regulation under the Explosives Act	*Sûreté du Québec*	1 months
Mining operations	Depollution Attestation	Section 31.10 of the EQA	MDDELCC	4-5 months
Mining	Operation CoA	Section 22 of the EQA	MDDELCC	3 months
Installation of devices or equipment aimed at preventing, reducing or stopping contaminant emissions into the air	CoA	Section 48 of the EQA	MDDELCC	3 months
Railway to Schefferville				
Railway project (EA procedure)	CoA and Decree	Section 23 of the JBNQA Section 31.1 of the Environment Quality Act (EQA)	MDDELCC and KEQC	24-28 months
Construction of secondary roads	CoA	Section 22 of the EQA	MDDELCC	3 months
Clearing	Clearing permit for mining activities	Forest Act	MFFP	3 months
Work affecting fish habitats or wildlife habitats	CoA	Section 128.7 of the Act respecting the conservation and development of wildlife	MFFP	3 months
Extraction or exploitation of surface mineral substances, borrow pits	Non-exclusive lease	Section 140 of the Mining Act	MRN	3 months
Permanent or temporary roads	CoA	Section 22 of the EQA	MDDELCC	3 months
Railway general construction and associated infrastructure	CoA	Section 22 of the EQA	MDDELCC	3 months
Drainage	CoA	Section 22 of the EQA	MDDELCC	3 months

Activities or Infrastructure	Permits – Certificate of Authorization (CoA) - Authorizations	Laws and Regulations	Government Authority	Timeframe
Provincial Government				
Pelletizing Plant				
Pellet Plant (EA procedure)	Government decree	Section 31.1 of the Environment Quality Act (EQA)	MDDELCC	24-28 months
Construction of secondary roads	CoA	Section 22 of the EQA	MDDELCC	3 months
Clearing	Clearing permit for mining activities	Forest Act	MFFP	3 months
Work affecting fish habitats or wildlife habitats	CoA	Section 128.7 of the Act respecting the conservation and development of wildlife	MFFP	3 months
Extraction or exploitation of surface mineral substances, borrow pits	Non-exclusive lease	Section 140 of the Mining Act	MRN	3 months
Setting up of infrastructure and equipment: Permanent or temporary roads Conveyors and handling facilities Electrical substation Silo Drainage Water treatment plant, basins Storage sites	Multiple CoA	Section 22 of the EQA	MDDELCC	3 months
Use of high-risk petroleum products	Permits	Section 120 of the Safety Code	*Régie du bâtiment du Québec*	1 month
Installation of devices or equipment aimed at preventing, reducing or stopping contaminant emissions into the air	CoA	Section 48 of the EQA	MDDELCC	3 months
Federal Government				
Mine Site and related Infrastructure				
Federal environmental assessment	Authorization	CEAA	CEAA	24 to 28 months
Activities affecting fish habitat	Authorization	Section 35.1 and 36.3 of the Fisheries Act	DFO	5 months
Disposal of mine tailings and/or deleterious substances into fish habitat	Assessment of alternatives	Section 36.3 of the Fisheries Act, MMER	Env. Can. and DFO	16 to 24 months
Railway to Schefferville				
Federal environmental assessment	Authorization	CEAA	CEAA	24 to 28 months
Activities affecting fish habitat	Authorization	Section 35.1 and 36.3 of the Fisheries Act	DFO	5 months
Pelletizing Plant				
Federal environmental assessment	Authorization	CEAA	CEAA	24 to 28 months
Activities affecting fish habitat	Authorization	Section 35.1 and 36.3 of the Fisheries Act	DFO	5 months

20.2 General Project Description

20.2.1 Project Elements

The Project consists in the establishment of an open-pit mine and an industrial complex for the production of iron concentrate at the mining property located approximately 85 km to the north of Schefferville. The open pit will be mined conventionally as per this type of pit. Drilling and blasting will be used to extract ore and waste rock. The material will be loaded onto trucks then transported to crushing plant. A conveyor will transport the ore to the run of mine stockpile and from there to the concentrator. The waste rock will be disposed on the nearby piles. The plant will use a conventional gravity separation circuit and as well as magnetic separation to increase iron concentrations and produce iron concentrate. The final concentrate will be filtered to produce a low water content concentrate allowing bulk shipping in rail cars and also preventing freezing during winter. Tailings will be thickened before being disposed of in the tailings management facility (“TMF”).

The TMF will be located in the mining property’s western area. Sedimentation and polishing ponds will be installed downstream to collect water from the tailings. This water will be treated before being discharged into the receiving environment to comply with applicable regulations and requirements, such as those contained in Directive 019 on the mining industry and the MMER.

So far, no geochemical characterization study has been conducted on the waste rock to determine the waste rock an acid-generating potential. A more extensive characterization program will be conducted in the future.

The Project also includes the construction a 91.5-km long railway to Schefferville in order to transport the iron ore concentrate to the multi-user terminal in Sept-Îles. From the terminal, the ore will be sent to one or two pelletizing plants. Power will be provided at the mining site using a new 215 km long 161 kV power line from the Brisay dam while the pelletizing plant will be powered by a 3 km long 161 kV power line. A 161 kV – 35 kV substation will be required at both the mining site and near the pelletizing plants.

The Project’s main structures include:

- Mine Site and Related Infrastructure;
  - An open pit;

- A waste rock disposal site;
- An overburden stockpile;
- A crushing plant, including two gyratory crushers;
- A concentrator;
- An office;
- Workshops;
- Drainage ditches and settling ponds;
- An haulage road between the mine pit and the crushing plant;
- Two conveyors including cones and screens – to the crushing plant;
- A run of mine ore stockpile, including conveyors to the concentrator;
- An iron concentrate silo or emergency stockpile;
- A rail loop, rail spurs (to the explosive storage, the maintenance workshop and for unloading);
- An accommodation camp designed for 300 workers;
- A 161 kV – 35 kV substation;
- An explosive preparation and storage area;
- A fuel storage area;
- A security gate house;
- A pump house (source of water for the concentrator and the accommodation camp);
- A tailings management facility, including dikes and sedimentation / polishing pond;
- A water management system, including a treatment plant (contact water);
- A 215 km long 161 kV power line from the Brisay dam to the new substation at the mine site;

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- Railway to Schefferville;
  - A 91.5-km long rail way from the mine site to the existing railway in Schefferville;

- Pelletizing Plant
  - Two pellets plants, each a 8.15 Mtpy capacity
  - A security gate house
  - An administration building
  - Drainage ditches and settling ponds
  - A 3-km long 161 kV power line and a 161 kV – 35 kV substation
  - A pump house near the Sainte Marguerite shoreline
  - A water treatment unit inside the pellet plant

20.2.2 Main Sources of Impacts

The main sources of impacts during the mine construction, operation and closure & rehabilitation activities are given in Table 20.5.

Table 20.5 – Construction, Operation and Closure & Rehabilitation Main Sources of Impacts

Construction	Operation	Closure and Rehabilitation
Mine Site and Related Infrastructure		
Site preparation Temporary and permanent access roads Borrow pit extraction Runoff water management Drilling and blasting Equipment / material transportation General construction activity (i.e.: buildings, dikes, substation) Waste management Workforce	Ore extraction and processing Ore haulage and piling Waste rock stockpiling Tailings management Mine dewatering, mine site runoff and processing water management Maintenance activities (i.e.: roads, buildings, dikes, water management system) Progressive site rehabilitation Waste management Workforce	Dismantling of buildings and facilities Management of contaminated soil (if any) Site rehabilitation Water management and retention structure maintenance
Railway to Schefferville		
Site preparation Temporary access roads Borrow pit extraction Drilling and blasting General construction activity Equipment / material transportation Waste management Workforce	Railway operation Maintenance Vegetation control within the right of way	Dismantling of facilities Management of contaminated soil (if any)
Pelletizing Plant		
Site preparation Temporary and permanent access roads Borrow pit extraction Runoff water management General construction activity Equipment / material transportation Local Circulation Waste management Workforce	Plant operation Conveyance (train / plant / multi-user terminal) Maintenance Waste management Workforce	Dismantling Management of contaminated soil (if any) Waste management Site rehabilitation

20.3 Environmental Settings and Social Requirement

No environmental baseline study has been conducted on the Rainy Lake property and therefore the information given in this section was collected from various database and existing data collected for other projects located in the Schefferville area. Desktop research and field surveys are required to complete the environmental and social baseline information for the Project. Moreover, consultation/information activities will be required from this stage of the Project until its completion.

20.3.1 General Description

The Property is located in northeastern Québec approximately 85 km northwest of the Town of Schefferville, Québec and 220 km northeast of the Brisay dam (Hydro Québec power dam in Québec). The Town of Schefferville is served by an airport with daily flight to Sept-Îles. There are no roads connecting the Rainy Lake property, so the Property can only be accessed by helicopter or floatplane in the summer or ski plane in the winter.

From north to south, the Project development area spans across different natural provinces. The northern portion of the Project area is located in the Ungava Bay Basin natural province. This province is a high plateau inclined to the north. Continuing southwards, the Project area crosses the North of Québec Central Plateau natural province, a large hilly area with shallow lakes and peat bogs. The southern portion of the Project development area goes through the Central Laurentides natural province, characterized by a hydrographic network that is oriented north-south and that flows toward the St. Lawrence River.

The Project is crossing a large area where the climate is sub-arctic from Sept-Îles to the mine site, with average temperatures varying between -43 °C in February and 32 °C in July. Extremes total precipitations (rain and snow) varying between 8 mm in February to 114 mm in July. The maximum monthly average snowfall (derived from Schefferville and Kuujjuaq) is 57 cm.

Perennial frozen ground or permafrost is discontinuous and occurs in isolated areas, primarily in wetlands. Overall, the continuity and the thickness of permafrost is increasing northward through Québec-Labrador. Permafrost in the vicinity of Schefferville, Labrador City, and Fermont ranges from less than 5 m to 50,100 m thick. A permafrost depth of about 120 m was observed in an area about 30 km north of Schefferville, and the ground temperature was as low as - 2.8 °C at a depth of 15 m.

The James Bay and Northern Québec regions, more than 1 million km², represent about two-thirds of Québec's total land area. They are located between the 49th and 62nd parallels. Human presence in these regions dates back about 4,000 years. The Cree, Inuit, Naskapi, and Innus are the main First Nations communities in these areas.

Mining activities are ongoing in the vicinity of the Project development area, for example in Schefferville and in Fermont (iron ore). Extraction activities (i.e.: from sandpits, quarry) are taking place. Numerous exploration activities (mining claims) are being conducted near to the Project development area. Production of electricity is also significant in the vicinity of the Project development area, from North to South: the La Grande Complex, the Hart-Jaune hydroelectric power plant located on the Petit-Manicouagan Reservoir, the Sainte-Marguerite River Complex and the Toulnoustouc River development.

20.3.1.1 Mine Site and Related Infrastructure

ENVIRONMENTAL SETTINGS

The Project development area is located in an area quite similar to that of a pristine environment. The Property is located within a relatively rugged physiography with rolling hills and valleys reflecting the structure of the underlying bedrock. Elevation in the mine development area can vary from 440 m on the east of the open pit up to 840 m at the highest point.

Local climate is characterized by continental long and cold winters and short and mild summers. According to the Natural Resources Canada (1993), the projected mine is located in a sporadic discontinuous permafrost area (surface cover between 10 and 50%).

A subarctic forest climate characterizes the mine site. The project site is located in the spruce-lichen bioclimatic domain within the boreal taiga subzone, which extends between the 52nd and 55th parallels (Saucier et al., 2003) in Québec. It is characterized by low density stands and vast wetland complexes.

Located between the Caniapiscau River and the Swampy Bay River, the mine site is situated in the Caniapiscau watershed where the water flows north to the Ungava Bay. The mine site is located at the edge of two sub-watersheds. One sub-watershed flows through in the Goodwood River and then into the Caniapiscau River. The other flows into the Swampy Bay River, which also connects to the Caniapiscau River. Rivers and lakes are numerous in the mine site, providing habitat for fish species. Wildlife and fish species are typical of northern environments. The George River Herd is the only caribou population likely to be found within the Project's area.

Despite that no environmental baseline studies were conducted in the mine development area, rare plant or species of conservation concerns ("SOCC") and species at risk ("SAR") may occur. Field surveys will be required to determine if such plant or wildlife species occur in the Project area. Among SOCC and SAR wildlife species found at this latitude, golden eagle (Aquila chrysaetos), the peregrine falcon (Falco peregrinus), the harlequin duck (Histrionicus histrionicus), the bald eagle (Haliaeetus leucocephalus), the short-eared owl (Asio flammeus), the rusty blackbird (Euphagus carolinus) and the barrow's goldeneye (Bucephala islandica) could occur in the mine site area.

COMMUNITY AND SOCIO-ECONOMIC

The Property is uninhabited and the closest inhabited areas are:

- The Indian Reserve of Kawawachikamach, located about 80 km south-east of the project site and inhabited by the Naskapi Nation of Kawawachikamach;
- The town of Schefferville, about 85 km south-east of the project site;
- The Indian Reserve of Matimekosh and Lac-John, located about 90 km south-east of the project site and inhabited by the Matimekush-Lac John Innu Nation;
- The Northern Village of Kuujjuaq, about 300 km north of the project site.

Land tenure and organization in the mine site area is governed under JBNQA. The Project is encompassed within the "Territoire non-organisé Rivière-Koksoak" (Koksoak river unorganized territory) which is administrated by the Kativik Regional Government (“KRG”).

The only access to the mine site is by air or snowmobile in winter. The closest railway link is Schefferville to Sept-Îles via, Wabush and Labrador City (TSH and QNS&L railways). The closest airport is located in Schefferville.

There is no intensive land use by the communities, which are all located more than 80 km away. The territory is most likely used for fishing and hunting by First Nation communities (Naskapi, Innu and possibly Inuit). In addition, there is recreational land use, mostly in the area of the Caniapiscau River (i.e.: hiking, camping, canoe).

Although no protected areas are directly encompassed within the mine site development area, the Collines Ondulées Provincial Park, located approximately 25 km to the north-east of the mine site is the only legally protected area present within a 50 km radius of the mine development area. Also, a land area, set aside as ecological reserve, is located in this radius: the Canyon-Eaton, approximately 25 km north west of the project site. Lac Sévigny projected biodiversity reserve is located at approximately 90 km to the west of the Project area. A second land set aside as an ecological reserve, the Lac Cambrien national park is located 100 km to the north-west of the projected mine site.

20.3.1.2 Railway to Schefferville

ENVIRONMENTAL SETTINGS

The proposed 91 km long railway between the mine site development area and the existing railway in Schefferville is located in an area quite similar to that of a pristine environment.

The railway development area is also part of the Hudsonian Vegetation Region, where the subarctic forest characterized by balsam fir, tamarack, white spruce, black spruce, paper birch, balsam poplar and trembling poplar. Peatlands are common in the railway development area.

The railway development area is entirely located in the Caniapiscau watershed that flows to the Ungava Bay. Rivers and lakes are numerous in the mine site, providing habitat for fish species. Wildlife and fish species are typical of northern environments. The George River Herd is the only caribou population likely to be found within the Project's area.

Despite that no environmental baseline studies were conducted in the railway development area, rare plant or SOCC and SAR may occur. Field surveys will be required to determine if such plant or wildlife species occur in the Project area. Among SOCC and SAR wildlife species found at this latitude, golden eagle, the peregrine falcon, the harlequin duck, the bald eagle, and the short-eared owl, the rusty blackbird and the barrow's goldeneye could occur in the mine site area.

COMMUNITY AND SOCIO-ECONOMIC

The railway development area extends across lands used for fishing and hunting by First Nation communities (Naskapi, Innu and possibly Inuit). Schefferville, near the border with Newfoundland and Labrador, is the only settlement of importance in the railway development area. Many recreational leases are found in the vicinity of Schefferville while other are scattered throughout the surrounding environment of the Project development area. The main airfield in or near the study zone is located in Schefferville.

Since the main water courses were natural transportation routes for the First Nations, the river banks are likely to contain a large number of archaeological sites. An archaeological potential study is recommended during the next step of the Project.

No landscape environmental baseline study was conducted in the mine site development area. The area is representative of the entire landscape of the plateau forming the center of Québec and the Eaton Canyon on the Caniapiscau River is the single exceptional site in the area.

20.3.1.3 Pelletizing Plant

ENVIRONMENTAL SETTINGS

The precise location of the pelletizing plant is still unknown but the plant would most likely be located in Sept-Îles, near to the multi-user terminal where the ore will be railed to from the mine site. The multi-user terminal area is located in a well-developed industrial area established on the south side of the Baie des Sept-Îles, in the Pointe-Noire area, at 10 km to the east of the Sainte-Marguerite River.

Sept-Îles has a borderline subarctic climate, despite being located at around only 50 degrees latitude. The two main seasons are summer and winter, as spring and autumn are very short transition seasons lasting only a few weeks. Winters are long, cold, and snowy, lasting from late October to late April, but milder than more inland locations, with a January high of −9.8 °C and a January low of −20.9 °C. Overall precipitation is unusually high for a subarctic climate, and snow totals correspondingly heavy, averaging 385 cm per season with an average maximum depth of 0.5 m. Summers are mild, with a July high of 19.6 °C; summers thus display stronger maritime influence than do winters. Precipitation is significant year-round, but it is lowest from January to March.

COMMUNITY AND SOCIO-ECONOMIC

Bordering the 50th parallel, Sept-Îles lies at the heart of the vast area of Duplessis region, on the Côte-Nord. Sept-Îles is bordered with the Laurentian Highlands to the North, Gallix to the West and Moisie to the East and covers a 2,182km² territory bordering the sea. Sept-Îles is home to more than 26,000 inhabitants, or nearly 30,000 including the Innu communities of Uashat mak Mani-Utenam.

Sept-Îles has a deep-water port where the commodities and goods necessary to the development of its major industries are transported. Iron ore concentrate from IOC activities in Labrador City are transported by the Québec North Shore and Labrador Railway and are shipped to many markets around the world from Sept-Îles port facilities. Iron ore from Wabush is also shipped at the Point-Noire port facilities. The Aluminerie Alouette, in activity since 1992 has a large part in the local employment since his construction started in 1989. Since a major expansion in 2005, it is now the largest primary aluminum smelter in the Americas.

20.3.1.4 Data Gaps

Data will need to be collected, in terms of biophysical and social environment, in order to support the EA procedure and fulfill provincial and federal government authority requirements. The recommended environmental baseline studies are provided in Table 20-6. In some cases, a desktop research should be done prior to any field survey to fill in those gaps.

Table 20.6 – Environmental Baseline Studies Recommended to Fill in Data Gaps

Environmental Baseline Study	Mine Site and Related Infrastructure	Railway to Schefferville	Pelletizing Plant
Vegetation, incl. wetland and rare plant	x	x	x
Avifauna, incl. migratory bird	x	x	TBD
Fish and fish habitat	x	x	TBD
Wildlife and habitat	x	x	TBD
Vegetation / Habitat photo-interpretation	x	x	TBD
Geochemical study of tailings and waste rock	x		
Hydrogeological study	x		x
Air quality modelling	x	x	x
Noise Modelling	x	x	x
Hydrology and surface water quality	x	x	x
Socio-economic consultation	x	x	x
Aboriginal consultation	x	x	x
Archaeological potential study	x	x	x
Landscape features survey	x	x	x

TBD: To be determined: the preliminary data regarding this Project component are insufficient to determine what environmental baseline information will be required.

20.4 Anticipated Issues or Impacts

Given the Project components described in the Section 20.2 and based on available data on the environment (Section 20.3), the Project's main anticipated issues or impacts has been determined for the construction, operation as well as closure and rehabilitation activities and are presented hereinafter. In addition, compensation program will most likely be needed for the Full Moon and is shortly discussed in this section.

20.4.1 Identification of Environmental and Social Impacts

The Project's related environmental and social issues for each of the three main Project components as well as the main mitigation measures are described in the following tables together with preliminary mitigation measures (Table 20 7 to Table 20 9). The optimization of the Project design will also be aimed at reducing the potential impact of environmental and social issues.

Table 20.7 – Mine Site and Related Infrastructure Anticipated Issues or Impacts

Environmental Components	Anticipated Issues or Impacts	Mitigation Measures
Atmospheric Environment	Emission of dust, GHG and other contaminants into the ambient air generated by the vehicles during construction, operation and closure Emission of dust and other pollutants during the construction activity Effect of noise during the construction activity Effect on ambient sound and air quality due to mining and concentrating operations and transportation of concentrate from the site. Emission of dust from water rock, overburden, ore piles due to wind erosion	Use dust suppressor The machinery used shall meet Environment Canada's emission standards for on-road and off-road vehicles Minimize machinery idling time Implement a dust management plan
Hydrology	Changes in the local flow regime during both the construction and operation	Temporarily disturbed flows will be progressively re-established after the work to avoid any sudden flow changes
Hydrogeology	Increase in runoff rate during both the construction and operation Changes in the local groundwater flow regime during both the construction and operation	During construction, and as needed during the operation, a network of monitoring wells will be established around the new infrastructure, to check for changes in water levels
Surface and Groundwater Quality	Risk of groundwater contamination through accidental spillage of oils, hydrocarbons or any other hazardous substances – during all mine life Discharge of fine particles and woody debris into the water during construction, operation and rehabilitation Potential for percolation during closure	The number of machinery fuelling sites will be minimized to reduce the number of at-risk sites. Any eventual leaks due to faulty valves or human error will be reported to the environmental overseer and, depending on the case, to maintenance for repair. Soaked surface soil will be immediately dug up and disposed of as per regulations
Soil Quality	Risk of soil contamination through the accidental spillage of oils, hydrocarbons or any other dangerous liquids – during all mine life	The number of machinery fuelling sites will be minimized to reduce the number of at-risk sites. Any eventual leaks due to faulty valves or human error will be reported to the environmental overseer and, depending on the case, to maintenance for repair. Soaked surface soil will be immediately dug up and disposed of as per regulations
Wetlands and Rare Plant	Loss of area during construction Disturbance of vegetation during operation	Minimize the new infrastructure's total footprint

Environmental Components	Anticipated Issues or Impacts	Mitigation Measures
Fish and Fish Habitat	Changes to fish habitat and its quality, even destruction of fish habitat and fish mortality during construction Fish habitat alteration due to mining effluent	Minimize as much as possible encroachment in lakes and watercourses. Reuse of process water Rigorous water management
Bird and Wildlife	Loss of habitat during construction Disturbance of wildlife during operation	The construction work will be conducted if possible outside the breeding season of the main species present at this latitude
Species at Risk	Unknown at this stage- lack of data	Reduce the Project's footprint.
Land Use	Changes in the practice of certain wildlife harvesting activities – during all mine line	At the closure of the mine, the site rehabilitation will restore a more natural state to the site adapted to the surroundings
Economy, Employment and Business	Economic spinoffs for Schefferville, Fermont and Côte-Nord suppliers	
Aboriginal Communities	Encroachment on traditional lands Potential interaction with hunting route	Signing of an agreement with the affected Aboriginal community
Landscape	Changes to landscape units and associated visual fields	During the design phase, the configuration of piles and the tailings management facility as much as possible in harmony with the surrounding relief's natural topography
Historic and Heritage Resource	Unknown at this stage- lack of data	If, during the course of the work, vestiges of historical or archaeological interest were to be discovered, the work site overseer would be immediately informed and provisions made for the site's protection

Table 20.8 - Railway to Schefferville Anticipated Issues or Impacts

Environmental Components	Anticipated Issues or Impacts	Mitigation Measures
Atmospheric Environment	Emission of dust, GHG and other contaminants into the ambient air generated by the vehicles during construction Effect of noise during the construction Effect on ambient sound and air quality during operation of the railway	Use dust suppressor The machinery used shall meet Environment Canada's emission standards for on-road and off-road vehicles Minimize machinery idling time
Hydrology	Insignificant	
Hydrogeology	Insignificant	
Surface and Groundwater Quality	Risk of groundwater contamination through accidental spillage of oils, hydrocarbons or any other hazardous substances – during construction Discharge of fine particles and woody debris into the water during construction	The number of machinery fuelling sites will be minimized to reduce the number of at-risk sites Any eventual leaks due to faulty valves or human error will be reported to the environmental overseer and, depending on the case, to maintenance for repair Soaked surface soil will be immediately dug up and disposed of as per regulations
Soil Quality	Risk of soil contamination through the accidental spillage of oils, hydrocarbons or any other dangerous liquids – during construction	The number of machinery fuelling sites will be minimized to reduce the number of at-risk sites Any eventual leaks due to faulty valves or human error will be reported to the environmental overseer and, depending on the case, to maintenance for repair Soaked surface soil will be immediately dug up and disposed of as per regulations
Wetlands and Rare Plant	Loss of area during construction Control of vegetation growth during operation	Minimize the new infrastructure's total footprint
Fish and Fish Habitat	Fish habitat alteration at stream crossing (potential turbidity, siltation and contamination)	Control runoff water with sediment barrier or pond Stabilize slope as soon as possible
Bird and Wildlife	Loss of habitat during construction Disturbance of wildlife during vegetation control activity (modification of the habitat within the right of way)	The construction work will be conducted if possible outside the breeding season of the main species present at this latitude
Species at Risk	Unknown at this stage- lack of data	Reduce the Project's footprint.

Environmental Components	Anticipated Issues or Impacts	Mitigation Measures
Land Use	Changes in the practice of certain wildlife harvesting activities – during all mine life	Produce an analysis of alternative to determine the optimal route
Economy, Employment and Business	Economic spinoffs for Schefferville, Fermont and Côte-Nord suppliers	
Aboriginal Communities	Potential interaction with hunting route	Signing of an agreement with the affected Aboriginal community Produce an analysis of alternative to determine the optimal railway route
Landscape	Changes to landscape units and associated visual fields	Produce an analysis of alternative to determine the optimal route
Historic and Heritage Resource	Unknown at this stage- lack of data	If, during the course of the work, vestiges of historical or archaeological interest were to be discovered, the work site overseer would be immediately informed and provisions made for the site's protection

Table 20.9 - Pelletizing Plant Anticipated Issues or Impacts

Environmental Components	Anticipated Issues or Impacts	Mitigation Measures
Atmospheric Environment	Emission of dust, GHG and other contaminants into the ambient air generated by the vehicles during construction, operation and closure Effect of noise during the construction and operation	Use dust suppressor The machinery used shall meet Environment Canada's emission standards for on-road and off-road vehicles Minimize machinery idling time Implement a dust management plan Use of appropriate measure to avoid noise disturbance
Hydrology	Unknown at this stage- lack of data	
Hydrogeology	Increase in runoff rate during both the construction and operation Changes in the local groundwater flow regime during both the construction and operation	During construction, and as needed during the operation, a network of monitoring wells will be established around the new infrastructure, to check for changes in water levels
Surface and Groundwater Quality	Risk of groundwater contamination through accidental spillage of oils, hydrocarbons or any other hazardous substances – during construction and operation Discharge of fine particles and woody debris into the water during construction	The number of machinery fuelling sites will be minimized to reduce the number of at-risk sites. Any eventual leaks due to faulty valves or human error will be reported to the environmental overseer and, depending on the case, to maintenance for repair. Soaked surface soil will be immediately dug up and disposed of as per regulations Rigorous water management
Soil Quality	Risk of soil contamination through the accidental spillage of oils, hydrocarbons or any other dangerous liquids – during all mine life	The number of machinery fuelling sites will be minimized to reduce the number of at-risk sites. Any eventual leaks due to faulty valves or human error will be reported to the environmental overseer and, depending on the case, to maintenance for repair. Soaked surface soil will be immediately dug up and disposed of as per regulations
Wetlands and Rare Plant	Unknown at this stage- lack of data	Minimize the new infrastructure's total footprint Avoid wetlands and sensitive habitat
Fish and Fish Habitat	Unknown at this stage- lack of data Alteration due to presence of effluent	Avoid encroachment in lakes and watercourses. Reuse of process water
Bird and Wildlife	Unknown at this stage- lack of data	The construction work will be conducted if possible outside the breeding season of the main species present at this latitude
Species at Risk	Unknown at this stage- lack of data	Reduce the Project's footprint

Environmental Components	Anticipated Issues or Impacts	Mitigation Measures
Land Use	Unknown at this stage- lack of data	At the closure of the mine, the site rehabilitation will restore a more natural state to the site adapted to the surroundings
Economy, Employment and Business	Economic spinoffs for Sept-Îles and Côte-Nord suppliers New local employment opportunities Pressure on health services Lack of accommodation facilities	
Aboriginal Communities	Unknown at this stage- lack of data	Signing of an agreement with the affected Aboriginal community
Landscape	Changes to landscape units and associated visual fields	During the design phase, the configuration of the plant and the power line as much as possible in harmony with the surrounding relief's natural topography
Historic and Heritage Resource	Unknown at this stage- lack of data	If, during the course of the work, vestiges of historical or archaeological interest were to be discovered, the work site overseer would be immediately informed and provisions made for the site's protection

20.4.2 Compensation Program

FISH HABITAT

A program that will compensate for serious harm caused to fish is also required as per Section 27.1 of the MMER to counter balance fish habitat losses associated with the storage of a deleterious substance in one or more water bodies sheltering fish.

Further, Subsection 35(2) of the Fisheries Act requires fish habitat compensation to compensate for the serious harm to fish associated with construction activities that encroach into fish habitat, such as dikes, bridges or culverts. A compensation measure is one that counterbalances unavoidable serious harm to fish resulting from a project with the goal of maintaining or improving the productivity of the commercial, recreational or Aboriginal fishery (DFO 2013). Compensation measures are intended to provide tangible conservation outcomes for fish and fish habitat that may reasonably be expected to counterbalance the loss of fish habitat and fisheries productivity as a result of the negative impacts of projects. Compensation is considered as a management option only if proposed activities affecting habitat cannot be avoided by redesign, relocation, or by mitigating potential impacts.

A compensation plan must be prepared to demonstrate that the measures and standards will be fully applied to first avoid, then mitigate, and finally offset any residual serious harm to fish that are part of or support commercial, recreational or Aboriginal fisheries. Proponents are also required to demonstrate that the compensation measures will maintain or improve the productivity of fisheries.

WETLANDS

Under the Environment Quality Act and the Act Respecting Compensation Measures for the Carrying out of Projects Affecting Wetlands or Bodies of Water, the MDDELCC is responsible to authorize or not any projects affecting wetlands, such as ponds, swamps, marshes and bogs. These measures specifically target the restoration, creation, protection or ecological enhancement of wetlands as well as water and upland environments, in the latter case nearby the affected area. In the case of projects affecting wetlands, the MDDELCC favours an “avoid-minimize-compensate” sequence of mitigation, which reduce wetland loss (avoid), proposes design and methods that optimize the project design in relation to wetlands while reducing impacts on the receiving environment (minimize) and establishes the environmental acceptability of proposed compensation measures (compensate).

A compensation plan helps to determine corrective measures to adopt and shortly describes the nature of the proposed compensation tasks. The plan specifies how these tasks are to be implemented and monitored.

20.5 Preliminary Environmental Management Plan

20.5.1 Environmental Monitoring

There will be environmental monitoring during the Project's construction phase, to ensure compliance with environmental commitments and obligations. It also aims to verify the incorporation of proposed mitigation measures into the Project and to ensure compliance with laws, regulations and other environmental considerations contained in the plans and specifications. This general environmental monitoring will be conducted by the Project proponent. Monitoring tasks include:

- Monitoring and overseeing all tasks requiring preventative, mitigation or corrective measures with regard to the environment;
- Ensuring that the work is carried out in compliance of the laws, regulations and conditions of the certificates of authorization;
- Monitor infrastructure under construction;
- Update project-generated hazardous waste storage and disposal condition monitoring registries;
- Monitor fuelling procedures for gas-powered equipment used for the project;
- Guide and monitor procedures to implement in case of accidental spillage, including monitoring temporary storage conditions for contaminated soil, if applicable;
- Ensure that schedules are met with regard to biologically-based restriction periods and adequate environmental monitoring related to the Project.

One of the surveillance program's activities will be to ensure that all required authorizations and permits have been requested and all certificates and permits have been duly obtained.

During the work, the mitigation measures shall be rigorously followed, especially when working near watercourses and water bodies. Care will be taken to ensure that as little suspended solid as possible is discharged into the water, as well as avoiding any accidental leaking of oil products.

In general, the person in charge of the monitoring shall regularly visit work areas, take note of the rigorous compliance by workers with various commitments, obligations, measures and other requirements, assess the quality and effectiveness of applied measures and note any non-compliance observed. He shall then inform the worksite foreman of his observations so that the appropriate corrective measures be agreed upon and implemented as soon as possible, if applicable.

20.5.2 ENVIRONMENTAL MONITORING PROGRAM

An environmental monitoring program for the entire mining site shall be developed, both for the production and the closure & rehabilitation activities. In accordance with federal and provincial requirements, the mining site's environmental monitoring program will cover the following aspects:

- Mining effluent quality;
- Domestic effluent quality;
- Surface water quality;
- Groundwater quality;
- Stability of retaining structures;
- Air quality;
- Monitoring of compensatory development and biological monitoring;
- Social environment;
- Post-closure monitoring.

20.6 Conceptual Closure and Rehabilitation Plan

20.6.1 Legislation

The Mining Act (CQLR, C. M-13.1) is another important piece of provincial legislation that concerns the management of mining activities in the Province of Quebec. "The purpose of this Act is to promote mineral prospecting, exploration and development in keeping with the principle of sustainable development, while ensuring that Quebecers get a fair share of the wealth generated by mineral resources and taking into account other possible uses of the territory" (s.17).

Section 232.1 of the Act states that:

> "Every operator who engages in mining operations determined by regulation in respect of mineral substances listed in the regulations must submit a rehabilitation and restoration plan to the Minister for approval and carry out the work provided for in the plan. The obligation shall subsist until the work is completed or until a certificate is issued by the Minister under Section 232.10."

Hence, as part of the project, a rehabilitation plan will be prepared (and approved by the MERN). The rehabilitation and restoration plan should be elaborated in accordance with the provincial Guidelines for Preparing a Mining Site Rehabilitation Plan and General Mining Site Rehabilitation Requirements (MRN[1] and MEF[2], 1997) which provides the proponents with the rehabilitation requirements. The financial feasibility of the project will have to take into account the costs of all the work needed for the rehabilitation of the mining site.

20.6.2 General Principles

The main objective of mine rehabilitation is to restore the site to a satisfactory condition by:

- Eliminating unacceptable health hazards and ensuring public safety;
- Limiting the production and circulation of substances that could damage the receiving environment and, in the long-term, trying to eliminate maintenance and monitoring;
- Restoring the site to a condition in which it is visually acceptable to the community;

Reclaiming the areas where infrastructures are located (excluding the accumulation areas) for future use.

Specific objectives are to:

- Restore degraded environmental resources and uses of the land;

[1] Ministère des ressources naturelles (active designation from 1994-1999)

[2] Ministère de l'environnement et de la faune (active designation from 1994-1999)

- Protect important ecosystems and habitats of rare and endangered flora and fauna, which favours the reestablishment of the biodiversity;
- Prevent or minimise future environmental damage;
- Enhance the quality of specific environmental resources;
- Improve the capacity of eligible organizations to protect, restore and enhance the environment; and
- Undertake waste avoidance projects and prevent and/or reduce pollution.

The general guidelines of a rehabilitation plan include:

- Promotion of progressive restoration to allow a rapid reinstatement of the biodiversity;
- Monitoring and surveillance program;
- Maximisation of the recovery of previous land uses;
- Research new vocations for land uses;
- Habitat rehabilitation using operational environmental criteria;
- Ensure sustainability of the results of the restoration efforts.

The mining site rehabilitation plan focuses on land reclamation, reclamation of the Tailings Storage Facility ("TSF") and waste rock piles as well as water ponds, and surface drainage patterns to prevent erosion. At the end of the mining activities, the rehabilitation plan ensures a minimum of disturbance over the area of the mine site. The site will need to be brought to the MERN standards before it can be returned to the Government and that the mine owner would not be found responsible for its care.

20.6.3 Environmental Aspects and Assumptions

This section presents the environmental aspects driving the rehabilitation concepts.

20.6.3.1 Drainage

Whenever possible, the surface water drainage pattern will be restored to conditions resembling the original hydrological system.

20.6.3.2 Topsoil Management

During site construction and ore body stripping, the overburden and topsoil will be salvaged separately and used for revegetation purposes. In the case where overburden would still be in place at the end of the mining operations, the slopes of the overburden storage area would be seeded.

20.6.3.3 Waste Management

Demolition waste will be:

- Decontaminated when required;
- Recycled when cost-effective;
- Disposed of/or burned on site;
- Buried at an appropriate site.

All non-contaminated wastes will be sent to a landfill.

20.6.3.4 Hazardous Materials

Facilities containing petroleum products, chemicals, solid wastes, hazardous wastes, and/or contaminated soil or materials will be dismantled and managed according to regulatory requirements.

Final restoration of the mine site and port facilities will be completed within three years following the end of commercial production.

20.6.3.5 Tailings and Waste Rock Characteristics and Disposal Requirements

Environmental considerations relative to TSF and waste rock piles are outlined below.

20.6.3.6 Geotechnical Studies and Stability Assessment

Geotechnical studies to assess the ground conditions at the site of the proposed tailings management facility and waste rock dumps will have to be carried out.

The selection of the site and design of the peripheral dykes will need to be optimized when that information becomes available.

Tests will be necessary to assess the geotechnical characteristics of the foundations of both the tailings impoundment area itself and, in particular, on the tailings retaining dykes.

At closure, the stability of such infrastructure will be assessed for long-term environmental and structural integrity and will assess the effect of seismic hazard.

20.6.3.7 Tailings Properties

As the TSF design remain conceptual, we can only assume that the tailings properties will be similar to those of other iron ore processing facilities, consisting of a pulp of fine particles.

20.6.3.8 Geochemistry

Mineral deposits of the Project are similar to other properties mined in the Schefferville area until the years 1980's (e.g. Iron Ore of Canada properties, Hollinger properties, etc.). Historically, these operations have not shown signs of acid generation for the waste rock or the ore and are not susceptible to metal leaching.

The acid generating and metal leaching potentials will have to be confirmed for the feasibility study using the appropriate test protocols.

20.6.3.9 Progressive Rehabilitation and Restoration

Progressive restoration is always favoured in order to rapidly reach the objectives of the rehabilitation program and help in an early habitat reestablishment to increase biodiversity.

For the Project, it is planned that the tailings will be stored into two distinct ponds: one for coarse tailings and the other for fine tailings. Progressive restoration of each tailings cell will not be possible since each tailings type will be stored in one big cell forming the whole tailings accumulation area. The complete restoration of the two tailings ponds may be achieved after the end of mine life.

Progressive restoration could be possible for the waste rock piles (or sectors in it), when at their maximum capacity.

20.6.4 Final Closure and Rehabilitation Concept

The conceptual plan for final rehabilitation and restoration can be summarized in the section below.

On a preliminary base, the revegetation process will typically involve soil conditioning and seeding. During the mine operation, WISCO, assisted with environment specialists, will assess the potential for revegetation with flora species that are indigenous or adapted to the area.

20.6.4.1 Tailings Storage Facility

The surfaces on the internal area of the tailings pond will be scarified to improve vegetation growth. The slopes of the dykes will be softened to s shallower angle, typically 3H:1V to reduce surface erosion and thus reducing suspended solids in run-offs.

The threshold of all tailings dikes' spillways will be lowered as much as possible to minimise the possibility of water accumulation in the various cells of the tailings pond. To protect the dikes against erosion and reduce the formation of water puddles inside the tailings area, drainage channels made with rip rap will be developed every 250 m around the TSF perimeter.

20.6.4.2 Waste Rock Piles

The waste rocks piles scarified and profile to control water drainage and dust migration. The slopes will be softened to a shallower angle, typically 3H: 1V to reduce surface erosion thus reducing suspended solids in run-offs.

20.6.4.3 Water Management Infrastructure

After five years of post-closure monitoring, or once it is confirmed that the effluent's quality complies with regulations, the water treatment plant will be dismantled. Water in the polishing pond will be pumped out and the dykes will be breached to allow free-flowing of surface runoff. If necessary, to prevent erosion of fine particles, the area of the pond will be covered crushed muck or coarse overburden (gravel of bigger) stocked in the pile.

Pumping stations and piping networks will be removed.

20.6.4.4 Access and Haul Roads

On-site haul roads and other mine roads will be scarified and seeded and culverts will be removed except for the sections leading to the TSF and the discharge point (total of about 24.5km). The later, as well as the access road to the site will be left intact and retroceded to a responsible entity (e.g., KRG, Innu nation or the Naskapi community.

20.6.4.5 Railways

The railway loop and side lines and the 91km-long future railway connecting to the main TSH railway near Schefferville will be dismantled. The steel rail and ties will be removed and disposed, or sold, but the foundation and ballast will be left in place. Areas with potential soil contamination, such as grease stations and switches, will be characterized and decontaminated following the applicable regulations. The railway foundation of the 91km-long section to Schefferville will be left intact and retroceded to a responsible entity (e.g., KRG, Innu nation or the Naskapi community).The railway loop and sidings on the site will be scarified and seeded.

20.6.4.6 Industrial Complex and Buildings

No building will be left in place. Whenever possible, buildings will be sold with the equipment they contain, completely or partially. During dismantling works, beneficiation/recycling of construction material will be maximized. Remaining waste will be disposed of in a landfill. For the cost estimate, no profits from the sales of equipment were considered and the dismantling assumes a demolition approach.

Reinforced concrete structures will be demolished and the rubbles will be covered with crushed muck or coarse overburden (gravel of bigger) stocked in the pile.

All equipment and machinery will be sent out of the site for sale or recycling.

Explosives magazine and related facilities will be dismantled.

The facilities for drinking water supply and domestic wastewater treatment may be transferred to a competent administrative authority or will be dismantled.

Infrastructure relating to electrical supply and distribution will be dismantled if of no use for other parties.

The area of the industrial site will be profiled and scarified.

20.6.4.7 Open Pits

After operation, the open pit will no longer be dewatered, and the pit will eventually fill up with groundwater and runoff water (rain and snow).

Security bunds will be constructed around the pit to prevent easy access.

20.6.4.8 Pellet Plan Complex and Buildings

No building will be left in place. Whenever possible, buildings will be sold with the equipment they contain, completely or partially. During dismantling works, beneficiation/recycling of construction material will be maximized. Remaining waste will be disposed of in a landfill. For the cost estimate, no profits from the sales of equipment were considered and the dismantling assumes a demolition approach.

All equipment and machinery will be sent out of the site for sale or recycling.

The area of the pellet plant site will be profiled, conditioned and seeded to help vegetation growth.

20.6.5 Post-Closure Monitoring

At the end of operations, the Project will submit a request to move into the post-operational monitoring phase. The monitoring program to be developed must be approved by the MDDELLC before implementation. The duration of the monitoring will depend on the time required to complete the restoration process, in order to then proceed with post-restoration monitoring. With respect to post-operational monitoring, a program will be developed for the post-restoration phase. This program must also be approved by the MDDELLC before implementation.

20.6.5.1 Physical Stability

The physical stability of the tailings accumulation areas and the waste rock piles will be assessed, and signs of erosion will be noted. These components will be monitored on an annual basis for three years following mine closure.

20.6.5.2 Environmental Monitoring

Monitoring of water quality (surface and groundwater) at specific locations such as tailings accumulation areas will continue for five years after the site is restored.

A program to monitor surface and groundwater quality at target locations such as, the tailings accumulation area will be carried out for at least five years after the site is restored.

Discontinuation of the post-restoration monitoring program must be authorized by the MDDELLC.

20.6.5.3 Agricultural Monitoring

The purpose of the agricultural monitoring program is to assess the effectiveness of revegetation done as part of the mining site rehabilitation efforts.

Documenting the success of revegetation of the accumulation areas, agricultural monitoring will be undertaken following the establishment of a plant cover in the areas subject to the progressive restoration program. Monitoring will be conducted annually for three years following revegetation.

Once the mine site is closed, the restoration plan will be implemented and the vast majority of the site will be vegetated. Revegetation success will be monitored for three years. If required, reseeding will be carried out at spots where revegetation is not deemed satisfying.

20.6.5.4 Water Treatment Plant Operations

The water treatment plant will be in operation for five years after the mine closure. After five years of operation during post-closure monitoring, it is assumed that the quality of the effluent will comply with regulations. It is assumed that the water treatment will mostly be required during the first or first two years after closure: while vegetation sprouts. It is assumed that the runoff of the following surfaces will have to be treated:

- Industrial site (51 ha);
- Waste rock dump: all the water will be redirected to the open pit;
- Tailings – internal areas (2,579 ha): the water will flow toward the water pond where it will be treated;
- Water pond: the water that could potentially have to be treated will come from the tailings pond and it is assumed that it will be treated.

It is assumed that these surfaces will generate about 266 cubic meters per day (266 m^3/d) to be treated, typically for high total solid content with agents that improve precipitation.

20.6.6 Financial Guarantee

Under the Quebec Mining Act (CQLR, c M-13.1 Section 96.5 to 96.16), the operator must provide a financial guarantee equal to 100% of all anticipated closure costs and post-closure environmental monitoring, including the engineering effort required to implement the closure plan.

The provincial authorities require that closure cost estimate does not account for any residual value of equipment, building, structure, land etc.

The amount of the financial guarantee is set aside progressively in the first three years of production following the schedule below:

- First payment of 50% of the total guarantee in the 90 days after approbation of the closure plan;
- Two payments of 25% each, each one at anniversary date of the approbation of the plan.

This amount can be recovered by the operator after the minimum post-closure monitoring period and once the Ministère has judged the mine closure satisfactory.

21 Capital and Operating Costs

21.1 Capital Cost Estimate

This section covers the capital cost estimate for implementation of the mining, concentrating, handling and pelletizing as well as related infrastructures required for the development of the Full Moon Project for each option as described in Section 1.13. The following paragraphs outline the methodology used by CIMA+ personnel for the estimation of the capital cost of the project. The resulting estimate is based on the application of standard methods required to achieve an estimate with an accuracy range of -25% and +35%.

21.1.1 Scope of Estimate

The capital cost estimate covers all or some of the following areas depending on the option:

- Mining: initial cost for rolling stock, field services, site infrastructures as well as electrical distribution;
- Crushing and stockpiling: gyratory crushers, access ramp, retaining wall, screens, cone crushers, stockpile feed conveyors, stockpile reclaim conveyors and transport conveyors;
- Magnetite plant: feed conveying from crushed ore stockpiles, grinding, regrinding, magnetic separation, classification and low silica concentrate refinement;
- Hematite plant: regrinding, magnetic separation, rougher flotation, and low silica concentrate refinement;
- Concentrate thickening and drying: dewatering thickener, filter and dryer;
- Load out facilities, concentrate storage and rail car loading facilities;
- Tailings: tailings thickeners, pumps and pipelines;
- Tailings management facilities: costs were provided by AMEC and included in the estimate as obtained;
- Infrastructures and services: access & plant roads, electrical substation and distribution, process & gland seal water, reclaim water, potable water, domestic waste water treatment plant, fire water

distribution, HVAC, compressed air, administration building, workshop, warehouse, accommodation camp, security gate;

- Rail works: rail loop at the process site, railway from mine site to Schefferville and railway sidings;
- The 450 km long, new 315 kV power line will be a turnkey project to start a the LG4/Tilley substation; and
- The construction of the pellet plant will also be a turnkey project and the estimate for that is included.

21.2 Summary of the Capital Cost Estimate

The capital cost of the project is the cost for the initial development of the project. When additional capital expenditures are planned for future capital equipment additions and replacements they will be charged as sustaining capital expenditures. Table 21.1shows the summary of the capital cost estimate.

Four options were analyzed, namely:

- Option 1: High Silica Concentrate without pelletizing plant;
- Option 2: Low Silica Concentrate without pelletizing plant;
- Option 3: High Silica Concentrate with a pelletizing plant; and
- Option 4: Low Silica Concentrate with a pelletizing plant.

Table 21.1 – Summary of Capital Cost Estimate

WBS No	Description	Option 1 ($'000) (Preferred)	Option 2 ($'000)	Option 3 ($'000)	Option 4 ($'000)
	Direct Cost				
00000	Project General	655,681	671,999	655,681	671,999
11000	Mine-Equipment	187,527	187,527	187,527	187,527
14000	Full Moon Mine	54,813	54,813	54,813	54,813
34000	Concentrator	2,513,852	2,626,031	2,513,852	2,626,031
44000	Tailings	450,379	450,379	450,379	450,379
54000	Railroad & Rail Yard	441,101	441,101	441,101	441,101
66000	Sept-Îles Pellet Plant	0	0	1,678,807	1,678,807
74000	Infrastructures	859,802	859,802	859,802	859,802
	Total Direct Cost	**5,163,154**	**5,291,651**	**6,841,962**	**6,970,458**
	Indirect Costs				
91000	EPCM Management	286,644	293,778	286,644	293,778
92000	Construction Services	137,548	140,971	137,548	140,971
93000	Construction Indirect	151,013	154,771	151,013	154,771
C0000	Contingency	603,051	618,059	603,051	618,059
E0000	Escalation	398,973	408,902	398,973	408,902
R0000	Risk	418,921	429,347	418,921	429,347
	Total Indirect Cost	**1,996,149**	**2,045,827**	**1,996,149**	**2,045,827**
	Other Costs				
	Mine – Pre-Production	48,013	48,013	48,013	48,013
	Total Project Cost	**7,207,316**	**7,385,492**	**8,886,124**	**9,064,299**

21.2.1 General Capital Cost

This section covers all direct costs to the project that cannot be expressly assigned to any dedicated WBS. The general capital cost is shown in Table 21.2.

Table 21.2 –General Capital Cost Estimate

WBS No	Description	Option 1 & 3 ($'000)	Option 2 & 4 ($'000)
00000	Project General		
	Allowance for casual overtime	26,680	27,344
	Heavy lifts - Process	48,786	50,000
	Scaffolding - Process	14,636	15,000
	Heating and Hoarding - Process	24,393	25,000
	Allowance for ocean and land freight for equipment	62,205	63,753
	Allowance for land freight for materials	44,343	45,447
	Freight for Rolling Stock; based on budget	5,968	6,117
	Duties + brokerage fees (included with freight)	0	0
	Assistance by contractors for commissioning	26,157	26,808
	Labour cost during Transportation - workers	46,347	47,500
	Transportation costs	70,739	72,500
	Construction Camp	86,936	89,100
	Catering Services	126,579	129,729
	Electrical power for construction camp	19,279	19,759
	Allowance for vendor reps - equipment	8,886	9,107
	Allowance for vendor reps - mobile equipment	5,865	6,011
	Allowance for vendor reps - commissioning	8,887	9,108
	Allowance for spare parts - construction	8,887	9,108
	Allowance for spare parts - commissioning	8,887	9,108
	Allowance for first fill	9,757	10,000
	Allowance for special tools	1,464	1,500
	Total	**655,681**	**671,999**

21.2.2 Mine Capital Cost

The capital cost for the Mining area includes the initial development of the open pit mine, including the haul roads from the gyratory crushers to the mine workshop. It includes the planned pre-stripping and development of the areas for the overburden stockpile and the waste dump. It includes the purchase of all initially purchased mining equipment required for the first two (2) years of operations (year of pre-production and the first year of production). The summary of the capital cost for the mine is shown in Table 21.3.

Table 21.3 – Summary of Mine Capital Cost Estimate

WBS No	Description	All Option ($'000)
11100	Mining Excavation & Mobile Equipment	186,027
11200	Ancillary Equipment	1,500
14200	Mine Dewatering	3,251
14400	Mine Haulage Roads	964
14700	Field Services	1,076
14800	Field Infrastructures	46,116
14900	Mine Substation & Elect. Dist.	3,406
	Total	**242,340**

21.2.3 Concentrator

The capital cost for the concentrator includes the costs for the buildings and foundations as well as the costs of all mechanical equipment for the crushers, the conveyors, the magnetite and hematite plants. It also includes the costs of the drying buildings and thickeners as well as all other related equipment. The cost for the fixed and mobile mechanical equipment is included as well. It also includes the costs for services, power and its distribution as well as that for communications. Table 21.4 shows the summary of the total estimated costs for the concentrator.

Table 21.4 – Summary of Concentrator Capital Cost Estimate

WBS No	Description	Option 1 & 3 ($'000)	Option 2 & 4 ($'000)
34000	Concentrator	25,295	25,295
34100	Mobile Equipment	5,053	5,053
34200	Crushing	361,936	361,936
34300	Magnetite Plant	844,318	926,458
34400	Hematite Plant	682,981	713,020
34500	Drying	394,428	394,428
34600	Tailing Thickener	139,032	139,032
34700	Load-out	48,984	48,984
34900	Process Water	11,825	11,825
	Total	**2,513,852**	**2,626,031**

21.2.4 Tailings Facilities Management

The capital cost for the tailings facilities management include the costs for the mobile equipment and the pump stations. It also includes the cost for the pipelines as well as that for the tailings dam construction. Table 21.5 shows the summary of the capital cost of the tailings facilities management systems.

Table 21.5 – Summary of Tailings Capital Cost Estimate

WBS No	Description	All Option ($'000)
44000	Tailings	27,016
44200	Pump Stations	73,759
44300	Tailings & Reclaim Water Pipelines	277,304
44400	Tailings Dam	72,300
	Total	**450,379**

21.2.5 Railroad and Rail Yard

The capital cost for the railroad and rail yard includes the costs for the railroad, railroad loop and the rail mobile equipment. A summary of the costs for the railroad and rail yard is shown in Table 21.6.

Table 21.6 – Summary of Railroad and Rail Yard Capital Cost Estimate

WBS No	Description	All Option ($'000)
54100	Mobile Equipment	206,722
54400	Stockpile Yard	3,382
54800	Yard Infrastructures	15,051
54200	Railroad	215,946
	Total	**441,101**

21.2.6 Pellet Plant

The capital cost for the pellet plant will be a turnkey project and it is estimated at M$1,679. The pellet plant is only valid for Option 3 and 4.

21.2.7 Infrastructures

The cost of the infrastructures includes the costs for the various site roads as well as the cost of the buildings. The main roads are the access road from Schefferville to the mine site, the roads between the accommodation camp, concentrator, crushers and the mine site. It also includes the road to the explosives storage facility. The accommodation camp and related facilities are included in this area, as well as the administration building and warehouse complex. The services are also included. A summary of the costs is shown in Table 21.7.

Table 21.7 – Summary of Infrastructures Capital Cost Estimate

WBS No	Description	All Option ($'000)
74000	Infrastructures	12,396
74200	Non-process buildings	578
74300	Off Site Infrastructures	68,700
74400	Port at Sept-Îles (Use of Facilities Fee)	52,000
74800	Field Infrastructures	37,826
74900	Main Substation & Elec. Dist.	688,302
	Total	**859,802**

21.2.8 Project Indirect Costs

The indirect costs for the projects consist of EPCM management, external engineering consultants, procurement, construction services, construction indirect costs, contingencies, escalation and risk. A summary of the project indirect costs is shown in Table 21.8.

Table 21.8 – Summary of Capital Cost Estimate of the Indirect Costs

WBS No	Description	Option 1 & 3 ($'000)	Option 2 & 4 ($'000)
91000	EPCM Management	80,540	82,545
91200	External Engineering Consultants	161,080	165,089
91300	Procurement Office	45,024	46,144
92000	Construction Services	137,548	140,971
93000	Construction Indirect	151,013	154,771
C0000	Contingency	603,051	618,059
E0000	Escalation	398,973	408,902
R0000	Risk	418,921	429,347
	Total	**1,996,149**	**2,045,827**

21.3 Capital Cost Basis of Estimate

21.3.1 Currency Base Date and Exchange Rate

The capital cost estimate is expressed in 1st quarter 2015 Canadian dollars. Prices obtained in other currencies were converted using currency exchange rates.

It is assumed that the construction phase will extend from 2nd quarter 2018 to 4th quarter 2021, i.e. mechanical completion, for a total of 42 months.

21.3.2 Freight, Duties and Taxes

Freight costs were estimated on the basis of historical data; a compounded ratio of ocean and land freight were applied to equipment costs and estimated at 7% of the equipment costs (5% for land freight and 9% for ocean freight). For bulk material, only land freight is required and is estimated at 5%. Ocean freight is inclusive of duties and taxes. Freight costs for rolling stock are based on a budgetary quotation.

21.3.3 Design allowances and contingencies

For the purposes of this PEA, no design allowances were added to the estimated costs. Contingency evaluated at 12.5% of all costs (excluding main power line, pellet plant, rolling stock, tailings dam and port fees) and 7.5% on the rolling stock was added to the estimate to reflect the engineering progress (evaluated at 2%).

21.3.4 Escalation and Risk

Escalation is included in the estimate and evaluated at 1.5% per year, for period starting 1st quarter 2015 and ending 4th quarter 2019, using mass gravity center. It is assumed that construction phase will extend from 2nd quarter 2018 to 4th quarter 2021, i.e. 42 months. Total percent escalation is 10% applied to one half of all costs including the contingency.

An allowance for Risk is included and evaluated at 5% of all costs.

21.3.5 Civil and building works

Civil and building quantities were generally provided by engineering. In order to ascertain the full scope coverage, some minor additional elements of scope were added. Unit rates for supply and installation were estimated on the basis of recent in-house data and compared against benchmarks obtained from projects similar in nature and in site conditions.

21.3.6 Equipment

Equipment for mechanical, electrical and instrumentation/control were provided by engineering; budgetary quotations were obtained for major mechanical equipment. The balance of the equipment was estimated based on recent in-house data. Installation man hours were estimated based on historical data or from well reputed estimating handbooks.

21.3.7 Piping and Pipelines

Piping was estimated at 30% of mechanical equipment cost. Pipeline quantities, complete with materials of construction, were provided by engineering; budgetary quotations were obtained for all piping materials. Installation man hours were estimated based on historical data from in-house data for similar projects or from well reputed estimating handbooks.

21.3.8 Electrical and Instrumentation Equipment and Material

Electrical and instrumentation quantities, complete with supply and installation rates, were provided by engineering. All rates were based on recent in-house data from a project similar in nature and in site conditions.

21.3.9 Labour Costs

Labour costs were developed on the basis of the labour decree in effect in the Province of Québec. Labour crew mixes were developed for all disciplines and contractors' indirect costs as well as construction equipment costs were added in order to have all-inclusive labour crew mix wage rates.

For the purposes of this PEA, it was assumed that workers at the process site would have a 58 hour workweek, i.e. 5 days at 10 hours per day, 1 day at 8 hours with a 26/2/7 rotation schedule. Travelling cost are included and estimated at $600 per rotation. Also included are the costs of labour during transportation, estimated at 16 hours per round trip (2 x 8 hours) at the workers base wage rate.

At the port area, workers would have a 50 hour workweek, i.e. 5 days at 10 hours per day, with no rotation, as it is assumed that 50% of the workforce would be local and 50% would be within weekly travelling distance. Living out allowances as well as transportation costs are included and based on the labour decree. Since workers are entitled to transportation costs, there is no need to add labour costs to cover for travelling time.

21.3.10 Indirect costs

Indirect costs were mostly estimated on the basis of historical ratios. EPCM services were factored from the direct costs and divided between project management & project controls (2.4% of all direct cost, excluding main power line, pellet plant, rolling stock, tailings dam and port fees), engineering services (4.8% of all direct cost, excluding main power line, pellet plant, rolling stock, tailings dam and port fees), procurement services (1.2% of all direct cost, including rolling stock and excluding main power line, pellet plant, tailings dam and port fees), construction management services (3.6% of all direct cost, excluding main power line, pellet plant, rolling stock, tailings dam and port fees) and construction management services (0.6% of cost for main power line, pellet plant, rolling stock, tailings dam). Construction field indirect costs were factored from the total direct costs, excluding main power line, pellet plant, rolling stock, tailings dam and port fees, at 4.5%.

21.4 Mine Closure and Remediation Cost Estimate

Mine closure costs for the Project are estimated at approximately $178,211,000 spread over three years at the end of Life of Mine. The closure costs consist of $134,550,000 of direct cost and $43,661,000 in indirect costs. Table 21.9 below presents a summary of the mine closure capital costs estimate.

The closure costs include the dismantlement of the railway from the Project site to TSH railway near Schefferville line and the dismantlement and restoration of pellet plant site.

Table 21.9 – Project Preliminary Mine Closure Costs

Closure Cost Item	Cost ($'000)
Direct Costs	
Dismantlement of buildings in the industrial complex	28,630
Dismantlement of the piping network, pumping stations and sanitary infrastructures	570
Dismantlement of electric infrastructures	410
Dismantlement of pellet plant and restoration of footprint (≈17.3ha)	15,261
Dismantlement of site railway infrastructures and restoration of footprint (≈ 0.6ha)	278
Restoration of the industrial complex footprint (≈ 50.8ha)	1,860
Restoration of haulage road footprint (5 years after final closure; 24.5km is left in place for public access)	81
Restoration of waste rock dumps (≈ 49.9ha)	2,134
Restoration of overburden pile footprint (≈ 90.0ha)	2,160
Restoration of tailings ponds internal area (≈ 2,579ha)	63,096
Restoration of tailings pond dykes (≈ 34.9ha)	1,048
Restoration of the polishing ponds (5 years after final closure; ≈ 248ha)	5,948
Full Moon Pit	487
Dismantlement of railway infrastructures to Schefferville (91.5km; rail foundation is left in place for public access)	12,587
Sub-Total – Direct Costs	**134,550**
Indirect Costs	
Engineering and Permitting	5,000
Post-Closure Monitoring – (year 1 to 5)	4,340
Post-Closure Monitoring – (year 6 to 10)	0
Contingencies on Direct Costs	33,637
Contingencies on Indirect Costs	684
Sub-Total – Indirect Costs	**43,661**
TOTAL	**178,211**

21.5 Sustaining Capital Cost Estimate

The Sustaining Capital costs are the capital expenditures during the life of the mine that are required to maintain or upgrade the existing asset and to continue the operation at the same level of production.

The sustaining capital cost estimates for the life of mine are summarized in the Tables 21.10 to 21.13.

Table 21.10 – Summary of Sustaining Capital Cost Estimate (Year 2 to 7)

Area	Year 2 ($'000)	Year 3 ($'000)	Year 4 ($'000)	Year 5 ($'000)	Year 6 ($'000)	Year 7 ($'000)
Mining Equipment	24,995	1,413		515	11,671	
Tailings Pond				93,500		
TOTAL	**24,995**	**1,413**		**94,015**	**11,671**	

Table 21.11 – Summary of Sustaining Capital Cost Estimate (Year 8 to 13)

Area	Year 8 ($'000)	Year 9 ($'000)	Year 10 ($'000)	Year 11 ($'000)	Year 12 ($'000)	Year 13 ($'000)
Mining Equipment	386		113,264	83,692		
Tailings Pond			126,300			
TOTAL	**386**		**239,564**	**83,692**		

Table 21.12 – Summary of Sustaining Capital Cost Estimate (Year 14 to 19)

Area	Year 14 ($'000)	Year 15 ($'000)	Year 16 ($'000)	Year 17 ($'000)	Year 18 ($'000)	Year 19 ($'000)
Mining Equipment			24,995			
Tailings Pond						
TOTAL			**24,995**			

Table 21.13 – Summary of Sustaining Capital Cost Estimate (Year 20 to 25)

Area	Year 20 ($'000)	Year 21 ($'000)	Year 22 ($'000)	Year 23 ($'000)	Year 24 ($'000)	Year 25 ($'000)
Mining Equipment		1,542				
Tailings Pond						175,700
TOTAL		**1,542**				**175,700**

21.6 Operating Cost Estimate

21.6.1 Scope and Methodology

The operating costs for the project were estimated annually, based on the mine plan developed by Met-Chem. A summary of these operating costs are shown in the followings tables. The operating costs of the average life of mine of operations have been detailed for each option and are considered representative of the typical average cost for the life of the mine. The operation has been divided into six (6) areas namely:

- Mining;
- Concentrating;
- Tailings;
- General and Administration;
- Rail Transportation and Port; and
- Pelletizing.

The material produced in each option is described in the Table 21.14.

Table 21.14 – Material Produced by Option

Area	Option 1 (Preferred)	Option 2	Option 3	Option 4
Final Produce (tonne)	**HSC.**	**LSC**	**HSC – Pellets**	**LSC - Pellets**
High Silica Concentrate	20,000,000		4,200,000	
Low Silica Concentrate		18,300,000		2,000,000
HSF Pellet			17,000,000	
DR Pellet				17,000,000

The summary of the annual operating costs and the cost per tonne of concentrate for an average year of operations (Year 5), are shown in Table 21.15 to Table 21.18. A part of the concentrate produced, will be pelletized and the remaining concentrate will be sold directly on the market. All operating costs are charged to the concentrate except for the pelletizing costs that is only charged as an additional cost applied to the part of the concentrate that is pelletized.

Table 21.15 – Summary of an Average Year of Operations per Area – Option 1 (Preferred)

Area	Annual Cost ($'000)	Unit Cost ($/t conc.)
Mining	111,975	5.60
Concentrating	259,544	12.98
Tailings	14,608	0.73
General and Administration	53,236	2.66
Rail Transportation & Port	557,625	27.88
Pellet Plant	0	0
TOTAL	**996,988**	**49.85**

Table 21.16 – Summary of an Average Year of Operations per Area – Option 2

Area	Annual Cost ($'000)	Unit Cost ($/t conc.)
Mining	111,975	6.12
Concentrating	329,084	17.98
Tailings	14,608	0.80
General and Administration	53,233	2.91
Rail Transportation & Portand Physiography	510,363	27.89
Pellet Plant	0	0
TOTAL	**1,019,263**	**55.70**

Table 21.17 – Summary of an Average Year of Operations per Area – Option 3

Area	Annual Cost ($'000)	Unit Cost ($/t conc.)	Unit Cost ($/t Pellets)
Mining	111,975	5.60	5.20
Concentrating	259,544	12.98	12.06
Tailings	14,608	0.73	0.68
General and Administration	53,236	2.66	2.48
Rail Transportation & Port	557,625	27.88	25.91
Pellet Plant	190,198		11.19
TOTAL	**1,187,186**	**49.85**	**57.52**

Table 21.18 – Summary of an Average Year of Operations per Area – Option 4

Area	Annual Cost ($'000)	Unit Cost ($/t conc.)	Unit Cost ($/t Pellets)
Mining	111,975	6.12	5.87
Concentrating	329,084	17.98	17.24
Tailings	14,608	0.80	0.77
General and Administration	53,233	2.91	2.79
Rail Transportation & Port	510,363	27.89	26.74
Pellet Plant	182,427	0	10.73
TOTAL	**1,201,690**	**55.70**	**64.14**

21.6.2 Mine Operating Costs

The mine operating cost estimate was prepared by Met-Chem. The mine operating cost was estimated annually and assuming an owner's fleet. The cost is based on operating the mining equipment, the manpower associated with operating the equipment, the cost for explosives as well as dewatering, road maintenance and other activities. A summary of the operating cost for the mine operation for an average year of operation are shown in Table 21.19 to Table 21.22.

Table 21.19 – Summary of an Average Year of Operation for Mine Sector – Option 1

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)
Major Equipment	49,504	2.48
Support Equipment	10,898	0.54
Services Equipment	1,627	0.08
Manpower	30,565	1.53
Explosives	19,340	0.97
Other	41	0.00
TOTAL	**111,975**	**5.60**

Table 21.20 – Summary of an Average Year of Operation for Mine Sector – Option 2

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)
Major Equipment	49,504	2.70
Support Equipment	10,898	0.60
Services Equipment	1,627	0.09
Manpower	30,565	1.67
Explosives	19,340	1.06
Other	41	0.00
TOTAL	**111,975**	**6.12**

Table 21.21 – Summary of an Average Year of Operation for Mine Sector – Option 3

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)	Unit Cost ($/t Pellets)
Major Equipment	49,504	2.48	2.30
Support Equipment	10,898	0.54	0.51
Services Equipment	1,627	0.08	0.07
Manpower	30,565	1.53	1.42
Explosives	19,340	0.97	0.90
Other	41	0.00	0.00
TOTAL	**111,975**	**5.60**	**5.20**

Table 21.22 – Summary of an Average Year of Operation for Mine Sector – Option 4

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)	Unit Cost ($/t Pellets)
Major Equipment	49,504	2.70	2.60
Support Equipment	10,898	0.60	0.57
Services Equipment	1,627	0.09	0.09
Manpower	30,565	1.67	1.60
Explosives	19,340	1.06	1.01
Other	41	0.00	0.00
TOTAL	**111,975**	**6.12**	**5.87**

In order to determine the operating costs, the following assumptions were used;

- Diesel Fuel Price: $0.90/litre;
- Power Cost: $0.0317/kWh + $12.63/kW/month (Hydro-Québec L-rate);
- Hourly maintenance and operating cost of equipment;
- Cost of explosives was estimated with the tonnage mined.

21.6.3 Concentrating Operating Costs

The concentrator operating cost was estimated with the annual tonnage. The various processing steps detailed in Section 17, are crushing, magnetite and hematite separation followed by drying and conveying to the concentrate load-out. The summary of the operating costs for concentrating operation of an average year of operation are shown in Table 21.23 to Table 21.26.

Table 21.23 – Summary of an Average Year of Operation for Concentrating Sector – Option 1

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)
Power	123,380	6.17
Mobile Equipment	2,861	0.14
Reagents	78,267	3.92
Consumables	27,406	1.37
Manpower	26,424	1.32
Other	1,206	0.06
TOTAL	**259,544**	**12.98**

Table 21.24 – Summary of an Average Year of Operation for Concentrating Sector – Option 2

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)
Power	130,744	7.14
Mobile Equipment	2,861	0.16
Reagents	134,172	7.33
Consumables	27,406	1.50
Manpower	32,695	1.79
Other	1,206	0.06
TOTAL	**329,084**	**17.98**

Table 21.25 – Summary of an Average Year of Operation for Concentrating Sector – Option 3

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)	Unit Cost ($/t Pellets)
Power	123,380	6.17	5.73
Mobile Equipment	2,861	0.14	0.13
Reagents	78,267	3.92	3.64
Consumables	27,406	1.37	1.27
Manpower	26,424	1.32	1.23
Other	1,206	0.06	0.06
TOTAL	**259,544**	**12.98**	**12.06**

Table 21.26 – Summary of an Average Year of Operation for Concentrating Sector – Option 4

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)	Unit Cost ($/t Pellets)
Power	130,744	7.14	6.85
Mobile Equipment	2,861	0.16	0.15
Reagents	134,172	7.33	7.03
Consumables	27,406	1.50	1.44
Manpower	32,695	1.79	1.71
Other	1,206	0.06	0.06
TOTAL	**329,084**	**17.98**	**17.24**

In order to determine the operating costs, the following assumptions were used;

- Power Cost: $0.0317/kWh + $12.63/kW/month (Hydro-Québec L-rate);
- Reagent unit cost:

<u>For High Silica Concentrate (Option 1 & 3)</u>

- ➢ Hematite plant rougher flotation: $3.50/t of concentrate;
- ➢ Tailings flotation: $0.41/t of concentrate;

<u>For Low Silica Concentrate (Option 2 & 4)</u>

- ➢ Magnetite plant LSC flotation: $1.83/t of concentrate;
- ➢ Hematite plant rougher flotation: $3.82/t of concentrate;
- ➢ Hematite plant LSC flotation: $1.23/t of concentrate;
- ➢ Tailings flotation: $0.45/t of concentrate;

- Consumables annual cost:
  - ➢ Gyratory – liners: $1,600,000/year;
  - ➢ AG Mill - liners: $12,000,000/year;
  - ➢ Maintenance Supplies: $9,624,000/year;
  - ➢ Operating Supplies: $1,444,000/year;
  - ➢ Laboratory Supplies: $2,600,000/year.

21.6.4 Tailings Operating Costs

The tailings impoundment area costs include the manpower and the equipment required to do hydraulic deposition. The cost also included the water treatment at the exit of the tailings pond.

Table 21.27 to Table 21.30 show the summary of the operating costs for the tailings impoundment area operations of an average year of operation for the four options.

Table 21.27 – Summary of an Average Year of Operation for Tailings Sector – Option 1

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)
Power	5,379	0.27
Mobile Equipment	1,841	0.09
Maintenance	4,020	0.20
Manpower	3,368	0.17
TOTAL	**14,608**	**0.73**

Table 21.28 – Summary of an Average Year of Operation for Tailings Sector – Option 2

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)
Power	5,379	0.29
Mobile Equipment	1,841	0.10
Maintenance	4,020	0.22
Manpower	3,368	0.19
TOTAL	**14,608**	**0.80**

Table 21.29 – Summary of an Average Year of Operation for Tailings Sector – Option 3

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)	Unit Cost ($/t Pellets)
Power	5,379	0.27	0.25
Mobile Equipment	1,841	0.09	0.08
Maintenance	4,020	0.20	0.19
Manpower	3,368	0.17	0.16
TOTAL	**14,608**	**0.73**	**0.68**

Table 21.30 – Summary of an Average Year of Operation for Tailings Sector – Option 4

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)	Unit Cost ($/t Pellets)
Power	5,379	0.29	0.28
Mobile Equipment	1,841	0.10	0.10
Maintenance	4,020	0.22	0.21
Manpower	3,368	0.19	0.18
TOTAL	**14,608**	**0.80**	**0.77**

In order to determine the operating costs, the following assumptions were used;

- Power Cost: $0.0317/kWh + $12.63/kW/month (Hydro-Québec L-rate);
- Mobile Equipment required:
  - Two (2) dozers 433 kW;
  - Three (3) dozers 231 kW;
  - One (1) Hydraulic excavator 45 t

21.6.5 General and Administration Operating Costs

The general and administration costs include the operation of all the services, manpower and infrastructures required to support the operations. The operations included are:

- Site mobile equipment;
- Accommodation camp,
- Site administration including accounting, human resources, health and safety, supply chain, site maintenance, IT and security;
- Fly-in / Fly-out cost; and
- Catering cost.

Table 21.31 to Table 21.34 show the summary of the operating costs for the general and administration operation of average year of operation for the four options.

Table 21.31 – Summary of an Average Year of Operation for General and Administration Sector – Option 1

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)
Mobile Equipment	950	0.05
Catering	11,583	0.58
Fly-in/Fly-Out	9,433	0.47
Power	12,337	0.62
Manpower	15,293	0.76
Other	3,640	0.18
TOTAL	**53,236**	**2.66**

Table 21.32 – Summary of an Average Year of Operation for General and Administration Sector – Option 2

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)
Mobile Equipment	950	0.05
Catering	11,583	0.63
Fly-in/Fly-Out	9,433	0.52
Power	12,334	0.67
Manpower	15,293	0.84
Other	3,640	0.20
TOTAL	**53,233**	**2.91**

Table 21.33 – Summary of an Average Year of Operation for General and Administration Sector – Option 3

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)	Unit Cost ($/t Pellets)
Mobile Equipment	950	0.05	0.05
Catering	11,583	0.58	0.54
Fly-in/Fly-Out	9,433	0.47	0.44
Power	12,337	0.62	0.57
Manpower	15,293	0.76	0.71
Other	3,640	0.18	0.17
TOTAL	**53,236**	**2.66**	**2.48**

Table 21.34 – Summary of an Average Year of Operation for General and Administration Sector – Option 4

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)	Unit Cost ($/t Pellets)
Mobile Equipment	950	0.05	0.05
Catering	11,583	0.63	0.61
Fly-in/Fly-Out	9,433	0.52	0.49
Power	12,334	0.67	0.65
Manpower	15,293	0.84	0.80
Other	3,640	0.20	0.19
TOTAL	**53,233**	**2.91**	**2.79**

In order to determine the operating costs, the following assumptions were used;

- Supply and operation of the camp: $90/person/day;
- Charter round trip cost: $60,000/flight;
- Site communications: $2,000,000/year;
- Office supplies: $500,000/year; and
- Transportation: $1,000,000/year.

21.6.6 Rail & Port Terminal Operating Costs

The rail operating costs include a transportation contract with a carrier who will haul the ore gondolas from the loading operation and the hauling to a multi-users terminal. The material handling at the port was estimated as a unit cost charged by the terminal. The handling cost include the unloading of the rail gondolas, the stacking of the concentrate, the reclaiming of the concentrate to the pellet plant, the stacking of the pellets and the ship loading of the concentrate and the pellets.

Table 21.35 to Table 21.38 show the summary of the operating cost for the four options of the rail operations for an average year of operation.

Table 21.35 – Summary of an Average Year of Operation for Rail and Port Sector – Option 1

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)
Haulage Cost	440,300	22.01
Gondola Maintenance	15,715	0.79
Port Terminal	100,000	5.00
Manpower	1,610	0.08
TOTAL	**557,625**	**27.88**

Table 21.36 – Summary of an Average Year of Operation for Rail and Port Sector – Option 2

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)
Haulage Cost	402,874	22.01
Gondola Maintenance	14,379	0.79
Port Terminal	91,500	5.00
Manpower	1,610	0.09
TOTAL	**510,363**	**27.89**

Table 21.37 – Summary of an Average Year of Operation for Rail and Port Sector – Option 3

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)	Unit Cost ($/t Pellets)
Haulage Cost	440,300	22.01	20.46
Gondola Maintenance	15,715	0.79	0.73
Port Terminal	100,000	5.00	4.65
Manpower	1,610	0.08	0.07
TOTAL	**557,625**	**27.88**	**25.91**

Table 21.38 – Summary of an Average Year of Operation for Rail and Port Sector – Option 4

Description	Annual Cost ($'000)	Unit Cost ($/t conc.)	Unit Cost ($/t Pellets)
Haulage Cost	402,874	22.01	21.11
Gondola Maintenance	14,379	0.79	0.75
Port Terminal	91,500	5.00	4.79
Manpower	1,610	0.09	0.09
TOTAL	**510,363**	**27.89**	**26.74**

In order to determine the operating costs, the following assumptions were used;

- Rail haulage sub-contract unit cost: $0.325/tonne/km;
- Hauling distance: 662km; and
- Port terminal sub-contract: $5.00/t

21.6.7 Pellet Plant Operating Costs

The pellet plant operating cost was estimated for the annual tonnage. The various processing steps detailed in Section 17, are additive handling, mixing, balling and induration. The summary of the operating costs for the pelletizing operation of an average year of operation are shown in Table 21.39 and Table 40 for the two pelletizing options (Option 3 and Option 4).

Table 21.39 – Summary of an Average Year of Operation for Pellet Plant Sector – Option 3

Description	Annual Cost ($'000)	Unit Cost ($/t Pellets)
Power	30,377	1.79
Heavy Fuel	60,083	3.53
Reagent	18,651	1.10
Consumables	55,250	3.25
Manpower	25,837	1.52
TOTAL	**190,198**	**11.19**

Table 21.40 – Summary of an Average Year of Operation for Pellet Plant Sector – Option 4

Description	Annual Cost ($'000)	Unit Cost ($/t Pellets)
Power	30,377	1.79
Heavy Fuel	60,083	3.53
Reagent	10,880	0.64
Consumables	55,250	3.25
Manpower	25,837	1.52
TOTAL	**182,427**	**10.73**

- Power Cost: $0.0317/kWh + $12.63/kW/month (Hydro-Québec L-rate);
- Heavy Fuel : 7.14 liters/t of pellets
- Heavy Fuel unit cost: $0.50/liter;
- Reagent unit cost:
  - Activator: $0.04/t of pellets;
  - Bentonite: $0.60/t of pellets;
- Consumables annual cost:
  - Filter Bags: $0.03/t of pellets;
  - Filter Sectors: $0.01/t of pellets;
  - Roller: $0.04/t of pellets;
  - Refractories: $0.03/t of pellets;
  - Spares: $2.00/t of pellets;
  - Grate Bars: $0.01/t of pellets;
  - Other Consumables: $1.13/t of pellets.

21.6.8 Manpower

The site will be operating continuously, 24 hour per day with 2 - 12 hour shifts, with a turnaround every 2 weeks.

21.6.8.1 Mine Operations Manpower

The mine operations manpower has been estimated for three sections namely Operations, Maintenance and Technical Services. The required manpower for the typical year (Year 5) has been shown in Table 21.41.

Table 21.41 – Estimated Mine Manpower Requirements

Position	Year 5
Operation	
Mine Manager	1
Mine Superintendent	1
Pit Foreman	12
Equipment Operator	164
Labourer	8
Dispatcher	4
Trainer	4
Blaster	2
Blaster Helper	2
Maintenance	
Maintenance Superintendent	1
Maintenance Foreman	8
Maintenance Planner	4
Mechanic/Electrician/Welder	32
Attendant	8
Technical Services	
Mine Technical Superintendent	1
Mining Engineer	4
Geologist	4
Grade Control Technician	4
Surveyor	4
TOTAL	**268**

21.6.8.2 Manpower for Crushing and Concentrating Operation

The crusher and concentrator operations manpower has been estimated for three sections namely Administration, Operations/Maintenance and Metallurgy and Laboratory. The required manpower for the typical year (Year 5) has been shown in Table 21.42.

Table 21.42 – Estimated Crushing and Concentrator Manpower Requirements

Position	Year 5
Administration	
Mill Superintendent	1
Mill General Foreman	1
Operation and Maintenance	
Mechanical Engineer	2
Electrical Engineer	2
Maintenance Planner	4
Control Room Operator	12
Operator	132
Mechanic\Welder	40
Electrician\Instrumentation	28
Programming Technician	4
Metallurgy and Laboratory	
Chief Metallurgist	1
Metallurgist	4
Metallurgical Technician	4
Chief Laboratory	2
Laboratory Technician	4
Laboratory Attendant	12
TOTAL	**253**

21.6.8.3 Manpower for Tailings Operation

The tailings operations manpower has been estimated for Operations/Maintenance of the tailing pond and related equipment. The required manpower for the typical year 5 has been shown in Table 21.43.

Table 21.43 – Estimated Tailings Manpower Requirements

Position	Year 5
Operation and Maintenance	
Foreman	2
Equipment Operator	12
Labour	12
Mechanic\Welder	4
Electrician\Instrumentation	2
TOTAL	**32**

21.6.8.4 General and Administration Manpower

The site services and administration operations manpower has been estimated and the required manpower for the typical year has been shown in Table 21.44 and Table 21.45.

Table 21.44 – Manpower General and Administration (On-Site) Manpower Requirements

Position	Year 5
Administration	
General Manager	1
Accounting	
Junior Accountant	2
Accounts Payable Clerk	6
Human Resources	
Human Resources Supervisor	1
Human Resources Administrative Assistant	1
Training Coordinator	2
Trainer	6
Health & Safety and Environment	
Health & safety Prevention officer	1
Senior Health & Safety Coordinator	1
Health & Safety Coordinator	2
Nurse	2
Environmental Coordinator	1
Environmental Technician	4
Supply Chain	
Buyer	1
Administrative Assistant	1
Warehouse Foreman	1
Inventory Analyst	1
Warehouse Clerk	8
Service Loader Operator	4
Site Maintenance	
Site Maintenance Manager	2
Electrical	16
Mechanical	12
Labour	16
Other	
Community Relations Manager	1
IT Manager	1
IT Technician	2
Security Officer	16
TOTAL	**112**

Table 21.45 – Manpower General and Administration (Off-Site) Manpower Requirements

Position	Year 5
Corporate (Toronto)	
Chief Executive Officer	1
Chief Financial Officer	1
Chief Operation Officer	1
Vice-President	2
Accounting (Toronto)	
Senior Accountant	1
Payroll Supervisor	1
Junior Accountant	2
Accountant Payable Supervisor	1
Accounts Payable Clerk	2
Human Resources (Toronto)	
Human Resources Manager	1
Health & Safety and Environment (Toronto)	
Sustainable Development Manager	1
Supply Chain (Toronto)	
Supply Chain Manager	1
Senior Buyer	1
Senior Logistic Coordinator & Camp Management	1
TOTAL	**17**

21.6.8.5 Rail and Port Site Operation Manpower

The Rail and Port operations manpower has been estimated and the required manpower for the typical year has been shown in Table 21.46.

Table 21.46 – Estimated Port Site Operations Manpower Requirements

Position	Year 5
Operation	
Foreman	2
Operator	4
Labour	8
TOTAL	**14**

21.6.8.6 Manpower for Pellet Plant Operation

The pellet plant operations manpower has been estimated for three sections namely Administration and Operations/Maintenance. The required manpower for the typical year has been shown in Table 21.47.

Table 21.47 – Estimated Crushing and Concentrator Manpower Requirements

Position	Year 5
Administration	
Plant Superintendent	1
Plant General Foreman	1
Junior Accountant	1
Accounts Payable Clerk	2
Senior Health & Safety Coordinator	1
Health & Safety Coordinator	2
Nurse	2
Environmental Coordinator	1
Environmental Technician	2
Buyer	1
Administrative Assistant	1
Warehouse Foreman	1
Inventory Analyst	1
Warehouse Clerk	4
Service Loader Operator	4
IT Technician	2
Security Officer	8
Site Maintenance Manager	1
Electrical	2
Mechanical	2
Labour	2
Operation and Maintenance	
Mechanical Engineer	2
Electrical Engineer	2
Maintenance Planner	2
Foreman	16
Control Room Operator	8
Operator	48
Labour	24
Mechanic\Welder	32
Electrician\Instrumentation	32
Programming Technician	4
TOTAL	**212**

22 Economic Analysis

22.1 General

A preliminary economic analysis has been carried out for the Full Moon Project using a cash flow model. The model is constructed using annual cash flows in constant first-quarter 2015 Canadian dollars and is based on a combined iron concentrate/pellet production of some 20 million tonnes per year over a mine life limited to 30 years. Four production options are considered: HSC only, HSC & HSF pellets, LSC only, and LSC & DR pellets.

The selling prices of the mine products are based on a 62% iron concentrate price forecast of US$95 per tonne (CFR China). An exchange rate of US$0.80 per CAD is assumed to convert the revenue estimates into Canadian dollars.

The financial assessment is carried out on a "100% equity" basis, i.e. the debt and equity sources of capital funds are ignored. No provision is made for the effects of inflation. Results are given before and after taxation. Current Canadian tax regulations are applied to assess the corporate tax liabilities while the recently proposed regulations in Quebec (Bill 55, December 2013) are applied to assess the mining tax liabilities.

The model reflects the technical assumptions documented in the foregoing sections of the Report and assumes the owner will operate the project.

22.2 Assumptions

22.2.1 Economic Assumptions

The selling prices of the mine products as discussed in Section 19 are based on a 62% iron concentrate price forecast of US$95 per tonne (CFR China). This price basis is adjusted for the actual iron and silica content of the concentrate produced, and a relevant pellet premium is added when required to determine the selling price of the mine products. Product transportation costs of US$15 per tonne from Sept-Îles to the China market are assumed to derive the FOB Sept-Îles prices. The resulting selling prices are shown in Table 22.1 below.

Table 22.1 – Selling Price Assumptions

Description	Units	CFR China	FOB Sept-Îles
HSC, 66 % Fe	US$/t	112	97
HSF Pellets	US$/t	135	120
LSC, 66 % Fe	US$/t	118	103
DR Pellets	US$/t	140	125

An exchange rate of US$0.80 per CAD is assumed to convert project revenues into Canadian currency. The U.S. content of the capital costs was converted using the same exchange rate.

The following Discount Rates are assumed to determine net present values ("NPV"):

- Base Case 8%
- Variant 1 6%
- Variant 2 10%

22.2.2 Royalty and Impact and Benefit Agreements

The project is royalty-free and no Impact and Benefit Agreement has been negotiated at this stage of project development.

22.2.3 Production Assumptions

As detailed in Table 22.2, four production options are considered for economic assessment.

Table 22.2 – Material Produced by Option

Area	Option 1 (Preferred)	Option 2	Option 3	Option 4
Final Produce (tonnes)	**HSC**	**LSC**	**HSC – Pellets**	**LSC - Pellets**
High Silica Concentrate	20,000,000		4,200,000	
Low Silica Concentrate		18,300,000		2,000,000
HSF Pellet			17,000,000	
DR Pellet				17,000,000

It is assumed that mine production starts in year 2020 and is limited to a life of 30 years.

22.2.4 Technical Assumptions

Technical assumptions required in the economic assessments and described in previous sections of the Report are summarized in Table 22.3.

Table 22.3 – Technical Assumptions

Description	Units	
Total Resources to Mill (LOM)	M tonnes	1,609.6
Average Grade of Resources to Mill (LOM)	% Fe	30.8
Total Waste and Overburden (LOM)	M tonnes	153.2
Average Stripping Ratio	w : o	0.095
Pre-production Period	years	3
Mine Life	years	30
Average Weight Recovery	%	37.1
Total Production (LOM)		
Option 1 – HSC	M tonnes	597.0
Option 2 – LSC	M tonnes	546.2
Option 3 – HSF Pellets	M tonnes	507.4
– HSC	M tonnes	125.4
Option 4 – DR Pellets	M tonnes	507.4
– LSC	M tonnes	59.7
Total Pre-production Capital Costs[1]		
Option 1	M$	7,207.3
Option 2	M$	7,385.5
Option 3	M$	8,886.1
Option 4	M$	9,064.3
Total Sustaining Capital Costs (LOM)	M$	658.0
Mine Closure Costs	M$	178.2
Average Operating Costs (from financial model)		
Option 1	$/t mill feed	18.49
Option 2	$/t mill feed	18.90
Option 3	$/t mill feed	22.02
Option 4	$/t mill feed	22.29
Average Operating Costs (from financial model)		
Options 1 and 3	$/t conc.	49.85
Options 2 and 4	$/t conc.	55.70
Option 3	$/t pellets	57.52
Option 4	$/t pellets	64.14
1. Includes mine development but excludes working capital.		

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22.2.5 Financial Model and Results

Figure 22.1 shows the before-tax cash flows as well as the cumulative cash flow over the project's life for Option 1 (preferred). The payback period corresponds to the time at which the cumulative cash flow becomes positive (between years 5 and 6). The cash flow statement for Option 1 is shown in Table 22.4.

Figure 22.1 – Before-tax Cash Flows and Cumulative Cash Flow – Option 1

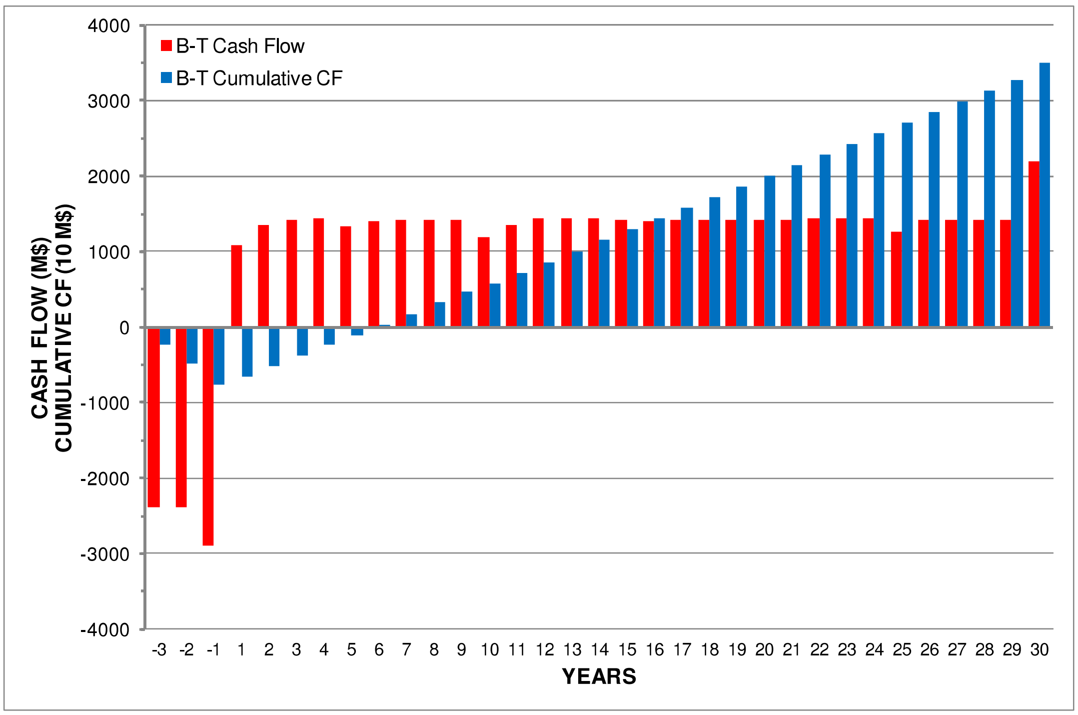


Table 22.4 – Cash Flow Statement – Option 1

PEA Full Moon

ANNUAL CASH FLOWS

All monetary values in CAD	Unit	Y-3 2017	Y-2 2018	Y-1 2019	Y1 2020	Y2 2021	Y3 2022	Y4 2023	Y5 2024	Y6 2025	Y7 2026	Y8 2027
REVENUES												
Low Silicate Option Production												
DR Pellets	t				0	0	0	0	0	0	0	0
Low Silicate Concentrate	t				0	0	0	0	0	0	0	0
Hi Silicate Option Production												
HSF Pellets	t				0	0	0	0	0	0	0	0
Hi Silicate Concentrate	t				16 979 385	19 975 095	20 019 305	20 023 356	19 976 576	19 958 683	20 033 974	20 020 670
Low Silicate Option Prices												
DR Pellets	$/t				175.00	175.00	175.00	175.00	175.00	175.00	175.00	175.00
Low Silicate Concentrate	$/t				147.50	147.50	147.50	147.50	147.50	147.50	147.50	147.50
Hi Silicate Option Prices												
HSF Pellets	$/t				168.75	168.75	168.75	168.75	168.75	168.75	168.75	168.75
Hi Silicate Concentrate	$/t				140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00
Revenue CFR	$				2 377 113 923	2 796 513 313	2 802 702 707	2 803 269 837	2 796 720 684	2 794 215 652	2 804 756 373	2 802 893 751
Concentrate Transport	$				318 363 472	374 533 033	375 361 970	375 437 925	374 560 806	374 225 311	375 637 014	375 387 556
Revenue FOB					2 058 750 451	2 421 980 280	2 427 340 737	2 427 831 912	2 422 159 878	2 419 990 341	2 429 119 359	2 427 506 195
OPERATING COSTS												
Mine	$				100 318 887	118 316 307	114 945 170	109 364 810	111 713 157	121 697 968	120 735 595	115 105 637
Concentrator	$				246 812 116	258 535 384	258 708 393	258 724 246	258 541 181	258 471 159	258 765 799	258 713 734
Tailing Operation	$				14 534 251	14 534 251	14 534 251	14 534 251	14 534 251	14 534 251	14 534 251	14 534 251
General & Administration	$				52 498 643	52 827 143	52 892 843	52 827 143	52 892 843	53 024 243	53 024 243	53 024 243
Rail & Port Operation	$				473 641 359	556 924 292	558 153 361	558 265 980	556 965 471	556 468 030	558 561 171	558 191 298
Pellet Plant Operation	$				0	0	0	0	0	0	0	0
Total Operating Costs	$				887 805 255	1 001 137 378	999 234 019	993 716 430	994 646 903	1 004 195 651	1 005 621 459	999 569 163
CAPITAL COST – DIRECT												
0000 - Mine Development	$	0	0	48 013 383								
1000 - Mine	$	80 779 769	80 779 769	80 779 769								
2000/6600 - Concentrator / Pellet Plant	$	837 950 801	837 950 801	837 950 801								
3000 - Tailings	$	150 126 312	150 126 312	150 126 312								
4000 - Infrastructures	$	286 600 774	286 600 774	286 600 774								
5000 - Rail	$	147 033 514	147 033 514	147 033 514								
CAPITAL COST – INDIRECT												
Total Owner's Costs & Contingency	$	883 943 124	883 943 124	883 943 124								
Total Direct & Indirect Capital	$	2 386 434 295	2 386 434 295	2 434 447 678								
CAPITAL COST – SUSTAINING												
Total	$				0	24 995 387	1 412 800	0	94 015 000	11 671 400	0	386 250
SALVAGE VALUE ($)	5.0%											
WORKING CAPITAL ($)	5.0			369 918 856	47 221 718	-793 066	-2 298 995	387 697	3 978 645	594 086	-2 521 790	-3 088 853
Cumulative Working Capital	$			369 918 856	417 140 574	416 347 508	414 048 512	414 436 209	418 414 855	419 008 941	416 487 151	413 398 298
SITE REHABILITATION												
Trust Fund Payments	$			89 105 500	44 552 750	44 552 750	0	0	0	0	0	0
Closure Costs	$			0	0	0	0	0	0	0	0	0
Total Tax Payments	$		0	0	18 146 835	20 626 649	20 596 483	90 252 024	249 915 361	288 676 028	325 387 749	393 188 585
Federal Corporate Taxes	$	0		0	0	0	0	36 730 549	121 847 897	138 752 003	155 650 246	160 070 830
Provincial Corporate Taxes	$	0		0	0	0	0	29 139 569	96 665 999	110 076 589	123 482 528	126 989 525
Mining Taxes	$	0		0	18 146 835	20 626 649	20 596 483	24 381 906	31 401 465	39 847 437	46 254 975	106 128 230
B-T CASH FLOW	$	-2 386 434 295	-2 386 434 295	-2 893 472 034	1 079 170 728	1 352 087 832	1 428 992 914	1 433 727 785	1 329 519 330	1 403 529 204	1 426 019 690	1 430 639 636
Cumulative	$	-2 386 434 295	-4 772 868 589	-7 666 340 623	-6 587 169 896	-5 235 082 064	-3 806 089 150	-2 372 361 365	-1 042 842 035	360 687 169	1 786 706 859	3 217 346 494
Payback Period work area				1.00	1.00	1.00	1.00	1.00	1.00	0.74	0.00	0.00
A-T CASH FLOW	$	-2 386 434 295	-2 386 434 295	-2 893 472 034	1 061 023 892	1 331 461 183	1 408 396 431	1 343 475 762	1 079 603 969	1 114 853 175	1 100 631 941	1 037 451 051
Cumulative	$	-2 386 434 295	-4 772 868 589	-7 666 340 623	-6 605 316 731	-5 273 855 548	-3 865 459 117	-2 521 983 355	-1 442 379 386	-327 526 211	773 105 730	1 810 556 780
Payback Period work area				1.00	1.00	1.00	1.00	1.00	1.00	1.00	0.30	0.00

All monetary values in CAD	Y9 2028	Y10 2029	Y11 2030	Y12 2031	Y13 2032	Y14 2033	Y15 2034	Y16 2035	Y17 2036	Y18 2037	Y19 2038
DR Pellets	0	0	0	0	0	0	0	0	0	0	0
Low Silicate Concentrate	0	0	0	0	0	0	0	0	0	0	0
HSF Pellets	0	0	0	0	0	0	0	0	0	0	0
Hi Silicate Concentrate	19 966 527	20 027 241	19 997 892	19 997 892	19 997 892	19 997 892	19 997 892	20 007 147	20 007 147	20 007 147	20 007 147
DR Pellets ($/t)	175.00	175.00	175.00	175.00	175.00	175.00	175.00	175.00	175.00	175.00	175.00
Low Silicate Concentrate ($/t)	147.50	147.50	147.50	147.50	147.50	147.50	147.50	147.50	147.50	147.50	147.50
HSF Pellets ($/t)	168.75	168.75	168.75	168.75	168.75	168.75	168.75	168.75	168.75	168.75	168.75
Hi Silicate Concentrate ($/t)	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00
Revenue CFR	2 795 313 824	2 803 813 786	2 799 704 819	2 799 704 819	2 799 704 819	2 799 704 819	2 799 704 819	2 801 000 541	2 801 000 541	2 801 000 541	2 801 000 541
Concentrate Transport	374 372 387	375 510 775	374 960 467	374 960 467	374 960 467	374 960 467	374 960 467	375 134 001	375 134 001	375 134 001	375 134 001
Revenue FOB	2 420 941 437	2 428 303 011	2 424 744 352	2 424 744 352	2 424 744 352	2 424 744 352	2 424 744 352	2 425 866 540	2 425 866 540	2 425 866 540	2 425 866 540
Mine	109 475 163	108 601 349	107 572 181	107 572 181	107 572 181	107 572 181	107 572 181	115 013 624	115 013 624	115 013 624	115 013 624
Concentrator	258 501 856	258 739 451	258 624 595	258 624 595	258 624 595	258 624 595	258 624 595	258 660 814	258 660 814	258 660 814	258 660 814
Tailing Operation	14 534 251	14 534 251	14 534 251	14 534 251	14 534 251	14 534 251	14 534 251	14 534 251	14 534 251	14 534 251	14 534 251
General & Administration	52 958 543	52 958 543	52 958 543	52 958 543	52 958 543	52 958 543	52 958 543	53 024 243	53 024 243	53 024 243	53 024 243
Rail & Port Operation	556 686 102	558 373 996	557 558 051	557 558 051	557 558 051	557 558 051	557 558 051	557 815 351	557 815 351	557 815 351	557 815 351
Pellet Plant Operation	0	0	0	0	0	0	0	0	0	0	0
Total Operating Costs	992 155 914	993 207 590	991 247 621	991 247 621	991 247 621	991 247 621	991 247 621	999 048 283	999 048 283	999 048 283	999 048 283
Capital Cost – Sustaining Total	0	239 564 247	83 691 812	0	0	0	0	24 995 387	0	0	0
WORKING CAPITAL ($)	438 198	-816 654	0	0	0	0	3 250 276	0	0	0	0
Cumulative Working Capital	413 836 496	413 019 842	413 019 842	413 019 842	413 019 842	413 019 842	416 270 118	416 270 118	416 270 118	416 270 118	416 270 118
Trust Fund Payments	0	0	0	0	0	0	0	0	0	0	0
Closure Costs	0	0	0	0	0	0	0	0	0	0	0
Total Tax Payments	432 743 639	439 197 927	444 286 802	459 310 316	473 342 129	484 611 943	492 848 700	493 938 728	498 084 492	501 942 244	504 771 130
Federal Corporate Taxes	165 191 382	168 821 638	168 045 807	171 553 863	175 763 501	178 767 047	181 063 519	181 817 988	182 728 489	183 898 767	184 789 593
Provincial Corporate Taxes	131 051 830	133 931 833	133 316 340	136 099 398	139 439 044	141 821 857	143 643 725	144 242 271	144 964 601	145 893 022	146 599 744
Mining Taxes	136 500 426	136 444 457	142 924 656	151 657 055	158 139 585	164 023 039	168 141 457	167 878 469	170 391 402	172 150 456	173 381 793
B-T CASH FLOW	1 428 347 325	1 196 347 829	1 349 804 920	1 433 496 731	1 433 496 731	1 433 496 731	1 430 246 455	1 401 822 871	1 426 818 257	1 426 818 257	1 426 818 257
Cumulative	4 645 693 819	5 842 041 648	7 191 846 568	8 625 343 299	10 058 840 030	11 492 336 761	12 922 583 216	14 324 406 087	15 751 224 344	17 178 042 601	18 604 860 858
Payback Period work area	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
A-T CASH FLOW	995 603 686	757 149 901	905 518 117	974 186 416	960 154 602	948 884 788	937 397 755	907 884 143	928 733 765	924 876 013	922 047 127
Cumulative	2 806 160 467	3 563 310 368	4 468 828 485	5 443 014 901	6 403 169 503	7 352 054 291	8 289 452 046	9 197 336 189	10 126 069 954	11 050 945 966	11 972 993 093
Payback Period work area	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00

All monetary values in CAD	Y20 2039	Y21 2040	Y22 2041	Y23 2042	Y24 2043	Y25 2044	Y26 2045	Y27 2046	Y28 2047	Y29 2048	Y30 2049	TOTAL
DR Pellets	0	0	0	0	0	0	0	0	0	0	0	0
Low Silicate Concentrate	0	0	0	0	0	0	0	0	0	0	0	0
HSF Pellets	0	0	0	0	0	0	0	0	0	0	0	0
Hi Silicate Concentrate	20 007 147	19 998 993	19 998 993	19 998 993	19 998 993	19 998 993	19 996 638	19 996 638	19 996 638	19 996 638	19 996 638	596 984 159
DR Pellets ($/t)	175.00	175.00	175.00	175.00	175.00	175.00	175.00	175.00	175.00	175.00	175.00	
Low Silicate Concentrate ($/t)	147.50	147.50	147.50	147.50	147.50	147.50	147.50	147.50	147.50	147.50	147.50	
HSF Pellets ($/t)	168.75	168.75	168.75	168.75	168.75	168.75	168.75	168.75	168.75	168.75	168.75	
Hi Silicate Concentrate ($/t)	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	140.00	
Revenue CFR	2 801 000 541	2 799 859 018	2 799 859 018	2 799 859 018	2 799 859 018	2 799 859 018	2 799 529 309	2 799 529 309	2 799 529 309	2 799 529 309	2 799 529 309	83 577 782 279
Concentrate Transport	375 134 001	374 981 118	374 981 118	374 981 118	374 981 118	374 981 118	374 936 961	374 936 961	374 936 961	374 936 961	374 936 961	11 193 452 984
Revenue FOB	2 425 866 540	2 424 877 899	2 424 877 899	2 424 877 899	2 424 877 899	2 424 877 899	2 424 592 348	2 424 592 348	2 424 592 348	2 424 592 348	2 424 592 348	72 384 329 295
Mine	115 013 624	109 186 036	109 186 036	109 186 036	109 186 036	109 186 036	110 653 823	110 653 823	110 653 823	110 653 823	110 627 996	3 342 376 938
Concentrator	258 660 814	258 628 905	258 628 905	258 628 905	258 628 905	258 628 905	258 619 689	258 619 689	258 619 689	258 619 689	258 619 689	7 747 183 331
Tailing Operation	14 534 251	14 534 251	14 534 251	14 534 251	14 534 251	14 534 251	14 534 251	14 534 251	14 534 251	14 534 251	14 534 251	436 027 540
General & Administration	53 024 243	53 024 243	53 024 243	53 024 243	53 024 243	53 024 243	53 024 243	53 024 243	53 024 243	53 024 243	52 998 034	1 589 058 576
Rail & Port Operation	557 815 351	557 588 671	557 588 671	557 588 671	557 588 671	557 588 671	557 523 198	557 523 198	557 523 198	557 523 198	557 521 854	16 644 656 068
Pellet Plant Operation	0	0	0	0	0	0	0	0	0	0	0	0
Total Operating Costs	999 048 283	992 962 107	992 962 107	992 962 107	992 962 107	992 962 107	994 355 204	994 355 204	994 355 204	994 355 204	994 301 825	29 759 302 453
0000 - Mine Development												48 013 383
1000 - Mine												242 339 307
2000/6600 - Concentrator / Pellet Plant												2 513 852 402
3000 - Tailings												450 378 936
4000 - Infrastructures												859 802 323
5000 - Rail												441 100 543
Total Owner's Costs & Contingency												2 651 829 372
Total Direct & Indirect Capital												7 207 316 267
Capital Cost – Sustaining Total	0	1 541 550	0	0	0	175 700 000	0	0	0	0	0	657 973 831
SALVAGE VALUE ($)											357 965 144	357 965 144
WORKING CAPITAL ($)	-2 535 907	0	0	0	0	580 457	0	0	0	-22 241	-414 292 427	0
Cumulative Working Capital	413 734 211	413 734 211	413 734 211	413 734 211	413 734 211	414 314 668	414 314 668	414 314 668	414 314 668	414 292 427	0	
Trust Fund Payments	0	0	0	0	0	0	0	0	0	0	-178 211 000	0
Closure Costs	0	0	0	0	0	0	0	0	0	0	178 211 000	178 211 000
Total Tax Payments	506 847 694	510 455 885	511 563 612	512 426 157	513 062 956	499 147 694	496 889 767	501 510 442	504 884 153	507 350 181	663 961 997	12 359 966 303
Federal Corporate Taxes	185 466 895	186 560 472	186 924 518	187 229 835	187 461 187	186 081 412	183 023 898	184 200 120	185 101 118	185 790 049	228 014 590	4 641 347 211
Provincial Corporate Taxes	147 137 070	148 004 641	148 293 451	148 535 669	148 719 209	147 624 587	145 198 959	146 132 095	146 846 887	147 393 439	180 891 575	3 682 135 454
Mining Taxes	174 243 729	175 890 772	176 345 643	176 660 653	176 882 560	165 441 695	168 666 911	171 178 227	172 936 148	174 166 693	255 055 832	4 036 483 637
B-T CASH FLOW	1 429 354 164	1 430 374 243	1 431 915 793	1 431 915 793	1 431 915 793	1 255 635 335	1 430 237 144	1 430 237 144	1 430 237 144	1 430 259 385	2 202 548 094	34 939 490 888
Cumulative	20 034 215 022	21 464 589 264	22 896 505 057	24 328 420 850	25 760 336 642	27 015 971 977	28 446 209 121	29 876 446 265	31 306 683 409	32 736 942 794	34 939 490 888	
Payback Period work area	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
A-T CASH FLOW	922 506 470	919 918 357	920 354 181	919 489 636	918 852 836	756 487 641	933 347 377	928 726 702	925 352 991	922 909 204	1 538 586 098	22 579 524 585
Cumulative	12 895 499 563	13 815 417 921	14 735 772 101	15 655 261 737	16 574 114 573	17 330 602 214	18 263 949 591	19 192 676 293	20 118 029 284	21 040 938 488	22 579 524 585	
Payback Period work area	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	

Before-Tax		
Payback Period	(years)	5.7
Net Present Value @	6.0%	9 233 634 673
Net Present Value @	8.0%	5 770 965 574
Net Present Value @	10.0%	3 395 153 783
Internal Rate of Return		15.2%
After-Tax		
Payback Period	(years)	6.3
Net Present Value @	6.0%	5 326 167 278
Net Present Value @	8.0%	2 965 301 261
Net Present Value @	10.0%	1 334 138 201
Internal Rate of Return		12.4%

The cash flow statement shows both the proceeds from the sale of the concentrate (and pellets for options 3 and 4) CFR China market and the revenue FOB Sept-Îles, net of concentrate shipping charges to China. The former is required in the Quebec mining tax assessment. The operating costs are listed by component. The pre-production capital costs are listed by component and have been allocated over a 3-year pre-production period in equal amounts. A salvage value of 5 % of total pre-production capital costs (excluding mine development) is assumed. A working capital equivalent to 5 months of operating costs is assumed. Working capital levels vary over the life of mine as annual operating costs increase and decrease. The estimated closure costs are secured in a trust fund at the beginning of mining operations. Accordingly, it is assumed that trust fund payments are made in the last pre-production year and in the first two years of production in the proportions of 50/25/25 %, respectively.

The financial indicators for the Full Moon Project are summarized in Table 22.5. On a before-tax basis, Option 1 (preferred) has a Net Present Value of M$5,771.0 at a discount rate of 8 %, an Internal Rate of Return of 15.2% and a payback period of 5.7 years. On an after-tax basis, the NPV is M$2,965.3 at a discount rate of 8 %, the IRR is 12.4% and the payback period is 6.3 years.

Table 22.5 – Summary of Financial Results

Description	Units	Option 1 (Preferred)	Option 2	Option 3	Option 4
Total Revenue FOB Sept-Îles (LOM)	M$	72,384.3	70,328.5	91,316.2	86,973.1
Total Operating Costs (LOM)	M$	29,759.3	30,424.2	35,436.6	35,869.5
Total Pre-production Capital Costs	M$	7,207.3	7,385.5	8,886.1	9,064.3
Total Sustaining Capital Costs (LOM)	M$	658.0	658.0	658.0	658.0
Initial Working Capital	M$	369.9	378.6	439.5	445.4
Mine Closure Costs	M$	178.2	178.2	178.2	178.2
Salvage Value	M$	358.0	366.9	441.9	450.8
BEFORE TAX					
Total Cash Flow	M$	34,939.5	32,049.5	46,599.2	41,654.0
Payback Period	years	5.7	6.3	5.4	6.0
NPV @ 8%	M$	5,771.0	4,806.7	8,196.0	6,626.3
NPV @ 6%	M$	9,233.6	8,026.4	12,772.2	10,779.7
NPV @ 10%	M$	3,395.2	2,604.2	5,048.3	3,779.1
IRR	%	**15.2**	**13.9**	**16.2**	**14.6**
AFTER TAX					
Total Tax Payments (LOM)	M$	12,360.0	11,170.1	16,321.7	14,323.0
Total Cash Flow	M$	22,579.5	20,879.4	30,277.5	27,330.9
Payback Period	years	6.3	6.8	5.9	6.5
NPV @ 8%	M$	2,965.3	2,335.8	4,418.9	3,409.1
NPV @ 6%	M$	5,326.2	4,560.4	7,539.7	6,285.5
NPV @ 10%	M$	1,334.1	802.8	2,258.5	1,423.5
IRR	%	**12.4**	**11.4**	**13.2**	**12.0**

Note: The calculation of the payback period is based on production start-up in 2020.

22.3 Sensitivity Analysis

A sensitivity analysis was performed with the following three (3) parameters:

- Pre-production capital costs (excluding mine development and working capital that vary with operating costs);
- Operating costs;
- CFR selling prices of iron concentrate and pellets.

Each variable was examined one-at-a-time. An interval of ± 30% with increments of 10% was used for all three variables.

The before-tax results of the sensitivity analysis for Option 1 are shown in Figure 22.2 and Figure 22.3. Results show that, within the limits of accuracy of the cost estimates in this study, the Project's before-tax viability does not seem significantly vulnerable to the under-estimation of capital and operating costs, taken one at-a-time. As seen in Figure 22.2, the project's net present value is more sensitive to changes in operating costs ("OPEX") than to changes in pre-production capital costs ("CAPEX"), as evidenced by the steeper slope of the OPEX curve. As expected however, the Project's financial performance is most sensitive to changes in selling price ("PRICE"). It can be observed that the Project breaks even (i.e., NPV=0) at a selling price about 23 % lower than the price forecast (i.e., at a CFR price of approximately US$86 per tonne of HSC).

The sensitivity graphs for Options 2, 3 and 4, not shown in the Report are quite similar to those of Option 1, except for the fact that the break-even selling prices are different. The relevant values are as follows:

- Option 2: about 20 % lower than the price forecast (i.e., at a CFR price of approximately US$94 per tonne for LSC);
- Option 3: about 27 % lower than the price forecasts (i.e., at CFR prices of approximately US$82 and US$99 per tonne, for HSC and HSF pellets, respectively);
- Option 4: about 23 % lower than the price forecasts (i.e., at CFR prices of approximately US$91 and US$108 per tonne, for LSC and DR pellets, respectively).

Figure 22.2 – Sensitivity Analysis: Before-Tax NPV @ 8%

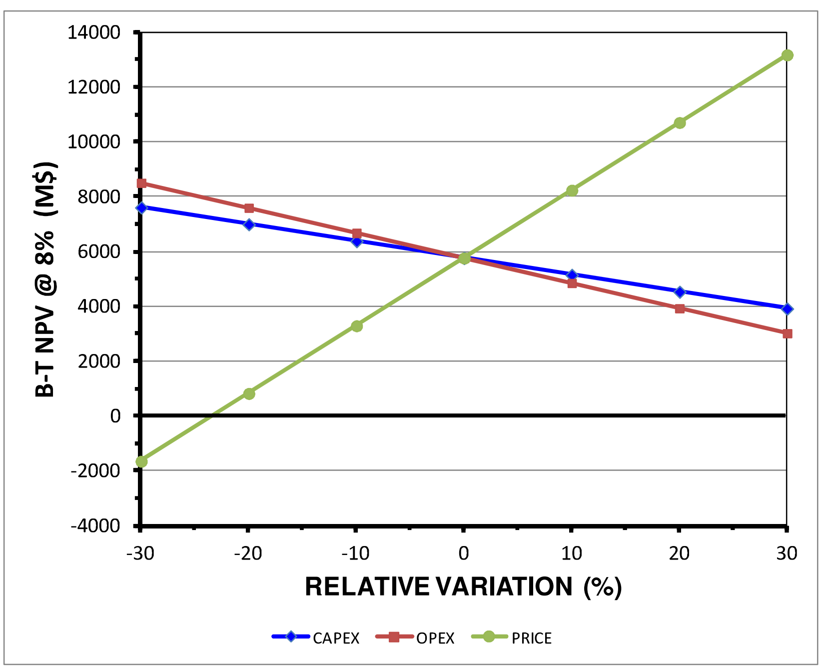
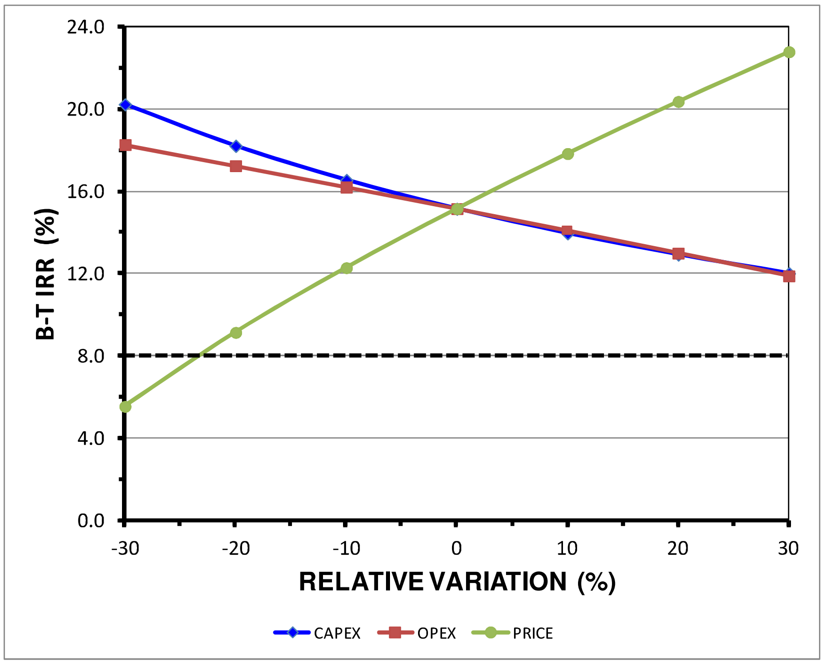


Figure 22.3, showing variations in internal rate of return for Option 1, provides the same conclusions. Compared to Figure 22.2, which shows linear variations in net present value for the three (3) variables studied, variations associated with internal rate of return are not linear. Because of the different timing associated with CAPEX versus OPEX, the IRR is more sensitive to negative variations in CAPEX than to OPEX, but is of equal sensitivity for positive variations.

Figure 22.3 – Sensitivity Analysis: Before-Tax IRR



The after-tax results of the sensitivity analysis for Option 1 are shown in Figure 22.4 and Figure 22.5. The same conclusions as those drawn for the before-tax case can be made here concerning the sensitivity of the project to variations in CAPEX, OPEX and PRICE. On an after-tax basis, the project breaks even at a selling price about 18 % lower than the price forecast (i.e., at a CFR price of approximately US$92 per tonne of HSC).

The sensitivity graphs for Options 2, 3 and 4, not shown in the Report are quite similar to those of Option 1, except for the fact that the break-even selling prices are different. The relevant values are as follows:

- Option 2: about 15 % lower than the price forecast (i.e., at a CFR price of approximately US$101 per tonne for LSC);
- Option 3: about 22 % lower than the price forecasts (i.e., at CFR prices of approximately US$87 and US$105 per tonne, for HSC and HSF pellets, respectively);

- Option 4: about 18 % lower than the price forecasts (i.e., at CFR prices of approximately US$97 and US$115 per tonne, for LSC and DR pellets, respectively).

Figure 22.4 – Sensitivity Analysis: After-Tax NPV @ 8%

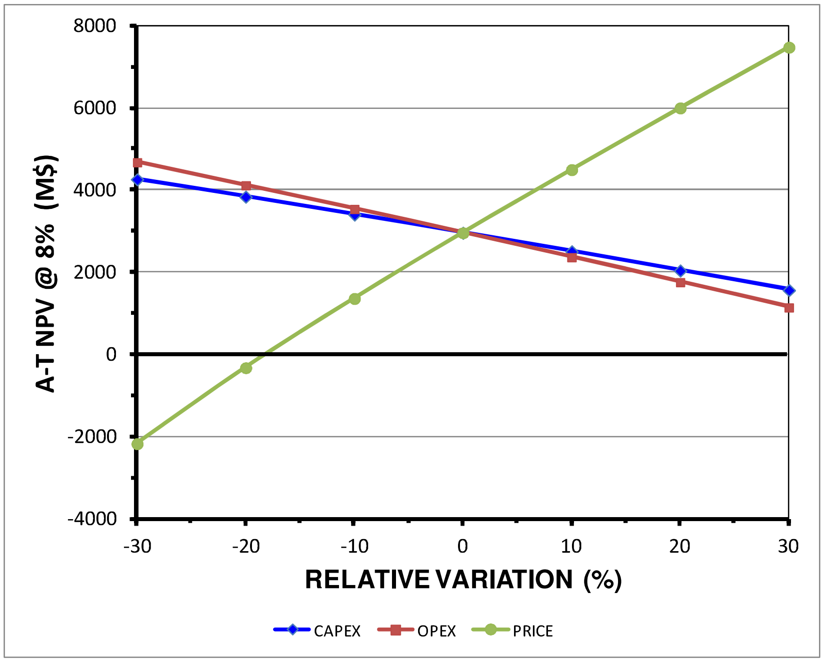
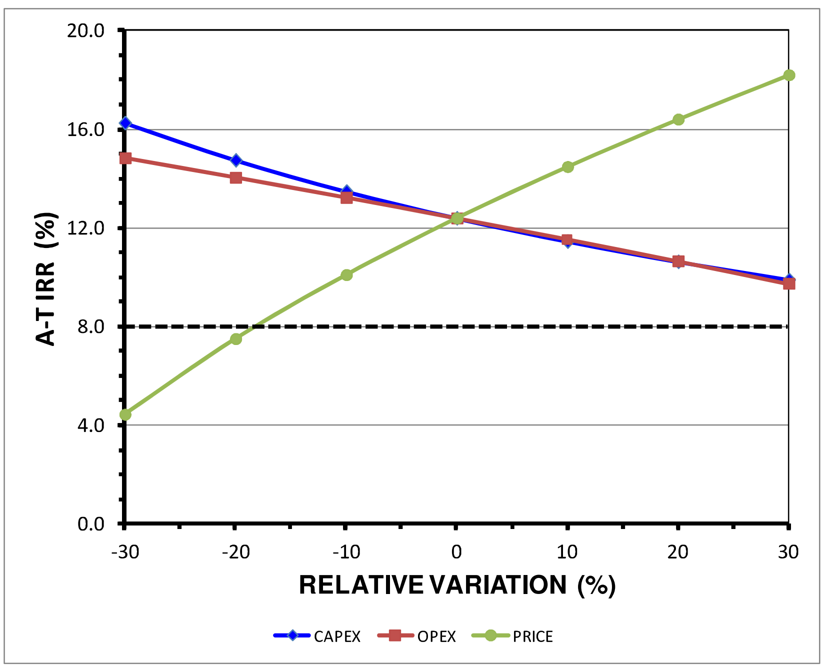


Figure 22.5 – Sensitivity Analysis: After-Tax IRR



22.4 Important Caution Regarding the Economic Analysis

The economic analysis contained in this report is preliminary in nature. It incorporates inferred mineral resources that are considered too geologically speculative to have economic considerations applied to them that would enable them to be categorized as mineral reserves. This should not be considered a prefeasibility or feasibility study. There can be no certainty that the estimates contained in this report will be realized. In addition, mineral resources that are not mineral reserves do not have demonstrated economic viability.

23 Adjacent Properties

A large number of iron properties are located in the Québec – Labrador area. The Labrador Trough is one of the largest iron ore belts in the world. Two types of iron ore are currently mined and/or explored namely magnetite deposits (requiring concentration before shipping) and hematite ore deposits (Direct Shipping Ore deposits). The Rainy Lake property, part of the Full Moon project, is located in this area.

The Rainy Lake property, including the mineral resource reported is located entirely within the Province of Québec and is the magnetite type of deposit.

The following companies have iron ore projects close by that currently under development or under investigation:

- ArcelorMittal Mines Canada;
  - Mount-Wright Mining Complex (taconite type deposit)
  - Fire Lake Mine (taconite type deposit)
- Cliffs Natural Resources;
  - Bloom Lake Mine (taconite type deposit)
  - Peppler Lake deposits (exploration of taconite type deposits)
- Champion Iron Limited;
  - Fire Lake North Project (exploration of taconite type deposit)
- Labrador Iron Mine Holdings Limited ("LIM");
  - James Mine and exploration (of DSO type deposits)
- Tata Steel Minerals Canada Limited ("TSMC"),¨), a joint venture between Tata Steel Limited and New Millennium Iron Corp:
  - TSMC DSO Project (DSO mining project);
  - TSMC Taconite Project (exploration of two taconite type deposits)
- Cap-Ex Iron Ore Ltd.
  - Block 103 (exploration of taconite type project);

- Lac Connelly and Snelgrove (exploration of DSO type projects)

- WISCO/Adriana Resources Inc.
  - Lac Otelnuk Iron Project (exploration of taconite type deposit)

The descriptions in this section are drawn from publicly disclosed information by the owners of the adjacent properties. The qualified person has been unable to verify the information.

23.1 ArcelorMittal Mines Canada

Mount-Wright Mining Complex in Fermont

There are some 1,000 employees at the Mont-Wright Mining Complex, which comprises an open-pit mine, an ore crusher and a concentrator, huge maintenance workshops, a large spare parts storage facility and a train loading system.

The mine, extending over 24 square kilometres, has reserves and resources of one billion tonnes of crude ore with an iron content of approximately 30%. Generally, every 2.6 tonnes of crude ore yields 1 tonne of concentrate.

According to an established plan, drilling machine operators carve deep holes (15.8 metres) in the ore-bearing rock, into which an explosive mixture is poured and blasted to break the rock. Blasting operations, each requiring about sixty holes, are carried out four to five times a week.

Electric power shovels (with bucket size of 35 m3 and – less frequently – large-capacity loaders, load the blasted rock onto 250 tonne production trucks.

Each day, production truck drivers make about 1,000 runs from the mine, most of them to the unloading point: the ore crusher. Truck boxes are unloaded into one of two gyratory crushers, which break the ore into pieces some 20 centimetres in diameter. The crushed ore is moved by conveyor to one of the six storage silos in the concentrator.

The concentration process is to finely grind the ore in one of the six autogenous mills, then screened over vibrating screens. Particles too large to pass through the screens are returned to the grinding mill. The remainder is routed to the concentrator's 8,640 spirals, divided into three separate circuits, in order to increase the iron content of the crude magnetite ore.

The concentrate is processed through filter tables to remove the water, and routed to the loading silo to be put on trains bound for Port-Cartier.

Fire Lake Open-Pit Mine

The open-pit mine at Fire Lake, located 55 kilometres south of the Mont-Wright Mining Complex, is another magnetite deposit now worked because of the high demand for iron ore products.. The mine operates solely between May and October, when the ground thaws.

The Fire Lake mine site has neither a crusher nor a concentrator, though the extraction sequence is the same as at Mont-Wright. All crude ore from Fire Lake is transported to Mont-Wright by train, over the rail link that connects Fire Lake to the ArcelorMittal main railway line.

At Mont-Wright, the ore is carried to the crusher and broken up into fragments some 20 centimetres in diameter. The fragmented ore is then carried to the concentrator, where it is goes through the regular concentration process.

23.2 Cliffs Natural Resources

Bloom Lake Mine

The Bloom Lake mine and concentrator are located approximately 14 km southwest of Fermont Quebec part of the southwest corner of the Labrador Trough iron range. Cliffs' acquisition of Consolidated Thompson in 2011 included a 75% percent ownership in the property.

Operations consist of an open pit truck and shovel mine a concentrator that utilizes single-stage crushing an autogenous grinding mill and gravity separation to produce an iron concentrate. From the site concentrate is transported by rail to a ship loading port in Pointe Noire Quebec.

In operation since 2010 Bloom Lake has an annual rated capacity of 7.2 Mt of iron concentrate.

Presently, the mining operations were shut down at the end of December 2014.

Peppler Lake

Cliffs is now the owner of the Quinto Mining Corporation assets and the Peppler Lake Holdings. The Peppler Holdings contain the Peppler Lake magnetite-hematite taconite Lake Superior-type iron deposit at

Lac Peppler and also a number of other iron ore prospects including Lamêlée Hill, Hobdad, Lac Jean and Faber, that have been identified from historical exploration and mapping programs.

The deposit is located approximately 48 km south of ArcelorMittal Mont-Wright iron mine and 20 km west of the Fire Lake Deposit which is under development by ArcelorMittal. ArcelorMittal completed a drill program on the Peppler Lake Property in 1955/56 and a (non-compliant with NI 43-101) reserve estimate in 1978.

The Property consists of the 55 claims that cover the Peppler Lake Deposit being part of the larger Peppler Holdings. The Peppler Holdings consist of several claim groups including the Property and the Lac Olga, Lac Casse, Lac Jean, Lamêlée Hill, Faber and Hobdad Hill groups. The Peppler Holding properties are located in the Manicougan - Mont-Wright district, approximately 240 km north of Port Cartier and Sept-Ilês and 50 km southwest of Fermont.

The Peppler Lake Property that covers the Peppler Lake Deposit is centred at approximately 52°21'N Latitude and 67°40'W Longitude, National Topographic Map reference 23B/05, Lac Peppler.

23.3 Champion Iron Limited

Champion's Fermont Holdings consist of 12 iron-rich mineral concessions totalling approximately 755 square kilometres in the Fermont Iron Ore District of northeastern Québec located 250 (km) north of the town of Port-Cartier and centered 60 km southwest of the town of Fermont. Currently, Champion holds a 100% direct interest in these projects.

Champion's Consolidated Fire Lake North Project is located in northeastern Québec contiguous to the north of ArcelorMittal's operating Fire Lake Mine, and located 60 km south of Cliff Natural Resources' Bloom Lake mine. The project is located within the Fermont Iron Ore District, a world renowned iron ore mining camp at the Southern end of the Labrador Trough also located within the Grenville Province where it was metamorphosed to a coarser grain size overall. The four current producers in the region account for Canada's total iron ore production which is estimated at 47 Mt of Iron-ore concentrate per year and is expected to increase to 200 Mtpy over the next ten years, based on current expansion plans.

23.4 Labrador Iron Mine Holdings Limited

LIM's mine operations involve the extraction of iron ore by developing open pit mines, starting with the James Mine, which commenced production in June 2011. Commercial production was achieved in April 2012 and LIM completed its first full season of production in November 2012. LIM is now entering its third operating season in April 2013. LIM continues to make progress in advancing the Schefferville Projects with ongoing active programs, including drilling, metallurgical testing, environmental permitting, marketing, engineering, purchasing and construction. At present, LIM has confirmed NI 43-101 compliant indicated resources of 45 Mt at an average grade of 56.5% Fe on the James, Redmond, Denault, Knob Lake and Houston deposits. Currently, the operation is closed.

23.5 Tata Steel Minerals Canada Limited

TSMC DSO Mining Project

The DSO property is comprised of 25 hematite deposits with a resource potential of 120 million tonnes. Detailed exploration and mine planning has been undertaken for 9 deposits. The NI 43-101 compliant resources for 10 deposits total 78 million tonnes.

Ore production of sinter fine products, by TSMC,which is a joint venture between Tata Steel Ltd and New Millennium Iron Corp. is progressing well, with approximately 250,000 tonnes of production to date. The average ore grade is over 63% Fe, and the ore is processed in a transportable dry crushing and screening circuit before shipping (according to a news release dated September 12, 2012). It was estimated that 300,000 t would have been produced in 2012. The installation of the dome structure is completed. The rail line was inaugurated last November 2014 and is connecting to Silver Yards Junction and ultimately to Tshiuetin Rail Transportation Inc. and Quebec North Shore and Labrador Railway Lines. TSMC is also developing plans to increase production to 6 Mtpy in 2015 to maximize the use of the current facilities.

Taconite Exploration Projects

TSMC is also involved with New Millenium Iron Corp. in undertaking a feasibility study of the LabMag and KeMag iron ore deposits which form a part of the 50 km long Millenium Iron Range in Northern Canada.

The LabMag deposit contains 3.5 billion tonnes of proven and probable Fe grading 29.6% Fe, 1.0 billion tonnes of measured and indicated resources grading 29.5% Fe, and 1.2 billion tonnes of inferred resources grading 29.3% Fe. The KeMag deposit contains 2.1 billion tonnes of proven and probable Fe grading 31.3% Fe, 0.3 billion tonnes of measured and indicated resources grading 31.3% Fe, and 1.0 billion tonnes of inferred resources grading 31.2% Fe.

23.6 Cap-Ex Iron Ore

Block 103 covers an area of 7,275 hectares and is located 30 km northwest of Schefferville. The claims are on strike to the Tata Steel/New Millenium magnetite deposits of KeMag and LabMag. The Block 103 host an initial NI 43-101 inferred resource of 7.2 billion tonnes at 29.2 T Fe.

The Redmond Claims cover an area of 16,830 hectares, 10 km south of Schefferville. The claims include strategic ironstone stratigraphy of the Sokoman Formation and the probable southeast extension of the key stratigraphic horizon that contains the high-grade direct shipping ores hosted in the Sokoman Formation. Cap-Ex's northern neighbour, Labrador Iron Mines Holdings Ltd., owns the main Redmond deposits, 3 to 6 kilometres northwest along strike, where approximately 35 Mt of DSO with an average grade of 52-54% Iron (Fe) was mined during the 1970s and early 1980s.

The Lac Connelly project covers approximately 3,720 hectares is located 250 kilometres north of Schefferville, Quebec. Field mapping has indicated two hematite horizons one 2.5 km long (30m wide) and one 1.2 km long (>50 m wide). Average grades of 40% Fe.

The Snelgrove mineral claims cover an area of 8,100 hectares, located 50 km south of Schefferville and is located adjacent to a property which has reported extensive occurrences of hematite grading >60% Fe.

23.7 Lac Otelnuk Iron Project

The Lac Otelnuk Iron Project is located 170 km north of the town of Schefferville, Quebec. The property lies within the Labrador Trough, one of the largest iron ore belts in the world. The belt contains world-class iron deposit that has been continuously mined since 1954.

The project covers an area of 68,449 hectares. The current project activities are working towards the Feasibility stage. The magnetite mineral resources are: 20.64 billion tonnes indicated and 6.84 billion tonnes Inferred. The ownership is comprised of 60% owned by WISCO and 40% owned by Adrianna Resources Inc.

24 Other Relevant Data and Information

No other relevant data and information is available for the Full Moon property.

25 Interpretation and Conclusions

25.1 Mineral Resources

The Rainy Lake property containing the Full Moon iron deposit is a delineation-stage taconite iron exploration project located near the town of Schefferville, Québec. It is underlain by Proterozoic sedimentary rocks of the Labrador Trough, which is known to host world class iron deposits. The property is accessible by air from Schefferville.

The database used by SRK for the preparation of the Mineral Resource Statement contained herein, consists of 124 core boreholes (22,853 metres) drilled by WCSLIM in 2011 and 2012. Drilling is predominantly distributed on section lines spaced at 500 metres and the borehole spacing on each section line is 400 metres, except in the central part of the deposit where line spacing is at 250 metres. As of the date the mineral resource model was constructed, a total of 3,635 sample intervals (0.93 to 7.2 metres in length) were submitted for chemical assaying. Exploration drilling to date has focused on defining the thickness and spatial continuity of the Sokoman Formation. The geological information is sufficiently dense to infer the continuity of the geological units containing the iron mineralization between sampling points and interpret the mineralization's geometry.

The experienced exploration team assembled by WCSLIM used industry best practices to acquire, manage and interpret exploration data. SRK reviewed the data acquired by WCSLIM and is of the opinion that the exploration data is sufficiently reliable to interpret with confidence the boundaries of the iron mineralization and that the assaying data are sufficiently reliable to support evaluation and classification of mineral resources in accordance with generally accepted CIM Estimation of Mineral Resource and Mineral Reserve Best Practices Guidelines.

The mineral resources for the Full Moon iron deposit have been evaluated in a systematic and professional manner. The Mineral Resource Statement reported herein is reported according to CIM Definition Standards for Mineral Resources and Mineral Reserves (November 2010). Open pit mineral resources are reported at a cut-off grade of 20 percent iron and include all Inferred blocks within the conceptual pit shell. The drilling information suggests that the iron mineralization potentially extends beyond the margins of the current geological model.

After review, SRK draws the following conclusions:

- Mineral resources can be increased by investigating iron mineralization located on the periphery of the current geological model;
- Resource classification can be improved with infill drilling along the more widely spaced drilling areas. A spacing of 200 by 250 metres may be required to demonstrate geological and grade continuity and improve the variogram models to support a Measured classification; and
- The mineral resource model should be updated to incorporate the Davis Tube testing results to characterize the nature of the iron mineralization and demonstrate that acceptable iron grade can be achieved by beneficiation.

SRK is not aware of any significant risks and uncertainties that could be expected to affect the reliability or confidence in the early stage exploration information discussed herein.

25.2 Mining Method

The mining method selected for the Project is a conventional open pit, drill and blast, truck and shovel operation with 10 meter high benches.

The 30 year open pit includes 1,283 Mt of Indicated Mineral Resources at a Total Fe grade of 30.8% (Weight Recovery of 36.9%) and 327 Mt of Inferred Mineral Resources at a Total Fe grade of 30.7% (Weight Recovery of 37.7%). In order to access these Mineral Resources, 90 Mt of overburden, 9 Mt of Menihek Shale and 54 Mt of low grade mineralization must be mined. This total waste quantity of 153 Mt results in a stripping ratio of 0.1 to 1.

In order to carry out the 30 year production schedule that was developed for the Project, 20 haul trucks (227 tonne), 3 hydraulic shovels (26.5 m3 bucket), 2 wheel loaders (1,100 kw), 3 production drills as well as a fleet of support and service equipment are required. The peak workforce for the mine reaches 276 employees.

25.3 Processing and Metallurgy

Metallurgical testwork was performed on drill core samples from seven (7) lithology samples from the Rainy Lake Property: JUIF-High & Low, LRC, PGC, URC, LRGC and GC. Metallurgical testwork included: grindability testwork, ore characterization, benchscale beneficiation and pelletizing testwork.

Grindability testwork showed that all tested lithologies were classified as hard or very hard ore.

Ore characterization testwork, including head assays, MLA analysis, Davis Tube testing and Dense Media Separation testing, showed that gravity concentration was not an appropriate concentration route and that a grind size of around 35-45 µm would be necessary to obtain a final concentrate at a 4.5 % silica grade with magnetic concentration. At this grind size, a magnetite recovery of 96-98 % was obtained with DT tests.

Bench-scale and semi-continuous pilot tests confirmed the feasibility to reach the 4.5 % SiO_2 grade with three (3) stages of magnetic separation and regrinding at 95 %-45 µm.

Preliminary reverse flotation tests on the magnetic 4.5 % SiO_2 concentrate permitted concentrates at 1.5 % SiO_2 to be produced. Results showed that optimization and regrinding of the rougher flotation froth is required to increase recoveries.

Beneficiation testwork was conducted on the non-magnetic products from the semi-pilot to evaluate the potential iron recovery of a hematite scavenging plant. It included the following tests: dense media separation, high intensity magnetic separation, selective flocculation and flotation.

Concerning the WHIMS tests, iron recoveries of 76-89 % were obtained with a mass rejection of 43-62 %, showing that WHIMS could be used as a rougher to treat the non-magnetic tails. Selective flocculation and reverse flotation tests were too preliminary to permit a final concentrate to be produced.

Pelletizing tests were conducted at COREM on the composite Wet LIMS concentrate produced by the semi pilot to investigate the suitability of the ore for producing commercial grade pellets. One (1) basket test with three (3) blast furnace pellet chemistries was conducted: two (2) acid pellets and one (1) fluxed pellet. After basket firing, all three (3) pellet samples showed good physical and metallurgical properties.

25.4 Infrastructures

The project infrastructures are mainly located in the mine/concentrator site. Here are the crusher and crushing/screening installations, tailings ponds, and office and maintenance buildings including an accommodation camp with a capacity of 500 workers. The project has also a 91 km long railroad from the mine to Schefferville and an access road that follow the railroad. In the Option 3 and Option 4, there will be infrastructures beside the multi-user terminal, relating to the pellet plant. The train unloading facilities and port handling equipment are not part of the project infrastructures, because these facilities will be built and operated by other. WISCO Century Sunny Lake Iron Mines will pay a usage fee for that service.

25.5 Environmental and Social Aspects

The Project will be subject to Environmental Assessment in accordance with provincial and federal requirements. Following release from the provincial and federal EA processes, the project will require a number of approvals, permits and authorizations prior to initiation and throughout all stages in the life of the project. In addition, WISCO will be required to comply with any other terms and conditions associated with the EA release issued by the provincial and federal regulators.

25.6 Economic Analysis

A preliminary economic analysis has been carried out for the Full Moon Project using a cash flow model. The model is constructed using annual cash flows in constant first-quarter 2015 Canadian dollars and is based on a combined iron concentrate/pellet production of some 20 million tonnes per year over a mine life limited to 30 years. Four production options are considered: HSC only, HSC & HSF pellets, LSC only and LSC & DR pellets.

The selling prices of the mine products are based on a 62% iron concentrate price forecast of US$95 per tonne (CFR China). An exchange rate of US$0.80 per CAD is assumed to convert the revenue estimates into Canadian dollars.

The financial assessment is carried out on a "100% equity" basis, i.e. the debt and equity sources of capital funds are ignored. No provision is made for the effects of inflation. Results are given before and after taxation. Current Canadian tax regulations are applied to assess the corporate tax liabilities while the recently proposed regulations in Quebec (Bill 55, December 2013) are applied to assess the mining tax liabilities.

The summary of the economic analysis is shown in Table 25.1.

Table 25.1 – Summary of Financial Results

Description	Units	Option 1 (Preferred)	Option 2	Option 3	Option 4
Total Revenue FOB Sept-Îles (LOM)	M$	72,384.3	70,328.5	91,316.2	86,973.1
Total Operating Costs (LOM)	M$	29,759.3	30,424.2	35,436.6	35,869.5
Total Pre-production Capital Costs	M$	7,207.3	7,385.5	8,886.1	9,064.3
Total Sustaining Capital Costs (LOM)	M$	658.0	658.0	658.0	658.0
Initial Working Capital	M$	369.9	378.6	439.5	445.4
Mine Closure Costs	M$	178.2	178.2	178.2	178.2
Salvage Value	M$	358.0	366.9	441.9	450.8
BEFORE TAX					
Total Cash Flow	M$	34,939.5	32,049.5	46,599.2	41,654.0
Payback Period	years	5.7	6.3	5.4	6.0
NPV @ 8%	M$	5,771.0	4,806.7	8,196.0	6,626.3
NPV @ 6%	M$	9,233.6	8,026.4	12,772.2	10,779.7
NPV @ 10%	M$	3,395.2	2,604.2	5,048.3	3,779.1
IRR	**%**	**15.2**	**13.9**	**16.2**	**14.6**
AFTER TAX					
Total Tax Payments (LOM)	M$	12,360.0	11,170.1	16,321.7	14,323.0
Total Cash Flow	M$	22,579.5	20,879.4	30,277.5	27,330.9
Payback Period	years	6.3	6.8	5.9	6.5
NPV @ 8%	M$	2,965.3	2,335.8	4,418.9	3,409.1
NPV @ 6%	M$	5,326.2	4,560.4	7,539.7	6,285.5
NPV @ 10%	M$	1,334.1	802.8	2,258.5	1,423.5
IRR	**%**	**12.4**	**11.4**	**13.2**	**12.0**

Both the project's net present value and internal rate of return are more sensitive to changes in operating costs than to changes in capital costs. As expected however, the project's financial performance is most sensitive to changes in selling price. See Section 22.2 for a description of the key economic operating and technical assumptions used in preparing the economic analysis.

The economic analysis contained in this report is preliminary in nature. It incorporates inferred mineral resources that are considered too geologically speculative to have the economic, considerations applied to them that would enable them to be categorized as mineral reserves. It should not be considered a prefeasibility or feasibility study. There can be no certainty that the estimates contained in this report will be realized. In addition mineral resources that are not mineral reserves do not have demonstrated economic viability.

25.7 Risks

There are some risks inherent to a mining project such as:

- Environmental Impact Assessment timing;
- Discussions with the different communities;
- Geotechnical Assessment;

Other, generally more important risks that could delay the construction and production of the project are:

- The assumption that a multi-user facility in the Port of Sept-Iles will be built. If the facility is not available, a business contract will have to be signed between QNS&L/IOC and WISCO Century Sunny Lake Iron Mines for the usage of their facility;
- The assumption, that they are available land to build the pellet plant beside the multi-user terminal;
- The assumption that the electric power line and thus electric power is available on time for the construction for the project and subsequent concentrate production. Contracts will have to be in place between Hydro-Québec and WCSLIM.
- And naturally the most important risk is the price of iron which will always be the key for a project start-up.

25.8 Conclusion

WISCO Century Sunny Lake Iron Mines has four options to develop the project. All options have shown that the project has a potential of economic viability.

For Option 1, that has the lowest estimated Capital Cost of the four options (M$7,207.3) and an average Operating Cost of $49.85/ tonne of concentrate, the Economic Analysis shows, at a selling price of $121.25/t of High Silica Concentrate FOB Sept-Îles, an IRR of 15.2% (Before Tax) and IRR of 12.4% (After Tax).

26 Recommendations

26.1 Geology

The geological setting and character of the taconite iron mineralization delineated to date on the Rainy Lake property are of sufficient merit to justify additional exploration and pre-development expenditures. The block model constructed by SRK is sufficiently reliable to support mine planning and allow evaluation of the economic viability of a mining project. The Full Moon taconite iron deposit is a very large deposit and a significant proportion of the mineral resources are classified in the Indicated category. On this basis, the work program recommended by SRK includes:

- Infill drilling along the more widely spaced drilling areas to an approximate drilling spacing of 200 by 250 metres spacing (70 to 90 core boreholes). Infill drilling should initially focus on the areas presenting the most favourable configuration for a viable mining operation;
- Preliminary rock geotechnical investigations (10 to 20 boreholes); and
- Additional geology and mineral resource modelling after reception of all Davis Tube testing results.

26.2 Mining

- The Mineral Resource Estimate should be updated to consider the results of the Davis Tube and Satmag tests that were completed on the 2012 drillhole assays.
- A geotechnical pit slope analysis should be done to determine the appropriate pit wall configuration.
- A geotechnical analysis should be done to confirm the stability of the dump and stockpile designs.
- Geochemical testwork should be done on the overburden and waste rock to evaluate if there is a potential for this material to be a generator of acid rock drainage.
- A hydrogeological study should be done to estimate the amount of groundwater that is expected to be encountered during the mining operation.

26.3 Metallurgy

The benchscale testwork performed during this study led to the definition of the Magnetite Plant flowsheet producing a concentrate at 4.5 % SiO_2. To bring the project to the Pre-Feasibility Study level, complementary testwork is required to firm up the Hematite Plant Scavenging flowsheet and the flowsheet sections producing a LSC from the HSC:

- Benchscale testwork including MLA and flotation tests will confirm the LSC circuit flowsheet;
- Benchscale testwork including MLA to confirm regrind size, WHIMS tests, flotation and selective flocculation will be necessary to better define the hematite recovery circuit flowsheet;
- Pelletizing tests will be realised to qualify the feasibility to produce pellets using magnetite hematite HSC and LSC.
- Samples should be collected for the Feasibility testwork:
    - Samples to evaluate the process variability (Grindability and magnetite & hematite plant beneficiation confirmation testwork);
    - A large bulk sample representative of the ore body for pilot plant testwork.

26.4 Environmental and Social Aspects

With respect to environmental considerations WSP recommends to:

- Carry out the Environmental Assessment as well as any related environmental baseline studies;
- Engage discussions with local community and include additional stakeholders to identify key areas and subjects to be addressed during the advancement of the exploration project and through the future EA phase of the Project;
- Conduct a geochemical testing to determine Acid Generating/Non-Acid Generating Potential of mineralized rock waste rock and tailings as well at the respective potential for metal leaching/non leaching.

26.5 Infrastructures

With respect to infrastructures CIMA+ recommends to:

- To initiate discussion with power electric company (Hydro-Québec) to confirm power availability;
- To initiate discussion with existing railroad operators;
- To initiate discussion with multi-user port operator.

26.6 Recommended Work Program and Estimated Costs

Table 26.1 shows the recommended work program and estimated costs.

Table 26.1 – Recommended Work Program

Description	Cost
Pre-Feasibility Study	$1,500,000
Geotechnical Drilling Oriented Diamond drilling (all inclusive)	$2,250,000
Hydrogeology study for the open pit	$750,000
Delineation Drilling (infill and step out) Diamond drilling (all inclusive)	$6,000,000
Resources Update	$150,000
Metallurgical Testwork	$ 350,000
Total	**$11,000,000**

27 References

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