Century Iron Mines Corporation / Augyva Mining Resources Inc.

NI 43-101 TECHNICAL REPORT
PRELIMINARY ECONOMIC ASSESSMENT OF THE DUNCAN LAKE IRON PROPERTY
JAMES BAY, QUEBEC – CANADA
NTS Sheets 33F05 and 33F12

Effective Date: March 22, 2013
Issue Date: May 6, 2013

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1.0 SUMMARY

1.1 Introduction

Met-Chem Canada Inc. ("Met-Chem") was retained in February 2012 by Century Iron Mines Corporation ("Century") to prepare an independent technical report for a Preliminary Economic Assessment ("PEA") of the Duncan Lake Iron Project ("DLIP") in Quebec.

This PEA is based on the updated Mineral Resources of DLIP prepared by Met-Chem in October 2012 and filed under title: "NI-43-101Technical report on the mineral resources of the Duncan Lake iron project, James Bay area".

All the information on geology and resource estimation are taken from this report and there is no new technical information on those subjects.

This report documents the results of the PEA study and constitutes a Technical Report under the guidelines of NI 43-101. The classification of the Mineral Resources used in the PEA is compliant with the CIM Definitions, in accordance with NI 43-101.

This technical report was issued jointly to Century and Augyva Mining Resources Inc. ("Augyva").

1.2 Property Description and Ownership

The DLIP is located approximately 570 km north of Matagami, Québec, within the Municipality of James Bay, along Highway 109. The property is 40 km south of Radisson and 950 km to the NW of Montreal.

The DLIP consists of 534 contiguous claims covering 25,605 hectares. All the claims are registered under Augyva and Century, and all were in good standing at the time of writing this report.

A tract of land controlled by Hydro Québec truncates many claims along the center of most of the long axis of the property and 44 claims carry encumbrances related to an electrode grounding system and/or a power line corridor.

Although the DLIP lies in the northern part of the Province of Quebec, it is out of permafrost range and several Canadian mines are operated under harsher climatic conditions than the ones prevailing in the Radisson area.

On May 20, 2008, Century entered into an option and joint venture agreement with Augyva in respect of the DLIP (the "Duncan Lake Joint Venture Agreement"). In 2010, Century earned a 51% interest in the DLIP under the Duncan Lake Joint Venture Agreement after funding a commitment of $6.0 million. Currently, Century has earned a cumulative 65% interest in the DLIP, having funded a further $14.0 million on the DLIP, under the Duncan Lake Joint Venture Agreement.

Century has entered into a Joint Venture Agreement with WISCO International Resources Development & Investment Limited ("WISCO") pursuant to which WISCO may earn a 40% joint venture interest in Century's interest in the DLIP in exchange for an aggregate investment of $40 million.

Century, with Head Offices in Toronto, Ontario, is partnering with state-owned Chinese companies, WISCO and Minmetals Exploration & Development (Luxembourg) Limited S.à r.l. Augyva's Head Offices are located in Montreal, Quebec.

In 2005, Augyva acquired the DLIP from Virginia Mines Inc. ("Virginia"), to which a perpetual production royalty of $0.40 per ton of iron concentrate is payable. Augyva retained a buyback right to purchase 50% ($0.20 per ton of iron concentrate) of the royalty for a payment of $4 million, in addition to an option of buying back a further 20% royalty ($0.08 per ton of iron concentrate) by paying $4 million. A 2% net smelter return ("NSR") royalty on any metal other than iron is also payable and Augyva also has the right to purchase 50% of this NSR (1% NSR) for $ 5 million.

1.3 Geology and Mineralization

The DLIP lies within the western part of the La Grande Sub-Province of the structural Superior Province. The La Grande Sub-Province is characterized by an Archean tonalitic basement (Langelier Complex) unconformably overlain by the volcano-sedimentary Guyer and Yasinski Groups composed of iron formation, wacke, paragneiss, basalt to dacite and pyroclastic units. The alluvial or fluvial sediments of the Ekomiak Formation partly lie on the Yasinski Group. The sediments of the Sakami Formation were deposited in NE-trending sedimentary basins. All these rocks are intruded by several plutons (Duncan Lake and Radisson plutons) and mafic to ultramafic intrusions and dikes.

The Banded Iron Formation ("BIF") at Duncan Lake shares features characteristics of both the Superior Lake and Algoma types of iron formations. Regional metamorphism ranges from greenschist to amphibolite facies. The supracrustal rocks have been deformed by at least two structural events, forming a subvertical, N-S and a steeply south-dipping, E-NE trending schistosity, as well as folds and shears.

The DLIP is underlain by two parallel N-NE BIF units traced across the entire property by their magnetic signature and by drilling. Six main deposits have been identified along these two bands, with Deposits 1 to 4 located on the NW band and Deposits 5 and 6 along SE band.

Deposits 1 and 2 are part of one continuous N-NE trending band traceable over about 17 km and appear to join Deposit 3. They are separated by about 2 km from Deposit 5 on the SE. Deposit 3 is characterized by two main BIF units arranged as a large-scale, tight synform and antiform system. The NW branch of Deposit 3 is connected to Deposit 4 by one NE magnetic anomaly. Deposit 6 seems to be disconnected from the other deposits.

Stacking of BIF units by thrust faults is interpreted in most deposits. Mafic volcanic rocks dominate in the area of known BIF occurrence, but felsic rocks and possible basement granite prevail in the Deposit 6 sector.

Iron mineralization within the DLIP property consists of alternating bands of quartz and magnetite, with only minor amounts of hematite. The DLIP deposits are also associated with silicate and sulphide facies iron formations. On average, the iron mineralization at DLIP contains 15 to 35% total Fe and very low levels of deleterious elements, except for elevated average sulphur content that probably originates from widespread disseminated pyrite.

1.4 Exploration, Development, Operations

The first systematic exploration effort targeting the Duncan Lake iron mineralization since the discovery in 1949 consisted of an airborne magnetometer survey and 8 diamond drill holes completed in 1956. In 1973, 22 holes for 4,188 m were drilled into deposits 3, 4 and 6 and 10,460.25 m were drilled in 2008-2009 and 44,006.65 m in 2011-2012 into all six deposits. The Mineral Resources that served as the basis for this PEA were estimated after the drill program of 2011-2012 but disregarded the results from the holes drilled in 1973. Several ground magnetic surveys have been completed recently, since the method is an efficient tool to detect the BIF units.

1.5 Sample Preparation, Analysis and Security

1.5.1 2008-2009 Drill Program

The core was split using a hydraulic splitter and a diamond blade saw at nominal lengths of 3 m for the first 22 holes and 5 m for the rest. The samples were delivered by Augyva and Century to ALS-Chemex Laboratory, in Val-d'Or for preparation and analysis, thus preserving the chain of custody.

At the laboratory, the samples were crushed to 6 mm with a jaw crusher and then reduced to 90% passing 10 mesh. Finally, a 30-gram sub-sample is pulverized to 90% passing 200 mesh in a ring and puck pulverizer. The samples were analysed for major oxides via XRF- Lithium Borate fusion and for sulphur in a Leco furnace.

218 samples from Deposits 1 to 4 selected for metallurgical testing were re-analysed by COREM, an independent laboratory located in Québec City, for metallurgical testing and served as a check by a secondary laboratory. From these 218 samples, 144 Davis Tube concentrates and tails were analysed, in addition to the head analyses.

The control samples added by the geologists to the samples batches consisted of blanks, standards and duplicate samples representing 7.7% of the total. In addition, an equivalent of 9.7% of the total number of samples was sent to a second laboratory.

Both ALS-Chemex and COREM are ISO certified and used similar QA-QC protocols and procedures, and processed these samples with the same preparation and analytical methods.

1.5.2 2011-2012 Drill Program

Core splitting was done by IOS Services Géoscientifiques Inc. ("IOS") using a hydraulic splitter at the beginning of the program, and subsequently a diamond blade saw. Nominal samples length was 3 m, with variations between 1.5 m and 4.5 m when necessary to honour the main lithological contacts.

Sample preparation, except for the six holes also drilled to provide material for metallurgical tests, was contracted to IOS in Chicoutimi. The samples were crushed by IOS to less than 10 mm in a jaw crusher, and to less than 2 mm in a roll mill. A sub-sample of 200 to 300 g was extracted and sent to ALS Chemex in Val-d'Or, Quebec, for analysis.

All the samples were submitted to XRF-Lithium Borate fusion for analysis of the major oxides. Selected samples had determination of sulphur by Leco furnace, Loss on Ignition (LOI %), multi-element ICP-OES Analysis and Davis Tube tests. A batch of 100 samples from Deposits 3, 4 and 6 were later analysed for sulphur.

A total of 843 samples were submitted to Davis Tube tests of which 414 samples were tested at SGS Lakefield, Canada ("SGS"), 285 at IOS, in addition to the 144 tests performed at COREM in 2009.

IOS inserted duplicate samples, as well as blank and certified standard materials into the sample stream to monitor the laboratory performance. The percentage of control samples amounted to about 15%.

The specific gravity was determined by IOS on a total of 4,967 barren and mineralized samples selected from 93 different holes. The water displacement method was used as a primary method and all the samples were also processed by the pycnometer technique. A total of 394 samples from Deposits 3, 4 and 6 originally analysed by Activation Laboratories Ltd. ("Actlabs"), Ancaster, Ontario, were re-analysed by SGS used as a second laboratory.

IOS preserved the chain of custody between the field, the IOS facilities in Chicoutimi and the laboratory in Val-d'Or.

1.6 Data Verification

Met-Chem's QP Mr. Yves Buro visited the DLIP on August 9 to 12, 2011. The visit included a field trip and examination of the core from selected holes with the IOS' geologists.

Met-Chem selected 50 sample rejects covering a fair range of iron contents and depths in Deposits 3, 4 and 6 to be re-analysed and to serve as independent check samples.

The results from the drill program were transmitted by IOS to Met-Chem in dedicated logging software Geotic format and in Excel spreadsheets. IOS validated the data before sending them and Met-Chem did additional verifications in the master database and reviewed the results obtained from the control samples inserted by IOS into the sample stream.

1.7 Mineral Processing and Metallurgical Testing

In 2009, COREM laboratory performed Davis tube testing on samples from Deposit 3 and ground at 200 mesh giving acceptable results.

In 2011, the material for metallurgical testwork at SGS is from two holes totaling 2,349 m of HQ core that were drilled into each of the Deposits 3, 4 and 6.

At SGS, the samples were subjected to whole-rock analysis and full ICP-scan. The JK drop-weight, Bond Low-energy impact and Bond abrasion tests were performed on three composite samples. Sag Mill Comminution was conducted on seven different lithologies as well as Bond rod mill and Bond ball mill grindability tests. Coarse cobbing was evaluated with a dry magnetic drum to assess capability.

More than 400 samples from Deposits 1, 3, 4 and 6 were ground at 325 mesh at SGS and were submitted to Davis tube testing. Results showed that the average weight recovery is more than 25%.

1.8 Mineral Resource Estimates (2012)

The resources estimation completed on Deposits 3, 4 and 6 included the 2011-2012 drill data, whereas the resources for Deposits 1 and 2 were simply updated from the 2008-2009 data.

All the samples were submitted to XRF-Lithium Borate fusion for analysis of the major oxides, and selected samples had determination of sulphur and Loss on Ignition, multi-element ICP-OES analysis, Davis Tube tests and density determination.

A thorough QA-QC system using QC samples and secondary laboratories ensured proper monitoring of the laboratories performance. Several passes of verification ensured the reliability of all the data populating the master database.

Estimation methodology was based on interpreting vertical cross-sections which were meshed into 3D solids and used to constrain inverse distance squared estimates within 6 separate Block models. Solids boundaries were defined by a combination of lithology and Fe grade. Regular 20 m x 10 m x 5 m Block sizes were used for each of the Block models. Search ellipses reflecting unique dips and strikes to the various fold limbs were used to constrain the interpolation. Assay sample lengths were composited to a nominal 3-m length for grade interpolation. Total head Fe, Davis Tube Weight Recovery ("DTWR"), Fe% and SiO_2% in Davis Tube concentrates were modeled. A global density factor of 3.2 g/cm^3 based on 3,107 determinations was assigned to the block models.

The Mineral Resource estimate for Duncan Lake used 9,178 assays collected from 54,467 m of drilling in 177 drill holes. The estimate also rested on a total of 843 Davis Tube samples.

Mineral Resources were classified based on search ellipse ranges and minimum number of informing composites. A Measured Resource classification was assigned to blocks interpolated by a minimum of 12 composites and maximum search ellipse range of 300 m along the major axis, 150 m along the semi-major axis and 20 m along the minor axis. Indicated category was assigned to blocks interpolated by a minimum of 6 composites and maximum search ellipse range of 300 m along the major axis, 150 m along the semi-major axis and 20 m along the minor axis. Inferred Resource was assigned to blocks interpolated by a minimum of 3 composites and maximum search ellipse range of 450 m along the major axis, 225 m along the semi-major axis and 30 m along the minor axis.

The Mineral Resources calculated by Met-Chem in August 2012 are reported to a cut-off of 16% Fe and are not constrained by a pit shell. A list of Mineral Resources is provided in Table 1.1 below.

Table 1.1 – Summary of the Mineral Resource (Cut-Off of 16% Head Fe; 2012)

Mineral Resource Category	Metric Tonnes (Million)	Fe (%)	DTWR (%)	DT Fe (%)	DT SiO_2 (%)
Measured	405.6	23.92	26.78	67.26	5.25
Indicated	644.9	24.73	28.09	66.87	5.60
Measured + Indicated	**1,050.5**	**24.42**	**27.58**	**67.02**	**5.46**
Inferred	563.1	24.69	27.97	66.46	6.03

The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. However, Met-Chem is not aware of any known environmental, permitting, legal, title, taxation, socio-political, marketing or other issues that would materially affect the Mineral Resources. The quantity and grade of reported Inferred Mineral Resources in this estimate are uncertain in nature and there has been insufficient exploration to define the Inferred Mineral Resources as Indicated or Measured Mineral Resources and it is uncertain if further exploration will result in upgrading them to Indicated or Measured Mineral Resource categories.

The Mineral Resources are reported in accordance with Canadian Securities Administrators NI 43-101 and have been classified in accordance with standards as defined by the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM"), "CIM Definition Standards for Mineral Resources and Mineral Reserves". Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

No previous production or Mineral Reserves have been reported for the DLIP, or on adjacent properties.

1.9 Mining Methods

Met-Chem evaluated the potential of the Duncan Lake Iron Property ("DLIP"), targeting a production rate of 12,000,000 tonnes of iron pellets (acid pellets) per year.

To maximize the potential economics of the PEA, Met-Chem selected Deposits 3 and 4 as the basis for the PEA. These deposits have the largest tonnage and best minerology of the 6 DLIP deposits and can supply the concentrator and pellet plant for over 20 years of full production.

The mining method selected for the Project is a conventional open pit drill and blast operation with 400 st haul trucks and 40 m^3 hydraulic excavators. Pre-production stripping of waste and overburden material will be done by a contractor.

Open pit optimization was done on both Deposit 3 and 4 to derive the pit shell with the highest Project Net Present Value ("NPV"). A series of pit shells were generated using the Lerch Grossman algorithm in the Economic Planner optimizer of MineSight®. These shells were generated by varying the selling price.

The optimization was carried out during the initial stage of the PEA study using the cost, sales price and pit and plant operating parameters presented in Table 1.2 below. These parameters are preliminary estimates for developing the economic pit and should not be confused with the operating costs subsequently developed for the PEA and provided elsewhere in this report. A conservative pellet sales price of USD 140/t was used in the pit optimization, a value lower than the sales price used in the PEA economic evaluation. The pit optimization was re-evaluated after a preliminary mine plan was completed and the cost, sales price and pit and plant operating parameters were better defined. The results of the second pit optimization using the updated operating costs and sales price confirmed the original optimization results. Inferred Mineral Resources were used in the optimization and mine plan of the PEA as allowed in the NI 43-101 guidelines for such a study.

Table 1.2 – Pit Optimization Parameters

Item	Value	Units
Mining Cost – Mineralization	2.20	$/t (mined)
Mining Cost – Waste Rock	2.40	$/t (mined)
Mining Cost – Overburden	1.75	$/t (mined)
Processing and Pipeline Cost	18.00	$/t (pellet)
Pelletizing Cost	12.00	$/t (pellet)
Shipping Cost	37.00	$/t (pellet)
General, Admin & Infrastructure Cost	5.70	$/t (pellet)
Sales Price	140	USD/t (pellet)
In-Situ Dry Density – Overburden	2.00	t/m^3
In-Situ Dry Density – Mineralization	3.20	t/m^3
In-Situ Dry Density – Waste Rock	2.90	t/m^3
Overall Pit Slope	52	Deg

* The cost parameters are preliminary estimates for developing the economic pit and should not be confused with the operating costs subsequently developed for the PEA Study and given elsewhere in this report.

The economic pit limits derived from the pit optimization were used as a guideline for the detailed pit design. 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 initially selected equipment. The ramps and haul roads were designed with an overall width of 36 m (3 times the overall width of a 400 st haul truck, i.e. 9.8 m plus berms and ditches).

The pit designs and mine plan of combined production from Deposits 3 and 4 identified a total of 660 Mt of Measured and Indicated Resources and 157 Mt of Inferred Resources, (fully diluted and recovered) with a combined stripping ratio of 1.8:1 for 20 years of production. During the first five years of production, overburden and waste stripping was kept at a low stripping ratio of 1.36:1 and increased gradually over the remaining years.

The total mine operation workforce for the Project ranges from 251 employees in Year 1 to a maximum of 419 from Years 11 to 20. This workforce is comprised of staff as well as hourly employees.

1.10 Recovery Methods

Test work program was held at SGS Lakefield and the summarized flow sheet is therefore presented in this report. Run of mine ("ROM") material will be crushed using gyratory crushers before being conveyed to three concentrator process lines. Met-Chem has included, for each process line, the use of standard SAG mill with screening to produce a P_{100} of 3.36 mm. Cobber magnetic separators are part of the SAG mill circuit to reject a portion of the liberated non-magnetic gangue. Then, standard ball mills are used in closed-circuit with cyclones to produce a P_{85} of 75 microns. The magnetite will then be recovered using multiple stages of Low Intensity Magnetic Separators ("LIMS").

The iron concentrate is thickened to 65% solids and pumped through a pipeline to the pellet plant which will process the concentrate in two 6 Mtpy pellet production lines. Each pelletizing line consists of vacuum disc filters, mixing units for bentonite and concentrate, balling units to produce green pellets and induration machine to produce the final pellets grading 66.3 % Fe and 5.1% SiO_2.

The pellet storage area is designed to store up to eight months of pellet production. The project will thus be able to support shipping 12 months of pellet production during the 4 month ice-free shipping season. The storage area will be close to the pellet plant and the dedicated Duncan Lake port on James Bay.

1.11 Project Infrastructure

The major project infrastructure includes the dedicated port facilities at Stromness Island, near Chisasibi, the tailings dykes construction, the concentrate pipeline from the concentrator to the pellet plant, the site roads, maintenance facilities, permanent camps at Radisson and near the pellet plant, administration buildings, warehouses, emergency vehicle and first aid buildings, assay laboratories, the final product storage yard and the fuel storage areas.

1.12 Market Study

The QP has relied on long term iron ore pricing and market assumptions prepared by independent consulting firm Raw Materials & Ironmaking of Bethlehem Pennsylvania, who prepared an independent marketing and sales price analysis of the Duncan Lake Iron pellets. The report, titled "Century Iron Mines Ore Marketing Study", was prepared by Dr. Joseph J. Poveromo, a world renowned iron and steel marketing specialist and president of Raw Materials & Ironmaking. The report is dated February 25, 2013. The QP has reviewed this report and the results support the assumptions in this technical report.

Met-Chem has summarized the findings of Dr. Poveromo below:

The DLIP Project will start with the upgrading of a lower grade magnetite mineralization to produce a fine sized concentrate at 67.6% Fe and 5.0% SiO_2. This concentrate will be conveyed by slurry pipeline to a pellet plant located at a James Bay shipping point. The concentrate will be too fine sized to effectively transport it by vessel so we will consider blast furnace pellets as the only product. In any event the Atlantic Basin pellet feed market will be in oversupply, with the demand focused in China, so this absence of a pellet feed product will not be detrimental.

The pellet plant will produce a blast furnace acid pellet with 66.3% Fe and 5.1% SiO_2 with a very low Al_2O_3 level and low levels of other impurities and residual elements. Such a pellet will be well suited as a complement to high sinter burdens in steel plants in Asia (specifically China) and Europe. The very low (0.30 %) Al_2O_3 level will advantage DLIP for Asian ironmaking operations which have issues with high Al_2O_3 levels generally encountered with Australian iron ore. In Europe, the Duncan Lake acid pellet quality will be comparable to other North American produced pellets, well accepted in European blast furnaces.

The near term blast furnace pellet market globally suggests a potential oversupply, so the off take agreements by WISCO and MinMetals, along with a potential contract with one or more European customers, will be essential to guarantee the revenue stream for this project. On a longer term basis, the reduction in lump ore supply due to quality issues in Australia and virtual elimination of lump ore exports from India and Brazil will increase the demand for pellets.

The long term pellet price will follow from the long term fines price plus a pellet premium. A long term pellet premium of USD 35/t will be assumed; it is supported both by market evidence and the required price differential to justify pellet plant investment.

The consensus opinion among iron ore experts is that the so called long term equilibrium price of iron ore fines (62 % Fe, CFR China) will be driven by the costs of the higher cost Chinese production as this production would ultimately shut down if iron ore prices stay well below this level for a sustained time period. This high cost level is in the vicinity of

USD 120/t to 130/t so the choice of USD 125/t seems reasonable. However there will be periods of higher and lower prices.

The long term fines price, under a worst case scenario, could fall below USD 100.00/t with a "perfect storm" of many new merchant projects, much steel company equity iron ore investment, new steel plants in iron ore rich areas and a levelling off of global steel demand.

However, long term higher prices of USD 125/t, driven both by the costs of the higher cost producers and new iron ore projects, are also driven by:

- Grade depletion globally means that more ore is needed for the required Fe units;
- Shortages of equipment, supplies, labor and skills will not only delay new projects but impact on availability at existing operations; the tire shortage of several years ago impacted existing mines;
- Misguided government and steel industry promoted policies in restrictions of both iron ore exports and mining itself will cause India's iron ore industry to grossly underperform;
- Natural disasters, floods, typhoons, etc., could impact on both mining operations and shipping;
- Political unrest could affect some new mines being built in more unstable regions such as West Africa.

Aside from the real reasons for supply reductions, a major "contrived" reason for reduced supply could be oligopic behavior by the "Big Three" VALE, BHPB and Rio Tinto, in slowing down expansions or simply reducing production at existing less favored sites when ore prices drop too low, as a means of inducing shortages that will propel spot prices upward.

1.13 Environment

No hydrometric stations have yet been established but initial data have been collected in three gauging stations in 2011 and 2012. One limnimeter in Esprit Lake and one in Desaulnier Lake have been collecting data since 2011. Groundwater samples were collected in 2011 and 2012 in the deposit area. Studies of the ecosystem and vegetation within the DLIP were also conducted in 2011. No soil contamination by oil or fuel was observed during a site visit by Le Groupe Desfor in August 2012.

The DLIP is subject to the Québec Environmental Assessment Act and the Canadian Environmental Assessment Act. The former requires that large projects undergo an environmental assessment, including provisions for active participation of the First Nations, while the latter applies when a federal agency is required to make a decision on whether to issue authorizations that may include matters related to fish habitat or navigable waters.

Met-Chem is not aware of any agreement under which aboriginal communities may hold title or historical agreement to the mineral land for the DLIP. Met-Chem is not aware of any environmental liabilities to which the DLIP is subject, and none is mentioned in the GESTIM management system for the DLIP. Century made sure all exploration programs on the DLIP have and will be conducted in an environmentally friendly manner.

1.14 Capital and Operating Costs

All dollars are Canadian dollars unless noted differently.

The total life-of-mine capital cost for the 12 Mtpy pellet production rate is estimated at \$4,546 M of which \$3,881 M is initial capital and \$665 M is sustaining capital as summarized in Table 1.3 below.

Table 1.3 – Total Capital Costs

Item Description	Total Rounded ($ Millions)
Initial Capital	
Pre-Production Direct Capital Cost	2,967
Pre-Production Indirect Capital Cost	363
Contingency	503
Total Pre-Production cost	3,833
Ramp-Up Capital	48
Total Initial Capital	**3,881**
LOM Sustaining Capital	**665**
LOM Total	**4,546**

Initial capital of \$3,881 M includes \$3,833 M for pre-production period and \$48 M for mining support and service equipment as well as mining systems to be procured in the first year ramp-up period.

The pre-production indirect capital cost is estimated at \$363 M while the contingency is estimated at \$503 M.

The total average life-of-mine operating costs were estimated at \$59.17 per tonne of pellet produced as shown on Table 1.4. The mine production cost is estimated at \$24.02 per tonne of pellet. The concentration and slurry transportation cost is estimated at \$16.86 per tonne of pellet. The Pellet production and handling is estimate at \$11.45 per tonne of pellet. The G & A and site services cost is estimated at \$4.84 per tonne of pellet. The ship loading cost is estimated at \$2.00 per tonne of pellet.

Table 1.4 – Total Operating Costs (Average life-of-mine)

Operating Costs	$/tonne of pellet
Mine production	24.02
Concentration and slurry transportation	16.86
Pellet production and handling	11.45
G&A and site services	4.84
Ship loading	2.00
Total	**59.17**

Table 1.5 – Total Operating Costs (Average first 5 years)

Operating Costs	$/tonne of pellet
Mine production	18.09
Concentration and slurry transportation	17.27
Pellet production and handling	11.45
G&A and site services	4.84
Ship loading	2.00
Total	**53.65**

Table 1.5 presents the average operating costs for the first 5 years of operation. The operating costs for the first 5 years are lower due to lower stripping ratio and slightly lower weight recovery.

The selected shipping scenario assumes the use of Capesize (185,000 dwt) and Suezmax (240,000 dwt) ships during the 4 month ice-free simmer season of James Bay. Costs are estimated at USD 35/t pellet for shipment to Quindao for 70% of the pellet production. The other 30% of the production would be shipped to Rotterdam at an estimated cost of USD 15/t. The average shipping cost taking into consideration the 70% to China and 30% to Europe averages USD 29/t. This cost is not used in the DLIP operating costs but is used for estimating FOB James Bay selling prices in the economic evaluation.

1.15 Economic Analysis

The pre-tax economic analysis results are summarized as:

- Net Present Value ("NPV") of $4.1 billion at an 8% discount;
- Internal Rate of Return ("IRR") of 20.1 %;
- Payback period of 4.2 years;
- Mine life of 20 years at 12 Mtpy of pellet production;
- Cost estimate accuracy of ± 35%.

The post-tax economic analysis results are summarized as:

- Net Present Value of $2.2 billion at an 8% discount;
- Internal Rate of Return ("IRR") of 15.9 %;
- Payback period of 4.8 years;
- Mine life of 20 years at 12 Mtpy of pellet production;
- Cost estimate accuracy of ± 35%.

The economic assumptions used are summarized as:

- USD 125 per tonne of 62% iron concentrate, CFR China (basis);
- USD 134 per tonne for 66.3% Fe grade of Duncan Lake Pellet;
- Iron Pellet Premium of USD 35 per tonne;
- Transport cost to China USD 35 per tonne;
- Transport cost to Europe USD 15 per tonne;
- Ship loading costs USD 2 per tonne;
- Market split LOM tonnage of pellets shipped to China: Europe assumed at 70:30;
- Weighted average CFR price of USD 169 per tonne of Duncan Lake pellet;
- Life of Mine for financial analysis 20 years;
- Exchange rate at par for 2013 to 2017 and 0.95 USD/CAD for 2018 and beyond;
- Fuel prices of $1.05 per liter of diesel and $0.62 per liter of bunker C (pellet plant);
- Electricity rate of $0.09 per kWh for mine and concentrator (primary transformation) and $0.045 per kWh for secondary transformation and pellet plant;
- Mine mobile production and auxiliary equipment are leased;
- Camp facilities are leased.

1.16 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 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.

The results of the economic analysis are forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. See Section 22.0.

1.17 Conclusions

The DLIP is planned as a 20 year operation producing 12 Mtpy of acid pellets, with its mine and concentrator situated close to the town of Radisson in northern Québec, and its pellet plant and port located near the town of Chisasibi on the shores of James Bay, some

135 km away from the mine. The port would ship the pellets on ocean-going vessels during the 4 month ice-free shipping period. The project is also in very close proximity to Hydro Quebec's La Grande hydroelectric complex.

The drilling program of 2011-2012 and the data from the 2008-2009 holes allowed defining ~75% of the Mineral Resources in Deposits 3, 4 and 6 in the Measured and Indicated categories. The two drill programs have been successful in providing sufficient data on all six DLIP Deposits to produce in 2012, new or updated Mineral Resource estimates totalling 1,051 Mt of Measured and Indicated resources grading 24.42% Fe and 563 Mt of Inferred resources grading 24.69% Fe. The DLIP deposits that were considered for the PEA (Deposits 3 and 4) contain an estimated total Measured and Indicated Resources of 797 Mt at 24.44% Fe, and 277 Mt of Inferred Resources grading 25.07% Fe.

The present mineral resource estimation is compliant with the CIM Definitions, in accordance with NI 43-101 and Met-Chem believes to be a sound foundation for the PEA.

In-Pit resources used for the mine plan and the economic evaluation were estimated by designing a pit around an optimal economic pit defined by the Lerch Grossman method. An estimated 660 Mt of Measured and Indicated resources and 157 Mt of Inferred resources would produce 12 Mtpy of pellets over 20 years with an average stripping ratio of 1.8:1.

The PEA's economic evaluations shows that, using an 8% discount rate and an initial investment of \$ 3.8 billion, Century would obtain a potential positive return based on a pre-tax scenario of NPV of \$ 4.1 billion, 20.1% IRR and 4.2 year payback, An after-tax scenario shows an NPV of \$ 2.2 billion, 15.9% IRR and 4.8 year payback. The accuracy of the cost estimates is ± 35%.

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. The results of the economic analysis are forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

Based on the results of the PEA, Met-Chem recommends that Century continues to the next phase of project development.

1.18 Recommendations

Considering the positive results of DLIP PEA and discussions with Century, Met-Chem recommends that the project continues to the next phase of DLIP development, the Feasibility Study.

To establish a good base for the feasibility study and minimize the risks, Met-Chem recommends a series of studies and tests which are listed below: The main recommendations include:

- Increase the percentage of Measured and Indicated category relative to the Inferred Resources within Deposits 1 and 3 by additional diamond drilling;
- Firm up the definition of the geometry of Deposit 3, particularly the SE limb and the contact at depth of the synform;
- Investigate by a first pass of drilling some of the magnetic anomalies near the main deposits, such as the N-S trending anomalies of Deposits 3 and 6, or the anomaly branching off the north of Deposit 4;
- Increase the number of Davis Tube tests to 50% of the samples to improve the confidence level of the regression model and provide a better overall estimation of the Davis Tube Weight Recovery for the deposits;
- Determine the magnetic Fe content from Davis Tube and Satmagan tests on the same samples in order to calculate a correlation between the two;
- Use certified blank material and commercial standards, with certified Fe values close to the cut-off grade to the mode to monitor the laboratory performance;
- Perform a geotechnical analysis to increase pit wall slope and angle of repose of waste and overburden material, as well as hydrogeological and hydrological studies;
- Revisit the sequencing of Pushbacks for the Deposit 3 to maximize the project's NPV;
- Explore the potential of stockpiling and mining within Hydro-Québec property to be able to increase in-pit resources and shorten haul distances;
- Consider in-pit dumping to reduce environmental footprint and shorten haulage distances;
- Perform geochemistry study on more samples for better characterization and to confirm process conditions;
- Acid generation tests should be performed in order to know if there is a possibility of acid-generation on tailings and waste rock. Static testing has been performed and dynamic characterisation tests have to be carried out on the tailings.
- Perform grind size determination/optimization studies for all deposits (typical standard in taconite plant is a grind size of 44 micron (325 mesh);

- Perform mineralogical study on the iron mineralization to characterize the mineral species and to know the liberation size;
- Perform for each deposit, batch bench scale test work to confirm the flow sheet for the development of an overall magnetite processing plant;
- Obtain additional crusher, ball mill and rod mill bond work indexes (CW_i, BW_i, RW_i), to better define rocks hardness throughout the deposits;
- Determine detailed mineralogy of feed;
- Perform grindability test to evaluate variability of the mineralization;
- Perform additional bench scale testwork;
- Perform Pilot Plant investigation;
- Complete waste & tailings characterization (including leaching test and dynamic test);
- Confirm pellet feed characterization;
- Perform a series of balling and pot grate test on representative concentrate samples to define the pellet Fe and silica content as well as the grate factor temperature profile and all the other pellet quality parameters;
- Collect samples for vendor testwork (hydroclassifier, thickeners, filters, magnetic separators);
- Additional metallurgical tests will be necessary, such as: SG, mineral characterization, size distribution, bulk density determination, static thickening, dynamic thickening, pulp rheology, vacuum filtration, and pressure filtration.
- Explore a rougher magnetic separation stage in the ball mill grinding circuit to reject further portion of the non-magnetic gangue;
- Evaluate High Pressure Grinding Roll ("HPGR");
- Evaluate a second stage of crushing with cone crushers as an alternative to SAG mills;
- Perform test work with concentrate (from pilot plant) to define the pumping characteristics of the concentrate slurry and allow sizing of pumps and pipeline complete with a site visit to confirm pipeline routing and topography;
- Perform survey and geotechnical investigation at process plant buildings and infrastructure to provide soil and bedrock bearings elevation, depths and bearing capacities and provide information for more detailed quantity estimations;
- Explore transportation study to determine optimum shipping route and ship size;
- Confirm ice-free shipping season;
- Initiate an ice measurement program;
- Initiate a geotechnical investigation to collect design parameters for dredging and wharf design;
- Initiate bathymetric investigation to confirm bottom contours.

The estimated cost for the next study phase has been estimated and is provided in Table 1.6.

Table 1.6 – Estimated Cost for Next Study Phase

Study Phase	**Cost Estimate ($ M)**
Exploration Drilling Program	3.0
Feasibility Study	7.0
Metallurgical Testwork	2.0
Port	1.5
Geotech and Pit Slope	2.0
Other Site Studies	1.0
Environmental Studies	9.0
Total	**25.5**

2.0 INTRODUCTION AND TERMS OF REFERENCE

2.1 Terms of Reference – Scope of Work

Met-Chem Canada Inc. ("Met-Chem") was retained in February 2012 by Century Iron Mines Corporation ("Century") to prepare an independent technical report for a Preliminary Economic Assessment ("PEA") of the Duncan Lake Iron Project ("DLIP") in Quebec. This PEA is based on the updated Mineral Resources of DLIP prepared by Met-Chem in August 2012.

This report documents the results of the PEA study and constitutes a Technical Report under the guidelines of NI 43-101. The classification of the Mineral Resources used in the PEA is compliant with the CIM Definitions, in accordance with NI 43-101.

The present technical report is issued jointly to Century and Augyva Mining Resources Inc. ("Augyva").

2.2 Study Participants

Century retained Met-Chem services for preparation of the PEA Study report on the DLIP and related NI 43-101 Technical Report. This report is based on drill holes data provided by Augyva and Century and data gathered during the site visit by Met-Chem's qualified persons.

The qualified persons contributing to this report are listed Table 2.1. In addition to their respective sections, the qualified persons are also responsible for their portions included in the Summary (Section 1), in Capital and Operating Costs (Section 21), in the Conclusion (Section 25) and in Recommendations (Section 26).

Table 2.1 – PEA Areas of Responsibility

Name of QP	Company	Site Visit	QP Sections
Daniel M.Gagnon, Eng.	Met-Chem	September 2012	Sections : 1, 2, 3, 15, 16, 18.1, 19, 21, 24, 25, 26
Yves A. Buro, Eng.	Met-Chem	August 2011	Sections : 4-12, 14.1, 14.4, 14.6, 14.10-14.14, 23
Schadrac Ibrango, P. Geo. Ph. D.	Met-Chem	No site visit	Sections : 14.2, 14.3, 14.5, 14.7, 14.8 and 14.9
Stéphane Rivard, Eng.	Met-Chem	No site visit	Sections : 13 and 17
Charles H. Cauchon, Eng.	Met-Chem	No site visit	Sections :17.2.9
Daniel Houde, Eng.	Met-Chem	No site visit	Sections : 18.3-18.22
Raymond Gaudreault, P. Eng.	Portha Inc.	September 2012	Sections : 18.2 and 21.7
Mary Jean Buchanan, Eng., M. Env.	Met-Chem	No site visit	Sections : 20
Michel L. Bilodeau, Eng., M.SC. (App.), Ph. D.	Met-Chem	No site visit	Section : 22

2.3 Units, Abbreviations and Currency

Table 2.2 lists the units and other abbreviations used in the PEA.

All dollars are Canadian dollars unless noted differently.

Table 2.2 – Units and Abbreviations

Symbol	Abbreviation	Symbol	Abbreviation
ARG	Acid Rock Generation	MNR	Ministry of Natural Resource
AMSL	Above Mean Sea Level	Mt	Million metric tonnes
BIF	Banded Iron Formation	Mtpy	Million metric tonnes per year
¢/kWh	Cent per kilowatt hour	MW	Megawatts
CAD	Canadian Dollar	N	North
CAPEX	Capital Expenditures	NAG	Non Acid Generating
CIM	Canadian Institute of Mining and Metallurgy	NI	National Instrument
Cy, yd^3 or y^3	Cubic yard	NOx	Generic term for NO and NO_2
DDH	Diamond drill hole	NPV	Net Present Value
3D	Three Dimensional	NSR	Net Smelter Return
DLIP	Duncan Like Iron Project or Duncan Lake Iron Property	OPEX	Operating Expenditures
DT	Davis Tube	PEA	Preliminary Economic Assessment
DTWR	Davis Tube Weight Recovery	QA	Quality Assurance
DXF	Drawing interchange format	QC	Quality Control
E	East	QP	Qualified Person
EMAS	Eco Management and Audit Scheme	RQD	Rock Quality Designation
EPCM	Engineering, Procurement, Construction Management	S	South
g/cm^3	Gram per centimeter cube	S/R	Stripping ratio
h/y	Hour per year	SG	Specific Gravity
ha	Hectare	st	Short Ton (2,000 lbs)
IRR	Internal rate of return	t	Metric tonne (1,000 kg, 2,204.6 lbs.)
ISO	International Organization for Standardization	tpy	Metric tonnes per year
LOI	Loss on ignition	USD	United States Dollar
LOM	Life of Mine	XRF	X-Ray Fluorescence

Symbol	Abbreviation	Symbol	Abbreviation
LT	Long ton (2,240 lbs)	W	West
M	Million units		
mm	millimeter		
Mm^3	Million cubic meters		
m/h	Meters/hour		

2.4 Site Visit

The DLIP site was visited by Mr. Yves A. Buro between August 9 and 12, 2011.

Mr. Daniel Gagnon and Mr. Raymond Gaudreault visited the site on September 12, 2012.

3.0 RELIANCE ON OTHER EXPERTS

The Qualified Persons ("QPs") involved in this report are all QPs for the sections identified in the certificates of the QPs. The QPs have relied on expert opinions pertaining to mineral tenure and surface rights, environmental considerations and social/community impact as well as Iron Ore price assumptions and Taxation.

3.1 Mineral Tenure and Surface Rights

The QP has carefully reviewed the available information from the Duncan Lake Iron Property and the immediate surrounding area and believes the information to be correct and adequate. The information provided has been used to prepare section 4.0 of the report.

3.2 Environment Considerations and Social/Community Impact

The QP has relied on Le Groupe Desfor who was retained by Century as the environmental consultant for the DLIP environmental baseline studies. Le Groupe Desfor has the mandate to do all the environmental base studies for the project.

The QP has reviewed the information provided by Le Groupe Desfor and believes this information to be correct and adequate for use in the PEA. The information provided has been used to prepare Section 20 of the report.

3.3 Iron Ore Price Assumptions and Market Study

The QP has relied on long term iron ore pricing and market assumptions prepared by independent consulting firm Raw Materials & Ironmaking of Bethlehem Pennsylvania, who prepared an independent marketing and sales price analysis of the Duncan Lake Iron pellets. The report, titled "Century Iron Mines Ore Marketing Study", was prepared by Dr. Joseph J. Poveromo, a world renowned iron and steel marketing specialist and president of Raw Materials & Ironmaking. The report is dated February 25, 2013.

The QP has reviewed the information in the report by Dr. Poveromo and believes this information to be correct and adequate for use in the PEA. The results of this marketing evaluation and pricing forecasting are presented in Section 19 and have been used in the financial projections of Section 22 of the report.

3.4 Taxation

The QP has relied on fiscal information for use in the after-tax economic evaluation of DLIP by Mr. Marc Robert, CPA, CA.

The QP has reviewed the information and the results provided by Mr. Robert and believes this information to be correct and adequate for use in the PEA. The results of the after-tax economic evaluation have been used in the financial projections of Section 22 of the report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Description and Location

The Duncan Lake Iron Property ("DLIP") is located approximately 570 km north of Matagami, Québec, within the Municipality of James Bay (Figure 4.1). The property is 40 km south of Radisson and 10 km south of La Grande Rivière regional airport. The DLIP is 950 km to the NW of Montreal.

The property has an irregular, elongate shape, stretching over 30 km along a NE-SW direction (Figure 4.2). The property covers parts of NTS Sheets 33F05 and 33F12 and fits between coordinates 307,395 m E and 334,346 m E, and 5,925,320 m N and 5,944,962 m N in the UTM NAD 83, Zone 18 system. Different blocks have been defined in the DLIP property to describe sectors containing Deposits 1 through 6 which extend over 28 km southwest to northeast (Figure 4.2).

The DLIP property consists of one Block made up of 534 contiguous claims covering 25,605 hectares. An elongate rectangular tract of land controlled by Hydro Quebec truncates the claims along the center of most of the length of the property.

All the claims were acquired as Map-Designated Claims ("CDC") and the information on them is accessible through the Register of Real and Immovable Mining Rights in Quebec via the GESTIM application of the Quebec Ministry of Natural Resources ("MNR").

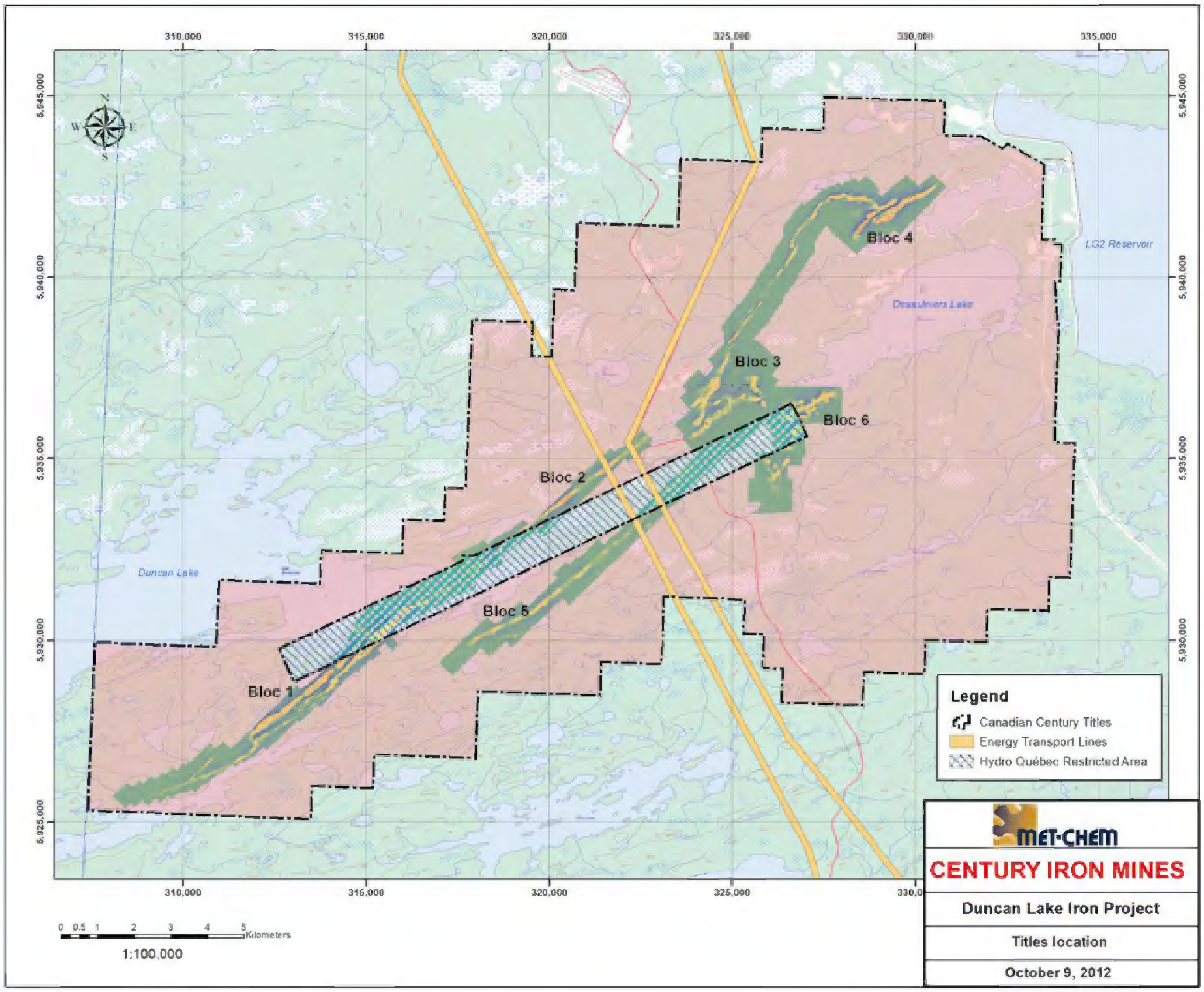
Forty-four of the 534 claims registered with GESTIM carry restrictions related to the Hydro-Québec electrode grounding system and/or a power line corridor, Table 4.1.

Table 4.1 – Registered Encumbrances on Claims within the Duncan Lake Property

Type of Encumbrance	Number of Claims Affected	Surface Area (ha)
Grounding Electrode	5	210
Grounding Electrode and Power Line Corridor	2	91
Power Line Corridor	37	1,811
Total	44	2,112

Figure 4.1 – Duncan Lake Iron Project Location, Québec

Figure 4.2 – Regional Location of Duncan Lake Iron Project



The DLIP claims are registered under Augyva and Century, Table 4.2.

Table 4.2 – Registered Owners of the Duncan Lake Block of Claims (GESTIM file as of April 15, 2013)

Title Holder (s)	Number of Claims	Surface Area (ha)
Augyva (35%)/Century (65%)	462	22,249
Augyva (42.14%)/Century (57.86%)	72	3,356
Total	534	25,605

All the claims were active and in good standing at the time of writing this report. A complete listing of the active claims is presented in Appendix A, extracted from the Register of mining rights in Quebec via GESTIM on April 15, 2013.

On April 1, 2013, Augyva announced that, together with Century, it has filed with the Quebec government, instructions to change DLIP claims to an ownership of 35% Augyva and 65% Century. The ownership change is consistent with completion by Century of its previously announced expenditures to earn-in to 65% ownership of the DLIP the claims.

The system shows registration dates from the earliest on July 26, 2004 to the most recent acquisition of three claims made on April 24, 2012. Expiration dates of the claims range from July 12, 2013 to April 18, 2015. The registered excess assessment work amounted to $9,944,207, the required work to renew the claims was $214,902 and the required fees were $63,350.00.

The size of a claim is generally a cell measuring 30 seconds longitude by 30 seconds latitude. Within surveyed territory, the outline of a claim is the same as that of a land lot, or part of. The claims give the owner exclusive rights to explore for mineral substances, with a few exceptions like hydrocarbons, sand and gravel.

Access to the claims is granted to carry out exploration work. However, the claim holder cannot enter land granted for non-mining purposes or land leased for mining surface mineral substances without permission from the current holder of these rights.

The claims have a validity of two years and can be renewed indefinitely for two-year periods, under certain conditions, provided the required exploration work is completed, and fees are paid. The assessment work requirement increases at each renewal. Excess work on one claim may be spread to other continuous claims held by the same owner within an area of 4.5 km^2.

4.2 Legal Titles Holders

Century Iron Mine Corporation ("Century") was incorporated under the Canada Business Corporation Act on July 10, 2007, with Head Offices at Suite 602, 170 University Avenue, Toronto, Ontario. The Company has two key strategic partners in WISCO International Resources Development & Investment Limited ("WISCO") and Minmetals Exploration & Development (Luxembourg) Limited S. àr. l. ("Minmetals") both state-owned Chinese companies.

Augyva Mining Resources Inc. ("Augyva") was incorporated under the Canada Business Corporations Act on December 5, 1986. Augyva's Head Offices are located at 1 Place Ville Marie, Suite 2500, Montreal, Quebec. The company is involved in the exploration of mineral properties located in the Province of Quebec, Canada.

On May 20, 2008, Century entered into an option and joint venture agreement with Augyva in respect of the DLIP (the "Duncan Lake Joint Venture Agreement"). In 2010, Century earned a 51% interest in the DLIP under the Duncan Lake Joint Venture Agreement after funding a commitment of $6.0 million. Since that date, Century has earned a cumulative 65% interest in the DLIP, having funded a further $14.0 million under the Duncan Lake Joint Venture Agreement.

Century has also entered into a Joint Venture Agreement with WISCO International Resources Development & Investment Limited ("WISCO") pursuant to which WISCO may earn a 40% joint venture interest in Century's interest in the DLIP.

4.3 Legal Agreements

Duncan Lake Joint Venture Agreement

Pursuant to the terms of the Duncan Lake Joint Venture Agreement, Century acquired the option to acquire up to a 65% interest in the DLIP as follows:

- To earn an undivided 51% beneficial interest in the DLIP by funding an initial $6.0 million on or before May 20, 2012; and
- Upon Century earning a 51% interest in the DLIP, to earn an additional undivided 14% beneficial interest in the DLIP by, on or before May 20, 2016, (i) expending an additional $14.0 million in exploration, construction or operating costs, or (ii) completing a feasibility report in respect of the DLIP.

In November 2010, Century completed its funding of an aggregate of $6.0 million to earn an undivided 51% beneficial interest in the DLIP in accordance with the terms of the Duncan Lake Joint Venture Agreement. Century subsequently completed the acquisition of an additional 14% interest by incurring a further $14.0 million in exploration, construction or operating costs.

Upon earning the initial 51% interest in the property, under the terms of the Duncan Lake Joint Venture Agreement, Century and Augyva formed a joint venture (the "Duncan Lake Joint Venture") for the exploration, and if warranted, development and exploitation of the DLIP and the operation of any mine or mines to be constructed on the property. Century is currently the manager of the joint venture and the operator of the property.

The Duncan Lake Joint Venture is directed and controlled by a management committee comprised of five members, three of whom are appointed by Century and two by Augyva. The management committee is responsible for, among other things, reviewing and approving exploration programs, preparing exploration programs (in the event the operator does not prepare an exploration program) and reviewing, amending and approving operating plans.

Upon earning the initial 51% interest in the DLIP, Century or its designee approved by the management committee will act as operator. Thereafter, the party with the largest interest in the DLIP or the party otherwise agreed to by the parties will act as operator. Under the terms of the Duncan Lake Joint Venture Agreement, the operator has such duties and obligations determined by the management committee from time to time including, proposing and, subject to the approval of the management committee, implementing exploration programs and any construction program and operating plans, managing, directing and controlling all exploration, development, construction and production operations in and under the DLIP, and preparing and delivering to Century and Augyva periodic progress and current reports and information on any material results obtained from active field work.

In accordance with the terms of the Duncan Lake Joint Venture Agreement, as Century has earned an additional 14% interest in DLIP, any additional exploration, construction program, and operating costs will be borne by each of Century and Augyva in accordance with their respective interest in the property determined in accordance with the terms of the Duncan Lake Joint Venture Agreement.

Pursuant to the terms of the Duncan Lake Joint Venture Agreement, if at any time after Century has earned an additional 14% in the Duncan Lake Property (or such other lesser additional pro rata interest) a participant elects or is deemed to have elected not to contribute to an exploration program or construction program, its respective interest shall be reduced, and the other participant's interest proportionately increased, in accordance with the formula set forth in the Duncan Lake Joint Venture Agreement. If the calculation results in a reduction of a participant's interest to less than 10%, its interest will be deemed to be converted into a royalty calculated in accordance with the terms of the Duncan Lake Joint Venture Agreement and thereafter such party will have no further rights or interest under the Duncan Lake Joint Venture Agreement except for the right to receive the net smelter return royalty.

WISCO Joint Venture Agreement

Century has also entered into a Joint Venture Agreement with WISCO pursuant to which WISCO may earn a 40% joint venture interest in Century's interest in the DLIP in consideration for an investment of $40 million. Following the execution of the Joint Venture Agreement, WISCO and Century concluded negotiations in November 2011 for a shareholder's agreement pursuant to which WISCO would make its investment into a newly formed company to be owned jointly by Century and WISCO and which would own Century's interest in the DLIP under the Duncan Lake Joint Venture Agreement. This shareholder's agreement has not been entered into as of the date of this Report.

Royalty Agreement with Virginia Mines Inc.

The DLIP is subject to an agreement concluded with Virginia Mines Inc. ("Virginia") in February 2005. A perpetual production royalty of $0.40 per ton of iron concentrate is payable to Virginia. In addition, a 2% net smelter return ("NSR") royalty on any metal other than iron is to be paid to Virginia.

Augyva retained a buyback right to purchase 50% ($0.20 per ton of concentrate) of the royalty for a payment of $4 million.

Furthermore, Augyva was granted an option of buying back a further 20% royalty ($0.08 per ton of concentrate) for an additional payment of $4 million. Any additional resource of iron mineralization identified on lands covered by any titles, in an area of 1 kilometer surrounding the property, acquired before or after February 2005 is also subject to the royalty. Augyva has a buyback right to purchase 50% of the NSR (1% NSR) for a payment of $5 million.

The property is located in an area ruled by the James Bay Convention, including aboriginal agreements for land use. The property is located in the Municipality of James Bay and should not be affected by specific land use Categories 1 or 2 as defined by James Bay Convention, which contained provisions such as exclusive land use by local aboriginal communities.

Met-Chem is not aware of any agreement under which aboriginal communities may hold title or historical agreement to the mineral land for this project.

4.4 Environmental Liabilities

Met-Chem is not aware of any environmental liabilities to which the DLIP is subject.

An active landfill, most probably used by the municipality of Radisson and by locals, exists on the property in the vicinity of Deposit 3. The landfill site is located at km 579 of the James Bay Road, at approximate coordinates 323 180E and 5 936 990N.

Environmental studies are underway on the project. Le Groupe Desfor has the mandate to review existing baseline studies performed to date and to conduct additional environmental studies deemed required for the project.

4.5 Permits

Permission from the current holder is required to access claims for non-mining purposes. This applies to Hydro-Québec land adjacent to the Duncan Lake property used for grounding electrodes.

A claim holder cannot erect or maintain a construction on lands in the public domain without obtaining a construction permit from the MRN, unless such a construction is for temporary shelters made of pliable material over rigid supports that can be dismantled and transported.

Exploration work including drilling, outcrop stripping, bulk sampling requires permitting from the MNR.

Met-Chem is unaware of any significant factors or risks that may affect access, title or the right or ability to perform work on the DLIP.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The project area can readily be accessed via Highway 109 connecting Radisson to Matagami, Amos, and further south, to Highway 117, between Rouyn-Noranda and Val-d'Or. The road is paved, well-maintained and ploughed in the winter. The road distance from Montreal to the DLIP is approximately 1,350 km.

Radisson is served by the La Grande Rivière regional airport located 30 km to the SW that offers daily scheduled flights to Montreal, Rouyn and other northern communities.

No access to the national railway system is available, the closest link being Rouyn-Noranda some 860 km South by road. The DLIP is situated approximately 130 km from the East shore of James Bay.

Six main deposits have been recognized in different sectors within the property and are referred to as Deposits 1 to 6. Highway 109 crosses the property in NW direction, between Deposits 1, 2, and 5 to the west, and Deposits 3, 4 and 6 on the east (Figure 4.2).

Deposits 1 and 5 are accessible via a gravel road off Highway 109 (at km 570) toward Duncan Lake. Access to Deposits 2, 3 and 6 that lie close to Highway 109, is provided by ATV trails. Deposit 4 can be reached via a gravel road connecting Highway 109 to Desaulniers Lake and the LG2 Reservoir, and westward along an ATV trail.

5.2 Climate

Radisson has a humid subarctic continental climate. presents the statistics compiled by Environment Canada since 1971 on the climate in the La Grande area.

The summer period lasts from mid-May to the end of September, while the winter season extends from late-October to mid-April. The DLIP region is out of the permafrost range. The average annual precipitation for the region consists of about 437 mm of rain and 267 cm of snow, see Table 5.1 for distribution per month. The prevailing winds stay relatively constant all year long, moving from West to South during part of the winter period.

Climatic conditions do not support any commercial agriculture or farming activities.

Table 5.1 – Statistics on Climate in La Grande Rivière – 1971-2000 (Environment Canada)

Parameter	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Daily Average Temperature (°C)	-23.2	-21.6	-14.6	-4.9	4.3	10.5	13.7	12.9	7.4	1.2	-6.3	-17.1
Daily Maximum Temperature (°C)	-18.3	-15.8	-8.2	0.7	10.3	17.1	20.0	18.4	11.6	4.4	-3.3	-13.0
Daily Minimum Temperature (°C)	-28.0	-27.4	-20.9	-10.5	-1.6	3.9	7.4	7.4	3.1	-2.0	-9.4	-21.2
Rainfall (mm)	0.1	1.2	3.0	11.5	30.2	62.3	79.5	85.1	100.8	52.5	10.2	1.2
Snowfall (cm)	34.2	23.3	29.5	20.9	11.1	2.5	0.0	0.1	6.4	35.5	60.8	42.5
Wind Speed (km/h)	14.4	13.8	14.6	14.4	15.0	15.5	14.5	14.6	15.9	16.2	16.0	14.3
Most Frequent Wind Direction	W	W	W	W	W	W	W	W	W	S	SW	S

5.3 Local Resources and Infrastructure

Radisson is the closest municipality to the DLIP and was founded in 1974 to provide housing and to support development of the La Grande hydroelectric complex.

Over the years, the population of Radisson has fluctuated according to the needs of the large construction sites in the area. The population of Radisson reached 2,500 persons toward the end of the 1970's, and was at 270 in 2011 (Statistics Canada, 2012).

Services such as accommodation, hospital, car rental agencies, helicopter or float planes chartering, contractors with heavy machinery can be found in Radisson. La Grande Rivière Airport is used to shuttle Hydro-Québec personnel between Radisson and the larger cities in Quebec. It is also served by daily scheduled flights.

The closest aboriginal communities are Chisasibi and Weminji located respectively to the northwest and southwest, near the James Bay shore. Chisasibi is connected by paved road 90 km from Highway 109 with modern facilities for a population estimated at 4,000 residents. Weminji has 2,500 residents and is connected by a gravel road branching off Highway 109 at km 518.

Personnel for a mining operation can be found locally, but most hired labour will require training and skilled professionals will likely be sourced from cities in the South.

Two power lines skirt the edge of Deposit 3 and cross the corner of Deposit 2 of the DLIP (Figure 4.2). Water supply is abundant, considering all the lakes within, or adjacent to the property.

The property, with a surface area of 256 km^2, is sufficiently large to accommodate required mining infrastructure including stockpiles, tailings storage, processing plants and associated buildings.

5.4 Topography, Physiography and Vegetation

The topography of DLIP is characterized by low relief with a NE grain marked by elongate, low rocky hills, glacial till and low-lying swampy areas. Relief on the property ranges from elevations 140 m to 182 m ("AMSL"), with Duncan Lake at 139 m and Desaulniers Lake at an elevation of 152 m ("AMSL").

The iron formations do not have a positive topographic surface expression. Scarce outcrops are found over Deposits 1, 2, 3, and 5 and fewer yet on Deposits 4 and 6. A fair proportion of the property is covered by bodies of water, including the two large Duncan and Desaulniers Lakes.

Vegetation in the area is characterized by stunted black spruce and jack pine accompanied by less frequent tamarack. Birch, poplar and alder are common along streams and lake shores, and in areas affected by forest-fires. Caribou-moss, shrubs, berries are common, lichen is ubiquitous. The area was never logged in the past and is not considered for mature commercial wood.

Beaver, otter, muskrat, marten, lynx and wolf are present in the area, as well as black bears and moose. Sizable populations of seagulls, partridges, geese, black ducks, blue jays and sparrows are also living in the region. Pike and walleye populate the lakes and streams of the area, and speckled trout are caught in small lakes. Whitefish and carp are found in some lakes which are connected to the La Grande Rivière and sturgeon is reported in La Grande Rivière. Duncan and Desaulniers Lakes are used for recreational fishing activities.

6.0 HISTORY

6.1 Prior Ownership

Duncan Range Iron Mines was created in 1957 and acquired the Duncan Lake Property. An agreement was reached with International Mogul Mines Ltd for a five-year period but the relationship between the two is unknown. Augyva acquired the DLIP property from Virginia Mines Inc. in 2005.

6.2 Exploration Work

Although the Geological Survey of Canada ("GSC") was active in the area, the discovery of iron formation around Duncan Lake in 1949 is attributed to a group of prospectors.

The same prospectors returned to the region in 1953, as interest increased in beneficiation of magnetic iron formation (taconite). A property was staked and a magnetic survey over 12 km of strike length was completed.

A large amount of work has been completed in the area by mining companies, Universities and government agencies. The main focus of the exploration efforts in the region has been directed at iron formation, but chrome and platinum group elements (PGE), uranium, gold and copper mineralization have also been targeted.

Table 6.1 provides a summary of the exploration work most pertinent to the Duncan Lake property. Additional information can be found in the previously filed Technical Report NI 43-101 on the Mineral Resources of the Duncan Lake Iron Project, dated December 2, 2010, prepared by Met-Chem.

The holes drilled by Augyva and Century between 2008 and 2011 form the basis of the current 3D model and August 24, 2012 resource estimate.

Table 6.1 – Summary of Main Historical Exploration Work

Period	Company	Exploration Work
1949	Prospector J.C. Honsberger et al.	• Discovery of iron formation at Duncan Lake
1953	Prospector J.C. Honsberger et al.	• Claim staking • Ground magnetometer survey
1956-71	Duncan Range Iron Mines Ltd. (DRIML); Geological Survey of Canada	• Ground and airborne (250 miles) magnetometer surveys • Mapping, sampling • Exploration for base metals • 8 diamond drill holes north of deposit 1 and 5, within Hydro-Québec restricted area, and into deposit 1 • Metallurgical testing
1972-73	DRIML – International Mogul Mines Ltd.	• Diamond drilling of 4,188 m in 22 holes (deposits 3, 4 and 6)
2005-06	Augyva	• Acquisition from Virginia Mines Inc. • Field mapping, sampling, staking • Ground magnetic survey • Metallurgical testing, by COREM
2008	Augyva	• NI 43-101 Technical Report by Geologica, Val-d'Or, on historical work
2008-2009	Augyva-Century	• 10,460.25 m of core in 52 holes drilled into deposits 1 to 5 • Metallurgical testing by COREM on 218 samples
2010	Augyva-Century	• NI-43-101 Technical Report by Met-Chem on the Resources Estimate
2011-2012	Augyva-Century	• 44,006.55 m of core in 125 holes drilled into Deposits 3, 4 and 6
2012	Augyva-Century	• NI-43-101 Technical Report by Met-Chem on the Resources Estimate

6.3 Historical Mineral Resources

Mineral resources estimations were completed in 1958, 1973 and 1976. However, these historical resources are non-compliant with the NI 43-101 rule, are outdated and the details on the methodology used and the investigated area are lacking. Met-Chem's opinion is that they are irrelevant for the purpose of this report, are superseded by the current estimate and, consequently, they will not be discussed. Met-Chem estimated mineral resources for the Duncan Lake property in 2010, but these are superseded by the present updated resource estimate.

6.4 Production

No production of iron mineralization has been reported from the Duncan Lake property.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

All geological information in this chapter is drawn from public documents prepared by University, Government agencies or mining companies, as well as from the previously SEDAR filed Technical Report by Met-Chem (February 3, 2010 and October 11, 2012) which contains additional geological descriptions.

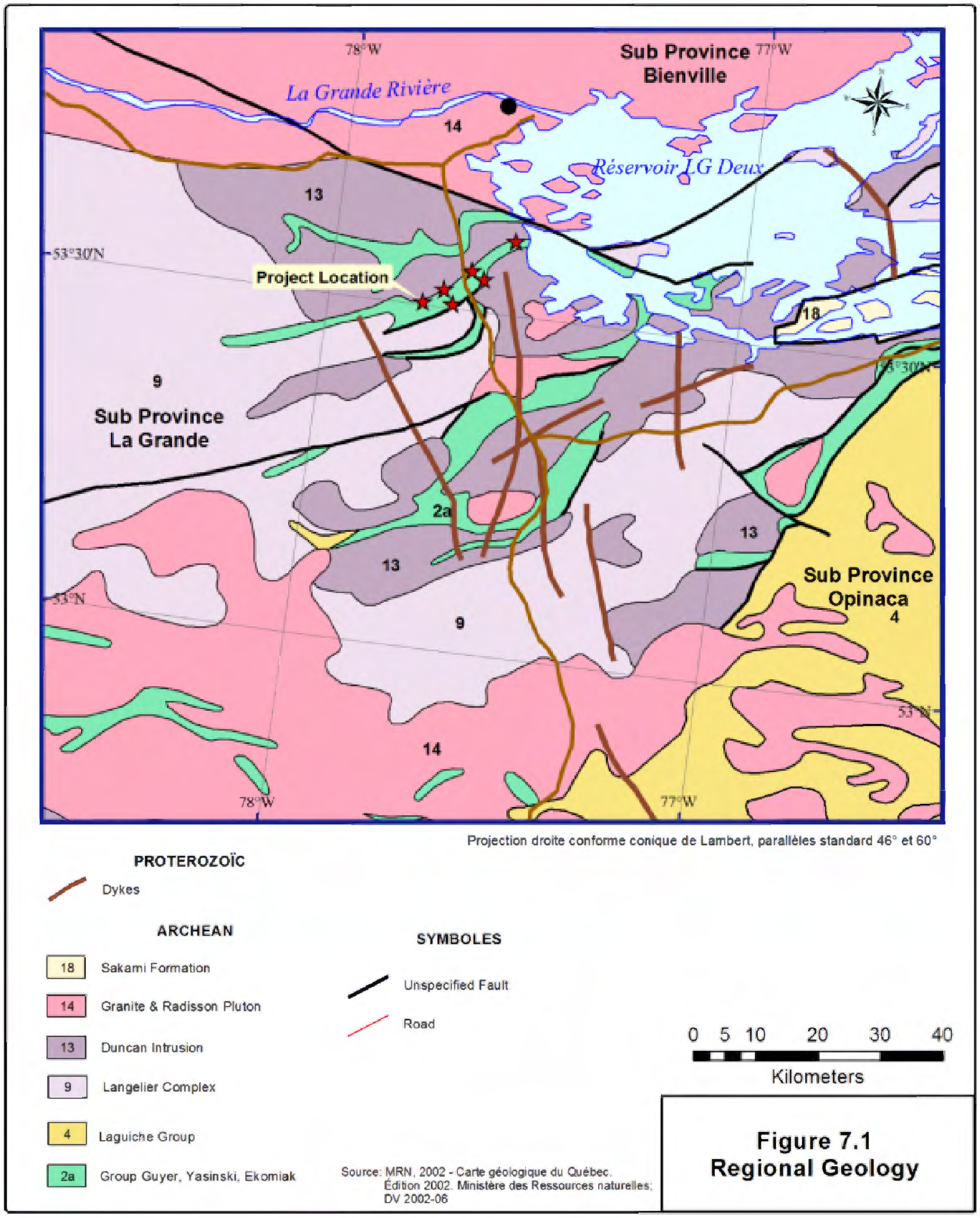
The DLIP lies within the Western Part of the La Grande Sub-Province of the structural Superior Province. The La Grande Sub-Province lies between the Opinaca Sub-Province to the E and SE and is separated from the Bienville Sub-Province to the north by a W-NW fault (Figure 7.1).

The La Grande Sub-Province is characterized by an Archean tonalitic gneiss basement and tonalite associated with the multi-phase Langelier Complex. The Langelier Complex is uncomfortably overlain by the volcano-sedimentary Guyer and Yasinski Groups composed of iron formation, wacke, paragneiss, basalt to dacite and pyroclastic units. The alluvial or fluvial sediments of the Ekomiak Formation partly lie on the Yasinski Group. The sediments of the Sakami Formation were deposited in NE-trending sedimentary basins.

All these rocks are intruded by several syn- to late-tectonic plutons and some ultramafic intrusions and diabase dikes. The Duncan Lake and the Radisson Plutons are the most important in the region of the DLIP.

The range of regional metamorphism varies from greenschist to amphibolite facies. The supracrustal rocks have been deformed by at least two structural events, an early one forming a subvertical, N-S penetrative schistosity followed by a dominant event resulting in a steeply south-dipping, E-NE trending schistosity. Folding and intense shearing developed along an E-NE trend.

Figure 7.1 – Regional Geology



7.2 Property Geology

7.2.1 General

The DLIP is underlain by two parallel Banded Iron Formation ("BIF") units cutting across the entire property in a north easterly direction (Figure 4.2). The mafic volcanic rocks dominate in the area of known BIF presence, but the felsic material, intrusive rocks and possible basement granite prevails in the Deposit 6 sector. All the mapped or interpreted lithological units are elongate and trend N-NE.

The maps by G. Ivanov, geologist working for IOS Services Géoscientifiques Inc. ("IOS"), show the area of Deposit 3 to be hosting BIF and basalt with subordinate amount of discontinuous, elongate felsic intrusive bodies. The Deposit 6 map shows bands of BIF interspersed with felsic material, one band of basalt along the full length of the Deposit area on the SE and a wide band of gneiss / felsic rocks along the length of the SE boundary of the drilled area. Deposit 4 is characterized by a central BIF unit, crossed along its central portion by a long greywacke horizon. Basalt underlies the NW portion of the area, whereas elongate basalt and felsic bodies occur on the SE.

The attitude of the bedding and schistosity measured by G. Ivanov is universally steep, sub-vertical in the area of Deposits 3 and 6, ranging from 65 to 90° over Deposit 4.

The mafic volcanic and sedimentary rocks in the DLIP area have undergone some degree of alteration, shearing, folding and regional metamorphism. The iron formations are typical BIF type with alternating 1 to 10 mm magnetite and silica beds.

The intrusive series vary in composition from granite to hornblende gabbro emplaced as dykes or sills. The volcanic assemblage is assumed to be on the top of the rock sequence. However, volcanic rocks are located at different levels and on each side of the BIF units.

7.2.2 Description of the Deposits

The arrangement of the DLIP deposits is well displayed on the magnetic maps. Two sub-parallel, north easterly trending bands forming Deposits 1-2 on the northwest are separated by about 1.8 km from Deposit 5 on the southeast and appear to join Deposit 3. The northwestern branch of Deposit 3 is connected to Deposit 4 by one NE magnetic anomaly exhibiting a sharp curve at the NE end to meet the center of the NE trending Deposit 4. Deposit 6 seems to be disconnected from the other deposits and is made up of a NE anomaly with an N-NE branch to the southwest.

Deposits 1 and 2 are part of one continuous N-NE trending band traceable over about 17 km, including the portion crossing the Block excluded from exploration for Hydro-Québec's use.

Deposit 1 consists of two main sub-parallel, "en-echelon", 60 to 130 m thick BIF separated by 70 to 80 m of greywacke. Those BIF have an open sigmoidal shape with a general strike of N230° and dips at -70° to the northwest. A 50-m wide gabbro dyke oriented N320° cross-cuts all rock units. Deposit 1 can be traced over more than 5.5 km along the strike, the northeastern 2.5 km of which are inside the Block excluded from exploration for Hydro-Québec's use. Thirty-one boreholes were drilled on this deposit in 2008-2009 by Augyva. 3D modelling and a resource estimate were completed after this drilling program and in 2012, but additional drilling has been completed on this deposit since.

Deposit 2 is a single BIF unit expressed by a weak magnetic anomaly containing a series of isolated peaks, the highest two of which were drilled and correspond to an increase in thickness of the BIF. The BIF units are generally narrow but locally reach widths of 30 to 70 m. Deposit 2 strikes N225° and dips at 80 to 85° toward the northwest. The core descriptions indicate that the BIF is accompanied with mafic volcanic units, sediments, felsic dykes and sills. Interpretation on six sections from only seven boreholes drilled in 2009 by Augyva shows that three iron lenses are present. A resource estimate was performed by Met-Chem after the 2008-2009 drilling program (Technical Reports of 2010 and 2012). No additional drilling or interpretation was done on this deposit since.

Deposit 5 is a 4-km long, slightly arcuate, 25 m thick BIF unit located to the southeast of Deposits 1 and 2. The general strike of Deposit 5 is N225° and the dip is at 60° to the northwest. Its possible southwestern extension is undetermined. The magnetic anomaly show at least two peaks (highs). Six boreholes drilled in 2009 indicate a true thickness of 5 to 25 m on the northern lens (magnetic anomaly), whereas the southern lens does not outcrop and was not drilled. A resource estimate was performed by Met-Chem (Technical Reports of 2010 and 2012). No additional drilling or interpretation was done on this deposit since.

Deposit 3 is a complexly folded BIF best described by its magnetic expression as a large V-shaped anomaly around a northeast axis and a smaller V-shaped anomaly with a northwesterly oriented bisector affecting the southeastern limb of the major structure. Drilling has shown two main BIF units arranged as a large-scale, tight synform (on the northwest) and antiform (on the southeast) system, with a shallow plunge to the northeast and steeply northwesterly dipping axial planes. A few BIF units intersected by drilling over parts of the deposits to the northwest of the main BIF units are interpreted to be in fault contacts and to have been thrust against the main units and added by stacking.

The northern flank of the synform is oriented N030°/-75°SE, generally 150 to 250 m thick, formed by two to three BIF units. The south limb of the synform is oriented N250°/-70°NW and is about 50 to 75 m thick. The southern flank of the antiform, oriented N050°/-55°SE is generally 50 to 150 m thick and is composed of one or two BIF units. The limb of the syncline on the southeast leaves the system open at depth.

A late decametric to pluri-decametric granitic dyke has been identified in the present mapping and drilling campaign. Its strike is sub-parallel to the axial plane of the synform and crosses the nose of the synform in its southwestern portion. Its dip has not been clearly defined because of lack of drilling in its extension at depth.

More drilling is needed to define the geometry of the sector at the northeastern end of the southern flank of the antiform where an N-S BIF unit occurs. The holes drilled during the 2011-2012 campaign led to a complete reinterpretation of the deposit and to a new resource estimate described in Met-Chem's Technical Report of 2012.

Deposit 4 is located about 5 km to the northeast of Deposit 3. It is 2.5 km long and consists of stacked BIF lenses 20 to 150 m thick, forming two clusters separated by 50 to 60 m of sediments. Volcanic and granitic intrusions are found above and under the BIF, potentially due to isoclinal folding. On the thickest section, the total width of six adjacent BIF and felsic rocks units approaches 450 m. The deposit is generally oriented N050° and dips 60 to 70° to the southeast. The holes drilled during the 2011-2012 campaign led to a complete reinterpretation of the deposit and to a new resource estimate, (Technical Report by Met-Chem, 2012).

Deposit 6 is located 1 km due east of Deposit 3. It is estimated to be 1.5 km long according to the magnetic data, including the southwestern most half-kilometer that falls into the Block restricted from exploration and therefore has not been drilled. The general strike direction is N065° and the dip is sub-vertical to steep to the southeast. One striking feature of Deposit 6 is the high proportion of felsic intrusive rocks accompanying the BIF, as compared to the other deposits. Indeed, Deposit 6 is heavily invaded and dilated by late granitic intrusions, part of which will probably represent internal dilution. The width of the BIF-Granite zone is highly variable, ranging from 50 to 500 m. A few mafic dykes sub-parallel to the BIF have been observed in the middle of the deposit but the relation between the two was not clearly established. The holes drilled in 2011-2012 led to a complete reinterpretation and to a resource estimate described in Met-Chem's Technical Report of 2012.

Except for Deposit 3 that is characterized by complex folds, the other deposits show essentially tabular, parallel BIF units, with various degrees of felsic intrusions and proportions of interbedded mafic volcanic and sedimentary rocks. Deposits 3, 4, and 6 are closely associated with granitic basement and contain more sills and dykes.

A more detailed description of this deposit can be found in Met-Chem's previously filed Technical Report by Met-Chem (February 2010).

7.2.3 Lithology

a) Banded Iron Formation ("BIF")

BIF units are observed at various levels within the sequence of altered mafic volcanic and volcano-clastic rocks. Three types of BIF are observed:

- Oxide BIF composed of magnetite-rich bands alternating with quartz-rich bands;

- Silicate BIF composed of magnetite-rich bands alternating with iron silicate bands including chlorite, actinolite, diopside, and hornblende as well as free silica and biotite;
- Lean BIF or low-grade iron units associated with greywacke. The lean BIF commonly contains silicate minerals and magnetite bands or disseminations.

Quartz (chert) and magnetite are the main constituents of the BIF. Hematite was not observed in the core but is present. The thickness of the iron beds varies from 1 to 10 mm but is generally in a 2-5 mm range. The sulphides are represented by pyrite and traces of pyrrhotite, chalcopyrite.

b) Basalt and Intermediate to Mafic Tuff

Mafic volcanic rocks are represented mainly by altered and locally sheared basalt and intermediate to mafic tuff. The basalt is usually massive and locally occurs as an amphibolite with garnet and biotite. The pillow basalt variety is amygdaloidal, of dacitic and andesitic composition, is stretched and interdigitated with tuff and sediment, and locally host magnetite bands or disseminations.

c) Quartzite and Greywacke

The greywacke is light to dark grey, fine- to medium-grained, composed of sand size grains set in a fine greenish chlorite mudstone matrix, with garnet-rich bands. The greywacke exhibits laminae of quartz 1 to 5 mm thick, alternating with darker, 1 to 3 mm biotitic bands. This unit is closely associated with the quartzite generally located at the base of the iron formation sequence.

d) Dykes and Sills

Intrusive rocks include gabbro, monzodiorite, granite, and syenite, feldspar porphyry sills and dykes. Late diabase dykes were mapped on outcrops at Deposit 1. The feldspar porphyry units are probably part of Amisech Wat Pluton, which consists of tonalitic porphyry and appears to be part of the youngest Archean intrusion in the area.

7.2.4 Mineralization

Iron mineralization within the DLIP property consists of BIF type represented by alternating millimetric to centrimetric bands of quartz and of magnetite with minor amounts of hematite. The DLIP deposits are also associated with silicate and sulphide facies iron formations. The silicates consist mainly of actinolite, tremolite, diopside, grunerite, hornblende, chlorite, epidote, biotite, as well as chlorite-rich mudrocks.

The BIF is black to dark blue and becomes dark-green with increased silicate content. It is generally fine-grained, with local variations due to recrystallization in response to contact metamorphism by the felsic intrusions near Deposits 3 and 6. Average head grade varies from 15 to 35% total Fe.

Disseminated sulphides, mostly pyrite, are widespread in the BIF sequence and some of the adjacent waste rocks and represent the main contribution in the sulphur deleterious element to the system.

The chemical analysis established that, on average, the iron formation at DLIP contains very low levels of deleterious elements, in particular phosphorus (0.02% P_2O_5), manganese (0.03% MnO) and magnesium (0.23% MgO). However, the average sulphur content is elevated. The unweighted average of 3,662 analyses for all deposits yielded 0.78% S in the head grade from core samples.

The sequence of the iron-rich units on the DLIP is not clearly defined and is related to a mix of Algoma and Lake Superior types BIF.

7.2.5 Structure

The rocks of the DLIP property area have been affected by at least two regional tectonic events, an early one forming a sub-vertical, N-S, penetrative schistosity followed by a dominant event resulting in a steeply south-dipping, E-NE trending schistosity.

The general E-NE trend is reflected by the general arrangement of the 6 deposits and the axial plane of the main folds of Deposit 3. The N-S trend of parts of the BIF of Deposits 3 and 6 may be relicts of the N-S tectonic event.

Deposit 3 is characterized by tight folds. Evidence of addition of BIF units by stacking was found, particularly in deposits 3 and 6. Fold repetition of BIF units may also have occurred locally by isoclinal folding but this has not been well documented during the drilling and mapping activities.

The understanding of the structural elements on the DLIP property is restricted by the general scarcity of outcrops and largely relies on drill hole and magnetic survey data.

8.0 DEPOSIT TYPES

The stratiform, iron-rich deposits in the world are classified as Lake Superior and Algoma types banded iron formations ("BIF"). The BIF are interpreted as finely bedded sedimentary rocks of chemical origin.

The Canadian Lake Superior-type iron formations are the most important source of iron, followed by the deposits of Algoma-type iron formations. Algoma-type iron formations are predominantly found in Archean greenstone belts, while the Lake Superior type is generally of Paleoproterozoic age.

The Lake Superior type of iron formation formed in stable continental shelf and platform settings, whereas the Algoma type is associated with volcanic arcs (tectonically active environment, greenstone belts) or rift zones and related volcanism. However, a complete gradation exists from one type to the other, as the Algoma type iron formation can be deposited at considerable distances from the volcanic centres.

The Algoma deposits are interbedded with felsic to ultramafic volcanic and volcaniclastic rocks, greywacke, black shale, argillite and chert. The Lake Superior iron formation is associated with a variety of shallow marine sedimentary rocks including quartzite, shale and carbonate rocks, and subordinate amounts of felsic to mafic volcanic rocks.

The Lake Superior type, formed on the large areal extents of the platforms, typically have higher lateral continuity and thickness than the Algoma type BIF originating from the more restricted volcanic environment.

The DLIP hosts iron deposits typical of the Algoma type oxide facies, considering the presence of an important volcanic rocks component. However, the large extent of the BIF in the DLIP deposits suggests a setting of deposition in a relatively stable environment. Consequently, the Duncan Lake BIF is interpreted to represent a type half-way between the Algoma and Superior types, sometimes referred to as Carajas iron mineralization. The oxide, silicate, carbonate and sulphide facies are common to both of these groups that contain a minimum of 15% Fe that typically varies between 30 and 35% Fe.

Banded iron formation consists of thin alternating beds of silica (quartz, chert) and iron oxides (magnetite and hematite), with variable amounts of silicate, carbonate and sulphides. The sequences are commonly metamorphosed. Grain size varies according to the degree of metamorphism and iron amphiboles are commonly developed in middle greenschist or higher metamorphic grade rocks.

The model used to design the exploration and drilling activities on the DLIP mineralization is based on the classification of the mineralization as large, Algoma type BIF deposits rich in magnetite. The definition of the deposits heavily relies on the large amount of available drill data, complemented with the results from field mapping, airborne and ground magnetometer surveys. The sheer extent of this type of deposits and their magnetic signature facilitates the design of drill programs.

9.0 EXPLORATION

A wealth of geoscientific data has been generated in the region of the DLIP property, since the discovery of the Duncan Lake deposits in 1949, principally by mapping, geophysical surveying and preliminary metallurgical testing completed by the Ministry of Natural Resources, research bodies and mining companies.

Compilation and re-interpretation of existing geophysical data, as well as several ground magnetic surveys, mostly aimed at filling in the gaps between existing grids, have been completed by Augyva and Century. Joël Simard, Geo., Geophysicist, was largely involved in the field activities, compilation and data analysis. The BIF have a distinct high magnetic signature which makes them stand out from the country rocks.

Gennady Ivanov, geologist with IOS, completed field mapping over deposits 3, 4 and 6. Two maps were generated by IOS in May 2012.

Laser radar (LIDAR) survey was flown over the DLIP by XEOS Imaging Inc., Quebec City, on June 27 and 28, 2012. An Optech ALTM Gemini mapping system was used and the flights were completed at two different altitudes. The data was being processed at the time of writing the report in 2012 and has not been used since.

10.0 DRILLING

10.1 Historical Drill Program (1973)

The first well-documented diamond drilling program was completed in 1973 under Duncan Range Iron Mines exploration period, in association with International Mogul Mines Limited. Twenty-two boreholes were completed for 4,141.54 m of AXT core into Deposits 3, 4 and 6.

However, the results from this drilling program were not included in the current geological model, because of the uncertainties related to the exact hole location, the unavailability of the core and the lack of details on the analytical procedures,.

In addition, the holes drilled in 2011-2012 were predominantly infill holes to the 1973 and 2008-2009 program, or adjacent to most of the 1973 holes, which reduced the necessity of using the data from 1973.

10.2 2008-2009 Drill Program

This drilling program was initiated in July 14, 2008, stopped for Christmas in mid-December, re-started in January 2009 and ended in May 2009. The drilling was conducted by Forage Baie James Inc. of Radisson, Quebec, using one diamond drill rig (model EF-50). A total of fifty-two boreholes for 10,461.4 m of NQ-sized core were drilled on all but Deposit 6 of the DLIP.

The core was logged onto paper sheets and the records were later entered into Microsoft Excel files. Photographic records, Rock Quality Designation (“RQD”) and Core Recovery (“REC”) values were included in the logs. Average core recovery was reported as approaching 98%.

Magnetic susceptibility measurements performed on the core of thirteen diamond drill holes did not provide much additional information on the mineralization and different rock types than can be gained by visual examination.

6,489.6 m of core were sampled and analysed by ALS-Chemex Laboratory in Val-d’Or, Quebec, in addition to the duplicates, standards and blanks inserted as QC samples to monitor the laboratory performance. In addition, 218 samples were sent to COREM, an independent laboratory located in Québec City, for metallurgical testing and served as a check by a secondary laboratory.

Nominal samples length was about 3 m for the first 22 boreholes and 5 m for the rest. The core was cut in half using a diamond saw and hydraulic splitters.

Details on the 2008-2009 drill program are provided in the previously filed Technical Report by Met-Chem (2010).

10.3 2011-2012 Drill Program

A drill program of 44,006.65 m of core in 125 holes was completed between January 27, 2011 and March 22, 2012 (Table 10.1). The holes were drilled into Deposits

3, 4 and 6 at angles between -90° to -50°, toward the NE or the SW, on the section lines, and to depths ranging from 120 m to 624 m.

Table 10.1 – Summary of the 2011-2012 Diamond Drill Program

Deposit	Number Of Holes	Meterage
3	69	24,226.55
4	37	13,354.70
6	19	6,425.40
Total	125	44,006.65

The main purpose of this drill program was to better define the geometry of the Deposits, delineate additional tonnages of resources and increase the confidence level of the resources to be used in the resources estimate and a Preliminary Economic Assessment of the project.

Three drill rigs equipped to retrieve NQ size core were active in 2011 and two were used in 2012. The drilling contractor in 2011 was DJM Drilling Inc. of Rouyn-Noranda operating two of its rigs, as well as one owned by Century. In 2012, drilling was completed by Orbit Garant Drilling Inc. of Val-d'Or, Quebec, with two drill rigs.

Two holes were drilled into each of the Deposits 3, 4 and 6 primarily to provide material for metallurgical testing (Table 10.2). These holes were drilled in HQ-sized core and had been designed to serve the needs of deposit definition and sampling, in addition to those of the metallurgical testwork.

Table 10.2 – Diamond Drill Holes for Metallurgical Testwork Purposes

Deposit	Drill Hole	Total Depth (m)
3	DUN-11-352	456.0
	DUN-11-237B	438.0
4	DUN-11-315	438.0
	DUN-11-324B	237.0
6	DUN-11-380	405.0
	DUN-11-382	375.0
Total		2,349.0

Core logging was done directly by IOS geologists into the dedicated software GeoticLog. Photographic records, Rock Quality Designation ("RQD"), core recovery and magnetic susceptibility were recorded, in addition to the lithologies (main units and sub-units), mineralization, alteration, structure.

7,689 samples representing 22,148.65 m of core were sampled, in addition to the duplicates, standards and blanks inserted as QC samples to monitor the laboratory performance. Nominal samples length was 3 m, but ranged from 1.5 m to 4.5 m in order to honour the main lithological contacts. The core was cut in half using a diamond saw and hydraulic splitters.

All the collars were surveyed with a Differential Global Positioning System ("DGPS") and the downhole deviation was measured using a gyroscopic instrument.

The field activities related to drilling and mapping were implemented and supervised by IOS, until the last day of drilling in March 2012.

The true thickness of the iron lenses relative to the core lengths varies as the holes intersected the mineralized zones at different angles.

No drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results were observed by Met-Chem in this drill program. The DGPS survey of all the hole collars and the use of a gyroscope instrument to measure the hole deviation provide accurate location of the holes in the deposits.

Met-Chem believes the density of drilling is adequate to have delineated the geometry and grade of the mineralization with sufficient precision to be used in a resource estimate.

11.0 SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 1973 Drill Program

Essentially, all is known about the samples from the 1973 drill program is the fact that the iron was analysed by titration of soluble iron. The results from this drill program were not incorporated into the present geological model.

11.2 2008-2009 Drill Program

11.2.1 Sampling, Analysis

The geologists from Augyva and Century marked the sample intervals on the core and a line along which to split it. Core splitting was done at the core storage area in Radisson using both hydraulic splitters and diamond blade saws. One half of the core was placed back in the boxes, along the original orientation and position, the other half was inserted into a plastic bag with a tag for identification. Nominal samples length was about 3 m for the first 22 holes and 5 m for the rest.

The samples were delivered in thick polypropylene bags to ALS-Chemex Laboratory, in Val-d'Or for preparation and analysis. The samples were analysed for major oxides via XRF and sulphur in a Leco furnace.

At the laboratory, the samples were reduced to 6 mm with a jaw crusher, cleaned with compressed air between each sample and with barren material between the sample batches. The samples were then reduced to 90% passing 10 mesh with a roll crusher cleaned between samples with a wire brush, compressed air, and with barren material between sample batches.

The first sample of each sample batch was screened to ensure that 90% passes 10 mesh. The mass of the samples was then reduced to about 300 g using a Jones riffle splitter. Excess material was stored as crusher rejects. The 300 grams portion was pulverized to 90% passing 200 mesh in a ring and puck pulverizer, cleaned between samples with compressed air and with silica sand between the sample batches. The first sample of each batch was screened to ensure that 90% passes 200 mesh.

In addition, 219 samples from Deposits 1 to 4 selected for metallurgical testing were re-analysed by COREM and served as check samples analysed by a secondary laboratory. From these 219 samples, 144 Davis Tube concentrates and tails were analysed, in addition to the head analyses.

Details on the 2008-2009 sampling are provided in the previously filed Technical Reports by Met-Chem (2010 and 2012).

11.2.2 Monitoring of the Laboratories

The blanks, standards and duplicates added by the geologists to the samples batches analysed by the ALS-Chemex laboratory as control samples represented 7.7% of the total.

In addition, an equivalent of 9.7% (144 samples) of the total number of samples (1,489) was sent to a second laboratory. These 144 samples were part of 219 samples sent to COREM for metallurgical test purposes, and were re-analysed to determine the head grade, in addition to the Davis Tube concentrate and tails.

Visually barren quartz vein used for 22 blank samples representing 1.5% of the total returned values ranging from 0.24 to 0.97% Fe with an average Fe content of 0.42%. Met-Chem attributed the presence of the low-level iron as an addition at the sample preparation stage.

From a total of 49 samples submitted as duplicate samples, only two pairs exceeded the acceptable threshold. In addition, no apparent bias was observed.

A total of 144 samples rejects originally analysed by ALS-Chemex were submitted as duplicate samples to COREM laboratory prior to metallurgical testing. Both laboratories used the same analytical methods. The COREM's results indicated higher Total Fe % than ALS-Chemex. Met-Chem recommended investigating the source of the difference.

Three certified standard materials of different iron grade acquired from Canmet, Toronto, Canada were used to monitor the accuracy of the assays from ALS-Chemex (Table 11.1).

Table 11.1 – Certified Reference Material Used in 2008-09

Standard ID	Certified Value (Fe %)
FER-2	27.43
FER-3	32.07
SCH-1	60.74 ± 0.09

A total of 43 analyses, equivalent to 2.9% of all the samples, are available for these standards. Although the available data for each standard is limited, a good correlation and a relatively low variability between the certified and the assayed values were obtained. However, the percentage Fe grade of the standard was slightly higher than the assays by ALS-Chemex. The same trend was observed in the COREM's results.

Still, the results were found to be acceptable to Met-Chem since most of the assays plotted within or close to the threshold of the "Mean ± 2 Standard Deviations". Consequently, Met-Chem did not consider the differences as critical to the resource estimate that was under way.

Both ALS-Chemex and COREM are ISO certified and used similar QA-QC protocols and procedures, and processed these samples with the same preparation and analytical methods.

11.2.3 Specific Gravity

No specific gravity determinations were performed during the 2008-2009 drilling program. Historical information indicates a specific gravity of 3.48 g/cm^3 for chip samples of mineralization on the north band of Deposit 1, for a grade estimated at 26% Fe total.

Met-Chem used a specific gravity of 3.20 g/cm^3 in the previous resource estimate, considering mining operations with similar grade of 25 - 27% Fe, are using a range of 3.1 to 3.3 for the BIF.

11.2.4 Security

The chain of custody was preserved by Augyva and Century by retaining control from pick-up of the core at the drill site to splitting and shipping of the samples to the laboratory.

11.2.5 Conclusions

COREM is independent, is ISO certified and used similar QA protocols, sample preparation and analytical methods to those at ALS-Chemex.

Met-Chem concluded that the sample preparation, security and analytical procedures were adequate and provided a degree of reliability sufficiently high to be used in the resources estimation.

11.3 2011-2012 Drill Program

11.3.1 Sampling, Analysis

The IOS' geologists marked the sample intervals on the core and a line along which to split it. Core splitting was done by IOS at the core storage area in Radisson using hydraulic splitters at the beginning of the program, and subsequently a diamond blade saw. One half of the core was placed back in the boxes, along the original orientation and position, the other half was inserted into a plastic bag with a tag for identification. Nominal samples length was 3 m, with allowance to lengthen or shorten the sample intervals to 4.5 and 1.5 m when necessary to honour the main lithological contacts.

The bags of split core were delivered in thick polypropylene bags by IOS personnel on site to the IOS facilities in Chicoutimi for sample preparation. Sample preparation, except for the six holes also drilled to provide material for metallurgical tests (DUN-11-237B, -15, -324B, -352, -380 and -382), was contracted to the IOS Services Geoscientifiques Inc. ("IOS"), an independent sample preparation facility located in Chicoutimi.

The samples were crushed by IOS to less than 10 mm in a jaw crusher, followed by a second pass in a roll mill, reducing the size to less than 2 mm. A sub-sample of 200 to 300 g was extracted using a Jones riffle splitter and sent to ALS Chemex in Val-d'Or, Quebec, for analysis.

The holes drilled for the needs of metallurgical testwork were sampled in a different way. The six holes of HQ core drilled into Deposits 3, 4 and 6 were transported from IOS to SGS-Lakefield for sample preparation and analysis. A portion of the core was removed for head analysis, since these holes served the dual purposes of geological investigation and metallurgical testing. To that end, a thin sliver was cut off lengthwise along the core axis and cut in two, one half of which was submitted to analysis and the second half was saved as a reference sample. The rest of the core was used for the metallurgical testwork. An internal report dated January 25, 2012 and entitled "Selection of Samples for Metallurgical Testwork" was issued by Met-Chem to document the procedures and the actual sampling activities.

All the samples were submitted to XRF-Lithium Borate fusion for analysis of the major oxides, including total iron. Selected samples had determination of sulphur by Leco Furnace, Loss on Ignition (LOI %), ferrous iron titration, multi-element ICP-OES Analysis and Davis Tube tests.

A batch of 100 samples from Deposits 3, 4 and 6 were later analysed for sulphur, and used to investigate the acid-generating potential of the mineralization.

11.3.2 Davis Tube Tests

A total of 843 samples were submitted to Davis Tube tests of which 414 samples were tested at SGS Lakefield, Canada, 285 at IOS, in addition to the 144 tests performed at COREM in 2009.

A total of 285 Davis Tube Tests were completed by IOS. The operating parameters are listed in Table 11.2.

Table 11.2 – Davis Tube Tests - Operating Parameters by IOS

Parameter	Description
Granulometry	85% -325 mesh (rod mill)
Sample Weight	Core samples : 20 g Blanks : 10 g Standards : 10 g
Stroke frequency	30/min
Stroke length	
Magnetic strength	4000 Gauss
Tube angle	45 degrees
Tube diameter	
Tube rotation	
Flow rate	400 mL/min
Washing Time	20 minutes

A suite of additional 414 samples were sent to SGS Lakefield in order to have some of the tests done by an accredited laboratory. These samples had previously been sent to Actlabs but they were all rejected over some issues and doubts on the results that showed a systematic bias in the Davis Tube test results.

11.4 Laboratory Monitoring (2011-2012 Drilling)

11.4.1 QA-QC Protocol

The QA-QC protocol for the drill program of 2011-2012 was designed and applied by IOS, a consulting firm who had been mandated by Century to implement and supervise all the field activities. Met-Chem's role in the drill program consisted of acting as the QP for the resources estimate and, consequently, was limited to follow the progress of the field activities, examine and perform preliminary interpretation of the results as the work progressed, carry the QP site visit and perform spot checks on the analytical results and the assay database.

Karen Gagné, IOS'chemist, was responsible for monitoring the analytical results from the laboratory. The results were checked as they became available by maintaining control graphs and statistics up to date. Re-analysis of a lot was requested if the differences between the certified value and the analytical results of the standards exceeded the mean and two standard deviations threshold.

IOS inserted duplicate samples, as well as blank and certified standard materials into the sample stream to monitor the laboratory performance.

Since shipping of the samples to ALS-Minerals was done by individual batches for each hole separately, IOS included the following suite of QC samples as the first samples, before routine insertion of QC samples: Blank-Standard-Standard-Blank. Afterward, the following QC samples were systematically added to the core sample sequence:

- Blanks: 3%;
- Standards: 4%;
- Duplicate samples: 3%.

The actual number of control samples with valid analyses is listed in Table 11.3.

Table 11.3 – QC Samples in the 2011-2012 Program, with Valid Analytical Results

Material	Number	(%)
Blank	443	5.76
Duplicates	307	3.99
Standards	399	5.19
Total	1,149	14.94
Analyses	7,689	

11.4.2 Quality Control Samples

a) Blanks

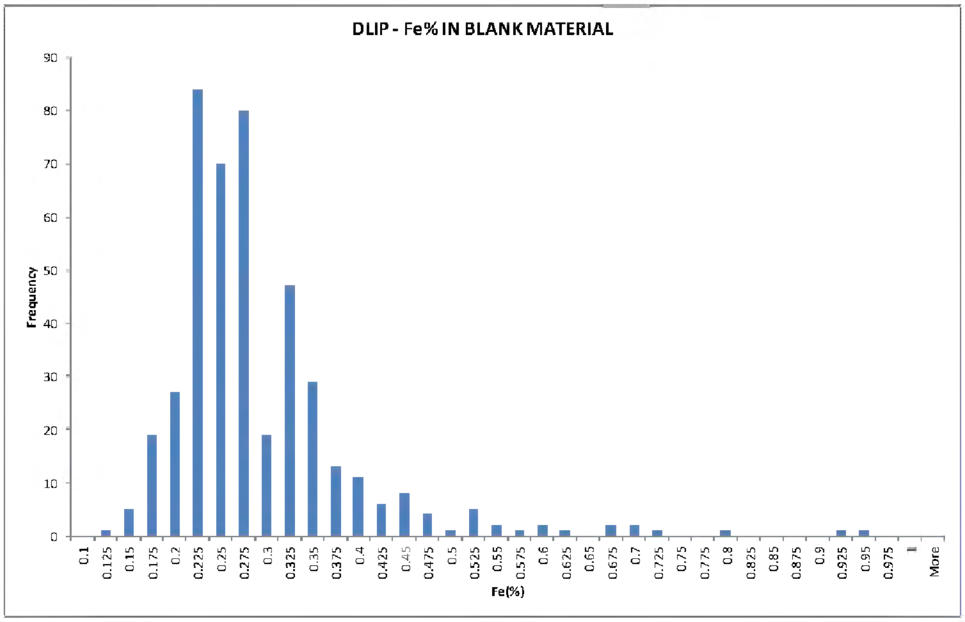
Samples identified as Blank QC samples total 518 in the database. However, some missing analyses and ten Blanks that returned high values (14.57 to 62.85% Fe_2O_3)

reduced the total to 443 valid results. The majority of the Fe% results (mode) for the blanks are around 0.25% Fe (Table 11.4 and Figure 11.1).

Table 11.4 – Fe% Content of the Blanks – Basic Statistics

Blank Samples Parameter	Fe (%)
Average	0.28
Minimum	0.12
Maximum	0.94
Number of samples	443

Figure 11.1 – Histogram Displaying the Fe% Content of the Blanks



The blank material used in the 2008-09 program was non-certified quartz material and suffered from the same problem, albeit with slightly higher values (0.24 to 0.97% Fe range, 0.42% average). Met-Chem attributed the presence of iron as an addition at the sample preparation stage.

Halite was used as Blank material at the beginning of the project and was analysed 34 times. Iron values were all below detection limits, but the lack of abrasiveness of the salt would probably not be expected to pick up any remnant iron dust from the previous samples or to generate contamination by wear on the pulverizer.

Two samples of barren rocks, commonly greywacke, felsic or mafic volcanic rocks, were systematically collected to bracket the BIF intervals. These samples are not devoid of iron but may still allow detecting sample mix-up with BIF samples, if not the sample to sample contamination at the preparation stage.

b) Certified Reference Material

Eleven standard reference materials were used by IOS to monitor the laboratory accuracy. Four of them were commercial certified standards (Table 11.5), the other ones being internal or not fully documented standards. Some were so rarely inserted into the sample stream as to provide little information.

Table 11.5 – Standard Reference Materials

Standard	Supplier	Description
TLS-1	Canmet	Cu-Ni sulphide concentrate, Sudbury (Acid Digest, AA)
Fer-2	Canmet	Griffith mine
Fer-3	Canmet	Sherman mine
Fer-4	Canmet	Sherman mine
MRI-99-08	COREM	Magnetite concentrate, Lac Doré
MRI-99-09/10/11	COREM	Lac Doré mineralization
MRIMIL06	IOS	Heavy minerals concentrate
MRI2011-01	IOS	Mineralization

c) Duplicate Samples

Quarter splits of the split or sawed core were used as QC duplicate samples. The duplicates were inserted at a rate of about 4% and submitted to the laboratory under a different number. The number attributed to the duplicate sample corresponded to a number that would be several samples down the sequence from the original.

11.4.3 Davis Tube Testing

A total of 285 Davis Tube Tests were completed by IOS. The operating parameters are listed in Table 11.2.

Quality Control of the Davis Tube Tests results was achieved by IOS by applying the following procedures:

- Visual grain size control at roll mills;
- Grain size control at rod mills (Laser dispersion method);
- Micro-XRF analysis of concentrate (test rejected if <66% Fe);
- Susceptibility meter measurement on tailings (test rejected if > 2% Mgt);
- Mass balance (test rejected if more than 1 gram loss).

A suite of additional 414 samples were sent to SGS Lakefield in order to have some of the tests done by an accredited laboratory. These samples had previously been sent to Actlabs but they were all rejected over some issues and doubts on the results.

11.4.4 Specific Gravity

The specific gravity was determined by IOS on a total of 4,967 barren and mineralized samples selected from 93 different holes, in addition to 93 duplicate determinations (Table 11.6). The water displacement method was used as a primary method and all the samples were also processed by the pycnometer technique. The average weight of the samples was 7.05 kg and ranged from 1.3 to 15.6 kg.

The samples used to calculate the average density of the mineralization are those within the envelopes of the 3D model prepared for the present resources estimate.

Table 11.6 – Density Determinations by IOS

Deposit	Mineralized Samples			Barren Samples		
	Number	Average Density	Max-Min	Number	Density	Max-Min
3	1,672	3.19		666	2.94	
4	1,097	3.18		803	2.89	
6	338	3.12		391	2.84	
Total	3,107	3.179	2.41 to 3.94	1,860	2.913	2.61 to 3.78

The quality control applied by IOS consisted of:

- Monitoring the water temperature to keep it at 23-26°C;
- Perform a determination on all the samples using the pycnometer technique;
- Duplicate determinations (4.2% of the samples).

The maximum differences between the pycnometer measurements and the displacement methods were 0.04 to -0.04. The results from the duplicate tests were very close to those of the original determination, 70 of them being within differences of 0.02 and -0.02, (Table 11.7).

Table 11.7 – Summary Results of Duplicate Density Determinations

Parameter	Original	Duplicate
Average	3.25	3.25
Standard Deviation	0.233	0.230
Correlation	0.99	
Maximum	3.61	3.61
Minimum	2.65	2.65
n=	93	93

11.5 Security

The samples were shipped from the field by IOS geologists to the IOS facilities in Chicoutimi for sample preparation or processing, and on to the laboratory, which preserved the chain of custody. IOS had some control over most of the way the samples were travelling to and there are no reasons to believe that the integrity of the samples was compromised or that the samples were tampered with.

11.6 Conclusions, Recommendations

The percentage of control samples (close to 15%) to monitor the laboratory used by IOS is significantly higher than that in most projects targeting iron formation.

Quarter splits of the split or sawed core were used as QC duplicate samples. In general, Met-Chem believes preparation of a sub-sample at the crushing stage (coarse rejects) provides a more representative duplicate sample than quartering the unevenly broken pieces of the split core, especially if the core was split rather than sawn.

A few of the in-house standards used by IOS are not well documented or showed higher variability than the commercial standard-used. Some of the standards were not used often enough to provide statistically valid information. A relatively large number of mis-identification of the standards was found in the database although, to a large degree, the errors found in the QA-QC data were corrected. The use of standard TLS-1 which is a sulphide concentrate from Sudbury is problematic when analysed by fusion discs by XRF. All but three of the analyses for this standard fell outside of the upper limit of the certified value and the other ones show a strong high bias.

The use of non-certified blank material leaves the question of possible iron contamination at the preparation stage unresolved.

Until March 2012, IOS conducted the validation of the analytical data and did request re-analysis of several batches of samples that failed the acceptability threshold.

In conclusion, Met-Chem believes the QA/QC control system, represented by a large number of control samples and several passes of validation, has not detected any significant bias or error during analysis that has not been corrected. Met-Chem believes that the laboratory performance was adequately monitored and that the analytical results of the core samples are sufficiently reliable to serve in a resources estimate.

12.0 DATA VERIFICATION

12.1 Previous Verifications

12.1.1 Geologica

Prior to Met-Chem's visit, Geologica Groupe Conseil had visited Deposit 1 of the DLIP in 2008, and produced the first NI 43-101 Technical Report on the Duncan Lake Property. Eight grab samples were collected and analysed for iron by Atomic Absorption and for gold by Fire Assay. Iron analyses ranged from 25.47 to 33.18% Fe.

12.2 Verification by Met-Chem

12.2.1 Previous Site Visit (2009)

A previous site visit from a QP was made from September 14 to 16, 2009 by Mr. Raynald Jean, Geo., as Senior Geologist of Met-Chem at the time. The visit occurred after a major drilling program had been completed, and was carried out in preparation for a resource estimate documented in a Technical Report entitled "Technical Report NI 43-101 on Mineral Resources of the Duncan Lake Iron Ore Project", dated December 2010 by R. Jean, Geo.

Mr. Raynald Jean found evidence in the field of four drill sites out of the five holes drilled in 1973 in Deposit 6, but only one collar (73-01 casing) was positively identified and surveyed. The 1973 holes were included in the database and the data used for the resource estimate, at the time.

A total of sixteen recent drill sites over Deposits 1 to 5 were visited and the GPS readings by Met-Chem were very close to the coordinates entered into the database.

During the site visit, the drill core of five holes was examined. The description and the contacts of the lithological units and of the sample intervals were found to have been correctly recorded.

Five independent check samples of quartered core, or of pieces of core picked at regular intervals, were collected. Differences of up to 4.36% Fe were observed between the original and duplicate samples, which were attributed to the lack of representativity that is provided by core quartered from originally split core.

Two holes were drilled in 2009 to validate previous drilling information (DUN09-43 near 73-01 and DUN09-44 near 73-09) but the results were inconclusive. The distances between the pairs of holes are estimated at 36 and 43 m which are too high to be used for validation of grades.

The core from the 1973 diamond drill campaign has never been found. The core from the drilling campaign of 2008-2009 was stored in racks with a roof at a facility located close to Radisson.

The laboratories used for the sample preparation and analysis were not visited. Both ALS-Chemex and COREM laboratories are independent, are ISO certified and apply strict internal QA-QC protocols.

The main concern raised by R. Jean about the DLIP was related to the lack of uniformity in the lithological descriptions by the different geologists involved in the core logging.

However, Met-Chem was of the opinion that the assay results were representative of the Deposits. Several phases of validation of the database were completed by Met-Chem. Met-Chem concluded that the data was satisfactorily reliable to be used in the resources estimate.

12.2.2 Site visit by Met-Chem (2011)

a) Introduction

Met-Chem's geologists, Yves. A. Buro, Eng. and M.-A. Brulotte, Geo., were on site on August 9 to 12, 2011 in order to complete a personal inspection of the DLIP property for a QP visit, as required by the NI 43-101 Standards of Disclosure for Mineral Projects. The visit also served to insure that the drill density and pattern were adequate for Met-Chem to estimate resources, and that the best practices guidelines were followed.

An internal report entitled "Field Visit Report" dated August 15, 2011 was prepared by Y. A. Buro and M. A. Brulotte to document the observations and recommendations resulting from the visit by Met-Chem. The main points of this report are summarized in this section.

b) Activities

The exploration and drilling aspects of the Project were reviewed with IOS field geologists and Allan Wenlong Gan, Geo., Century's Exploration Manager.

A trip was taken to the field to examine outcrops and check the position of old and active drill sites in Deposits 3 and 6. Nine hole collars were checked with a hand-held GPS and were well within the accuracy of the instrument when compared against their coordinates entered into the database.

Part of the core drilled in 2008-09 was spilled or mixed-up when the core racks shifted on instable ground. Later on, the core had to be hastily removed from the shed, placed on pallets. The 2011-2012 core was placed outside, in an open yard, in covered core racks.

The core from selected holes was reviewed with the IOS' geologists and compared against the drill logs in Geotic format, as well as the temporary database on site.

The main recommendations brought up by the site visit are indicated below.

c) Main Recommendations

- Distinguish fracture zones from faults and provide qualitative estimate of impact (major, minor fault) based on presence of gouge, fault breccia;
- Distinguish types of felsic intrusive rocks that may be related with different ages of injection (pre-, post-mineralization);
- End the procedure of adjusting sample intervals to have rounded depth numbers, which is unacceptable as it dilutes the samples of iron formation at the contacts with waste or reduces the length of the iron formation by the amount left in the waste samples bracketing the mineralization; use the flexibility of varying the systematic 3-m sample length between 1.5 and 4.5 m to cut the samples at the exact contacts of the lithological units. This practice was stopped after the QP visit and corrective action was later taken by re-sampling;
- Have the contacts of the lithological units and of the samples reviewed by a peer before splitting/sawing the core.

12.2.3 Independent Check Sampling

a) Head Analyses

Met-Chem selected 50 samples covering a fair range of iron contents and depths in Deposits 3, 4 and 6 to be re-analysed and to serve as independent check samples. The rejects were used rather than re-splitting half core, which Met-Chem does not consider as a very good duplicate sample, especially if the half core has been split rather than sawn.

The analytical results for Total Fe and SiO_2 in Met-Chem's independent check samples are provided in Table 12.1 with the results for the other oxides, sulphur and Loss on Ignition provided under Appendix B.

Table 12.1 – Met-Chem's Independent Check Samples – Analytical Results for Total Fe and SiO_2.

Deposit	Hole ID	From (m)	To (m)	Sample ID	Fe Original (%)	Fe Duplicate (%)	SiO_2 Original (%)	SiO_2 Duplicate (%)
3	DUN-11-232	62.00	64.70	82810386	34.57	35.6	45.7	46.3
3	DUN-11-232	99.00	102.00	82810400	37.95	37.6	42.9	42.7
3	DUN-11-232	132.00	135.00	82810411	39.32	39.2	41.3	41.0
3	DUN-11-232	165.00	168.00	82810423	37.89	37.7	43.8	43.8
3	DUN-11-232	204.00	207.00	82810438	37.27	37.8	42.8	42.5
3	DUN-11-242	66.00	69.00	82810588	25.34	25.2	50.0	50.3
3	DUN-11-242	81.00	84.00	82810593	29.23	29.0	46.6	46.9
3	DUN-11-242	123.00	126.00	82810603	22.45	21.6	49.6	49.1
3	DUN-11-242	144.00	147.00	82810611	25.73	25.1	52.8	52.7
3	DUN-11-41	243.00	246.00	443457	24.68	24.2	48.8	48.7
3	DUN-11-41	309.00	312.00	443473	31.99	32.1	46.4	46.7
3	DUN-11-41	324.00	327.20	443479	30.06	29.9	46.3	47.1
3	DUN-11-44	97.10	100.10	445944	24.44	24.6	47.7	47.4
3	DUN-11-44	129.00	132.00	445951	31.55	31.3	46.5	46.7
3	DUN-11-44	174.00	177.00	445961	26.91	26.9	52.9	53.2
3	DUN-11-44	195.00	198.00	445968	31.13	31.0	46.4	47.2
3	DUN-11-44	255.00	258.00	445978	32.12	31.7	45.4	44.9
4	DUN-11-214	222.10	225.70	82813136	23.65	23.6	53.3	53.2

Deposit	Hole ID	From (m)	To (m)	Sample ID	Fe Original (%)	Fe Duplicate (%)	SiO_2 Original (%)	SiO_2 Duplicate (%)
4	DUN-11-214	308.50	311.50	82813160	27.62	27.0	51.1	51.9
4	DUN-11-214	323.80	326.80	82813166	23.89	23.9	54.3	53.0
4	DUN-11-214	341.80	344.80	82813173	27.23	27.2	51.9	51.9
4	DUN-11-215	28.45	31.45	82813237	28.10	28.2	52.6	53.0
4	DUN-11-215	69.00	73.00	82813256	25.80	25.3	50.5	50.9
4	DUN-11-215	85.95	89.00	82813262	22.54	22.7	53.2	53.1
4	DUN-11-215	100.00	103.00	82813267	22.26	22.1	54.3	53.7
4	DUN-11-215	115.00	117.00	82813273	24.94	25.0	50.4	50.6
4	DUN-11-215	127.15	130.65	82813277	22.08	22.0	54.7	54.6
4	DUN-11-320	134.60	137.70	82813344	31.99	31.8	49.7	49.3
4	DUN-11-320	144.80	148.00	82813349	29.29	29.5	49.4	49.3
4	DUN-11-320	162.25	165.25	82813356	28.38	28.3	50.9	50.4
4	DUN-11-320	171.25	174.25	82813359	26.98	27.3	53.3	53.0
4	DUN-11-320	201.60	204.60	82813371	24.19	24.3	52.5	52.2
6	DUN-11-75	192.00	195.00	445855	29.08	29.2	49.0	50.0
6	DUN-11-75	198.00	200.00	445857	32.72	33.2	46.0	47.2
6	DUN-11-75	204.60	206.60	445861	25.17	25.9	56.3	56.6
6	DUN-11-75	223.00	226.00	445864	30.68	31.1	46.2	46.6
6	DUN-11-75	235.00	238.00	445868	29.39	29.4	48.6	49.9

Deposit	Hole ID	From (m)	To (m)	Sample ID	Fe Original (%)	Fe Duplicate (%)	SiO_2 Original (%)	SiO_2 Duplicate (%)
6	DUN-11-75	268.00	271.00	445881	28.10	27.8	51.7	51.3
6	DUN-11-75	277.00	280.00	445884	23.45	23.3	55.9	55.8
6	DUN-11-75	324.00	327.00	445889	36.89	36.7	43.1	44.4
6	DUN-11-75	333.20	335.90	445894	33.64	33.6	42.2	44.0
6	DUN-11-75	339.20	340.90	445896	33.62	33.0	47.4	48.7

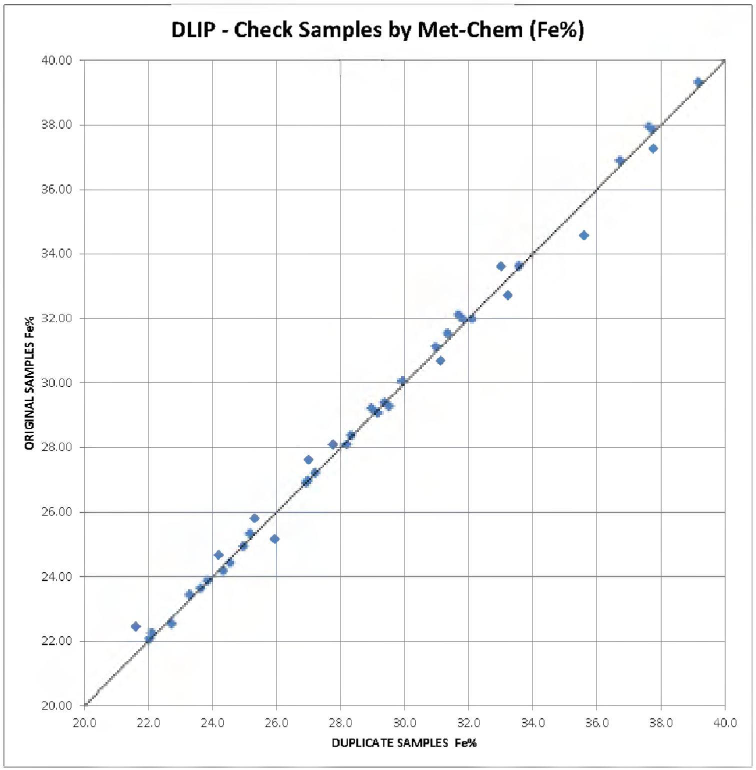
The check samples were shipped without the five QC samples (2 Duplicate samples, 2 Standards and 1 Blank) that were requested by Met-Chem. In addition, several requested samples could not be located or had insufficient material to perform the complete set of analyses.

It is fortunate that the duplicate analyses did not show any bias, which made the absence of Standards analysed with this batch of samples less crucial. The duplicate samples show high correlation and low variability relative to the original samples. The basic statistical parameters for head total iron and silica and the ranges of differences between the original and duplicate analyses for the individual pairs are listed in Table 12.2 and illustrated in Figure 12.1.

Table 12.2 – Met-Chem's Check Samples - Basic Statistics on Head Analyses of Original and Duplicate Samples

Parameters	Check Samples Total Fe (%)	Original Samples Total Fe (%)	Check Samples SiO_2 (%)	Original Samples SiO_2 (%)
n=	41	41	41	41
Average	28.94	28.99	49.25	49.06
Standard Deviation	4.85	4.80	3.79	3.99
Correlation	0.9973		0.988	
Range of Differences in the Pairs	-0.84 to + 0.78		-1.3 to + 1.9	

Figure 12.1 – Met-Chem's Check Samples - Head Analyses of Original and Duplicate Samples



Two samples were selected by the laboratory for duplicate analysis. The duplicate analyses yielded Tot Fe and silica values almost identical to the original results (Table 12.3).

Table 12.3 – Met-Chem's Check Samples – Laboratory Duplicate Samples

Sample ID		Original	Duplicate (Met-Chem Samples)	Laboratory Duplicate (Re-assay)
445968	Total Fe%	31.62	30.99	31.13
445864	Total Fe%	30.87	31.13	31.34
445968	SiO_2	46.14	47.2	47.3
445864	SiO_2	46.38	46.6	46.8

b) Davis Tube Tests

The Davis Tube tests performed on the check samples selected by Met-Chem showed high repeatability in the weight recoveries of the magnetic concentrate (Table 12.4 and Figure 12.2).

Table 12.4 – Met-Chem Check Samples - Davis Tube Tests Results

Sample Description					Original Samples			Duplicate Sample			
Deposit	DDH ID	From (m)	To (m)	Sample ID	Feed (g)	Magnetic (g)	DTWR (%)	Feed (g)	Magnetic (g)	DTWR (%)	Difference (%)
3	DUN-11-232	99.00	102.00	82810400	20.05	9.89	49.33	20.06	10.01	49.90	-0.57
3	DUN-11-232	132.00	135.00	82810411	20.09	10.42	51.87	20.04	10.33	51.55	0.32
3	DUN-11-232	165.00	168.00	82810423	20.05	9.69	48.33	20.07	9.72	48.43	-0.10
3	DUN-11-232	204.00	207.00	82810438	20.08	9.85	49.05	20.06	9.78	48.75	0.30
3	DUN-11-242	66.00	69.00	82810588	20.07	5.30	26.41	20.03	5.12	25.56	0.85
3	DUN-11-242	81.00	84.00	82810593	20.07	7.58	37.77	20.04	7.24	36.13	1.64
3	DUN-11-242	123.00	126.00	82810603	20.09	3.53	17.57	20.01	3.45	17.24	0.33
3	DUN-11-41	243.00	246.00	443457	20.05	5.95	29.68	20.03	5.84	29.16	0.52
3	DUN-11-41	309.00	312.00	443473	20.05	8.09	40.35	20.05	8.18	40.80	-0.45
3	DUN-11-41	324.00	327.20	443479	20.05	7.35	36.66	20.04	7.36	36.73	-0.07
3	DUN-11-44	97.10	100.10	445944	20.09	4.87	24.24	20.04	4.94	24.65	-0.41
3	DUN-11-44	129.00	132.00	445951	20.09	8.17	40.67	20.07	8.06	40.16	0.51
3	DUN-11-44	174.00	177.00	445961	20.06	4.44	22.13	20.02	4.50	22.48	-0.34
3	DUN-11-44	195.00	198.00	445968	20.03	8.07	40.29	20.09	7.98	39.72	0.57
3	DUN-11-44	255.00	258.00	445978	20.04	8.27	41.27	20.09	8.32	41.41	-0.15
4	DUN-11-214	222.10	225.70	82813136	20.05	4.06	20.25	20.12	4.10	20.38	-0.13
4	DUN-11-214	308.50	311.50	82813160	20.02	6.52	32.57	20.11	6.58	32.72	-0.15
4	DUN-11-214	323.80	326.80	82813166	20.03	4.09	20.42	20.14	3.99	19.81	0.61
4	DUN-11-214	341.80	344.80	82813173	20.06	6.79	33.85	20.03	6.70	33.45	0.40
4	DUN-11-215	28.45	31.45	82813237	20.09	6.75	33.60	20.03	6.75	33.70	-0.10
4	DUN-11-215	69.00	73.00	82813256	20.09	6.29	31.31	20.04	6.13	30.59	0.72
4	DUN-11-215	85.95	89.00	82813262	20.05	4.86	24.24	20.03	4.93	24.61	-0.37
4	DUN-11-215	100.00	103.00	82813267	20.11	4.54	22.58	20.08	4.50	22.41	0.17
4	DUN-11-215	115.00	117.00	82813273	20.06	5.69	28.36	20.06	5.55	27.67	0.70
4	DUN-11-215	127.15	130.65	82813277	20.06	4.70	23.43	20.05	4.63	23.09	0.34
4	DUN-11-320	134.60	137.70	82813344	20.30	8.29	40.83	20.08	8.00	39.84	-0.99
4	DUN-11-320	144.80	148.00	82813349	20.20	7.46	36.93	20.08	7.18	35.76	-1.17
4	DUN-11-320	162.25	165.25	82813356	20.08	7.21	35.91	20.04	7.20	35.93	-0.02
4	DUN-11-320	201.60	204.60	82813371	20.09	5.80	28.87	20.03	5.81	29.01	-0.14
6	DUN-11-75	192.00	195.00	445855	20.08	5.89	29.33	20.09	6.02	29.97	-0.63
6	DUN-11-75	198.00	200.00	445857	20.15	7.94	39.40	20.02	8.10	40.46	-1.06
6	DUN-11-75	204.60	206.60	445861	20.02	6.40	31.97	20.04	6.63	33.08	-1.12
6	DUN-11-75	223.00	226.00	445864	20.11	7.75	38.54	20.08	7.94	39.54	-1.00
6	DUN-11-75	235.00	238.00	445868	20.05	7.43	37.06	20.02	7.91	39.51	-2.45
6	DUN-11-75	268.00	271.00	445881	20.03	2.67	13.33	20.08	2.93	14.59	-1.26
6	DUN-11-75	277.00	280.00	445884	20.10	3.43	17.06	20.06	3.40	16.95	0.12
6	DUN-11-75	324.00	327.00	445889	20.06	10.61	52.89	20.04	10.76	53.69	-0.80
6	DUN-11-75	333.20	335.90	445894	20.08	8.00	39.84	20.06	7.97	39.73	0.11
6	DUN-11-75	339.20	340.90	445896	20.14	9.50	47.17	20.05	9.41	46.93	0.24

A repeat test performed by the laboratory on two submitted samples yielded very similar results for each of the two pairs (Table 12.5).

Figure 12.2 – Met-Chem Check Samples - Davis Tube Tests Results - Weight Recovery

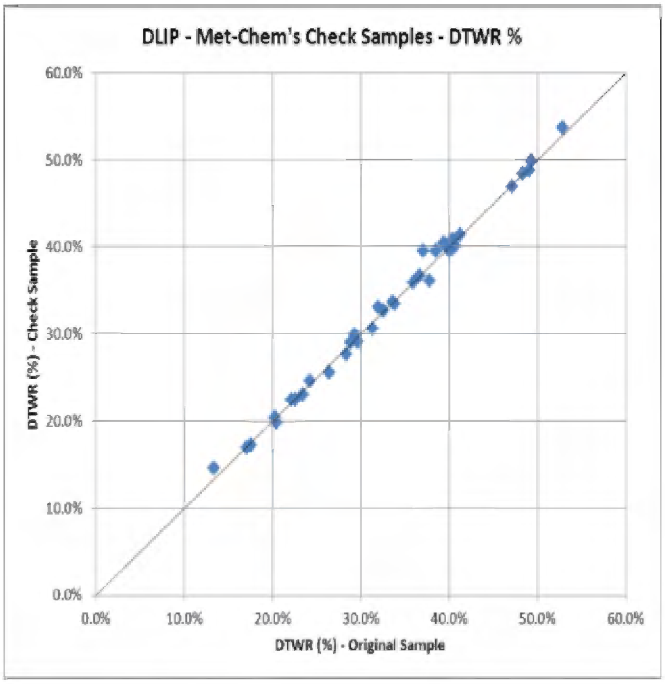


Table 12.5 – Met-Chem Check Samples - Davis Tube Tests, Laboratory Duplicate Analyses

	Original Sample			Duplicate Sample			Laboratory Duplicate		
Sample ID	Feed (g)	Magnetic (g)	DTWR (%)	Feed (g)	Magnetic (g)	DTWR (%)	Feed (g)	Magnetic (g)	DTWR (%)
445864	20.11	7.75	38.54	20.08	7.94	39.54	20.05	7.93	39.55
445968	20.03	8.07	40.29	20.09	7.98	39.72	20.05	8.04	40.10

12.2.4 Database Validation by Met-Chem

Most data were transmitted by IOS to Met-Chem in Geotic format and part of the results was in Excel spreadsheets. IOS checked the data before sending them and Met-Chem did additional verifications of the master database using formulas in Excel sheets, the software GeoticLog and MineSight.

Most errors were corrected with the help of IOS, R. Gauthier of Roche and Zhihuan Wan, the Century's Project Geologist who took over responsibility for the project after IOS's contract was ended.

The main problem Met-Chem found with the database was in the use of combinations of several rock codes for the main lithological units, which cannot be handled or plotted on sections by mining software. Lack of uniformity in the codes attributed to some of the rocks by the logging geologists was observed.

The un-sampled portions of barren rocks left between the sampled intervals were filled in if they represented 15 m in core length or less. This was done to facilitate interpretation and 3D modeling. Likewise, the samples that had mixed BIF and barren material at the time the geologists were using round depth numbers for the contacts of the samples were re-sampled.

However, Met-Chem believes that most of the errors were fixed and the uncertain data were left out and do not represent a significant percentage of the total. A comprehensive QA-QC system applied to the laboratory monitoring and to the sample preparation and Davis Tube tests carried out by IOS add to the reliability of the data. Consequently, Met-Chem believes the data are of sufficiently high quality to be used for a resource estimate and a preliminary economic assessment of the DLIP. Additional development work will be required to upgrade the knowledge of the project to advance the DLIP to the next study stage. Details on the additional work are given in Section 13.5 of this report.

12.2.5 Verifications of the QA-QC Implemented by IOS

a) General

Met-Chem examined the QA-QC system applied by IOS and completed spot checks of the results obtained by the control samples inserted by IOS into the sample stream. Some of the checks were done shortly before all the sample results had been received. All the QC samples were added to the sample stream at the IOS facilities in Chicoutimi.

The analytical results entered by IOS into the master database were validated by IOS but Met-Chem did some additional verifications.

b) Blank Samples

The majority of the Blank samples contain 0.25% Fe and range from 0.12% to 0.94% Fe, as discussed under Section 11.4 of this report.

Met-Chem attempted to determine whether the low levels of iron present in all the blank samples correspond to background values of the barren quartz or is introduced at the sample preparation stage. Analysis of a suite of blank material used by IOS after pulverisation in Tungsten Carbide pots, in order to ensure no iron contamination is introduced, was requested by Met-Chem. Ten samples were prepared by ALS Chemex in Val-d'Or and shipped to SGS Lakefield for analysis. The results from these samples strongly support the view that the blank material does contain some iron (Table 12.6).

Table 12.6 – Fe% Determination in Blank Samples After Pulverization in Tungsten Carbide Pot

Blank Sample ID	Check Sample (Fe %)	Original Sample (Fe %)
82816936	0.05	0.19
82816957	0.11	0.26
82816981	0.03	0.21
82817115	0.01	0.29
82817146	0.04	0.17
82817308	0.18	0.36
82817724	5.88	0.57
82817956	0.1	0.27
82818027	0.11	0.32
82818078	0.02	0.22

The fact that the level of Fe content in the blanks analysed with the core samples of 2011-2012 seems to be slightly but systematically higher than the level suggested by the ten samples that have been processed with special grinding apparatus points to the possibility of introduction of some Fe into the sample at the pulverization stage. The fact that up to 0.2% Fe can be added to the sample by the pulverization process has been documented on samples from other iron projects.

Met-Chem believes the blank samples used by IOS contain a low background level of iron and that some iron is added at the sample preparation stage.

c) Standard Reference Materials

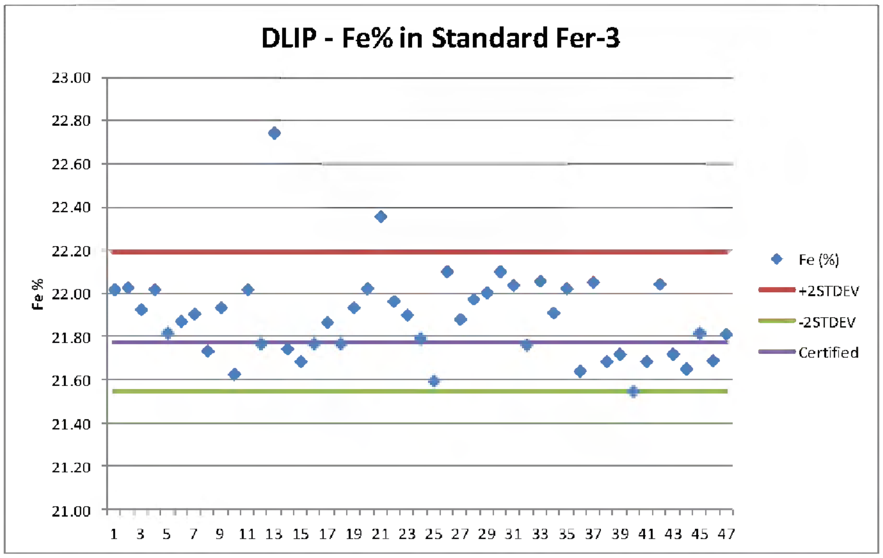
IOS used up to 11 standard reference materials, either commercial certified reference material or internal standards. Met-Chem found a significant number of mis-labeled standards, although the label on most of them could be arbitrarily re-assigned by their Fe content (Table 12.7).

Table 12.7 – Comments on Standard Reference Material Used by IOS (All Values in Fe %)

Standard ID	Number of Samples	Average	Minimum	Maximum	Declared Value	Value	Comments
Fer-1	1				52.55		
Fer-3	47	21.89	21.55	22.74	21.77		
Fer-3	1					41.82	mislabeled
Fer-3	1					0.11	mislabeled
Fer-4	14	28.8	27.59	31.15	27.92		
99-09	1					24.89	mislabeled
99-09	13	43.87	43.26	44.68			mislabeled
99-09??	1					24.07	mislabeled
99-09#26537	25	25.19	24.83	25.61			mislabeled
99-09#26537	45	43.72	42.65	44.77			mislabeled
99-10	1					40.19	
99-10#189391	68	39.72	38.53	40.99			
99-11#26537	60	43.77	42.49	46.28			
PFeMRI	1				25.26		mislabeled
PFEMRI11	9	15.49	7.73	26.40			mislabeled
PFeMRI11	55	27.94	24.59	29.46			mislabeled
MRIMIL06	31	18.25	16.56	21.13			
TLS-1	25	10.78	10.55	12.35	10.51 ±0.15		sulphide concentrate
TOTAL	399						
Standard	5		0.24	0.34			mislabeled
Undetermined	3		24.70	25.32			mislabeled

The results from the commercial certified reference material Fer-3 is presented as an example of Met-Chem's examination of the performance of the standards (Figure 12.3). The graph shows that all but two results are within the acceptable limits, with a possible slight high bias in the first samples and a low bias in the last ones.

Figure 12.3 – Plot of the Fe Percent in Standard Fe-3



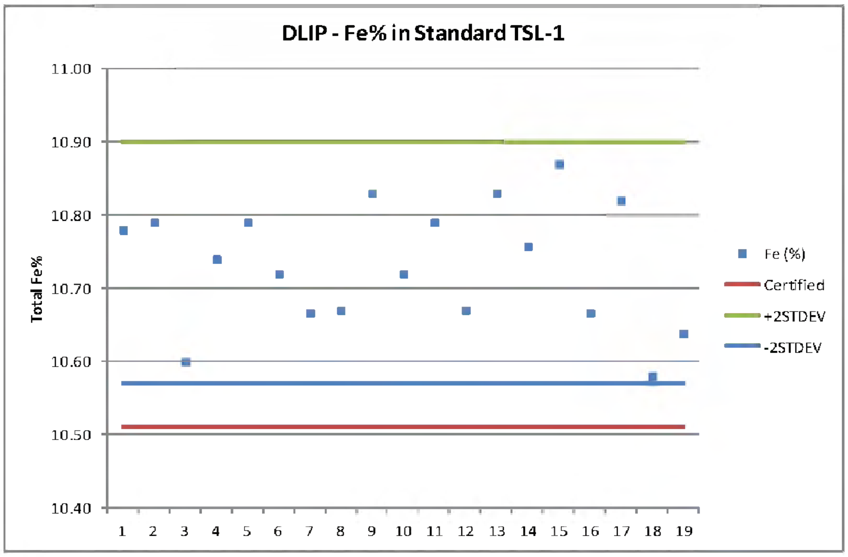
The commercial standard TLS-1 is a sulphide concentrate, which is a poor choice for an iron standard because its high sulphur content is problematic in XRF analysis on fusion discs. The analytical results for these standards are precise but lack accuracy (Figure 12.4).

The internal Reference Material PFEMRI11 and MRIMIL06 prepared by IOS show the highest variability of all the standards. The latter is heavy minerals concentrate with added pyrochlore to monitor niobium-tantalum.

The standards for which less than 25 or 30 analyses are available have restricted usefulness.

Clearly, the identification of the standards by the geologists could have been more diligent. However, Met-Chem believes the large number of valid analytical data provided by the standards has adequately monitored the laboratory performance. The analytical results for the standards examined by Met-Chem show that they generally stayed within the acceptable limits of the mean and two standard deviations.

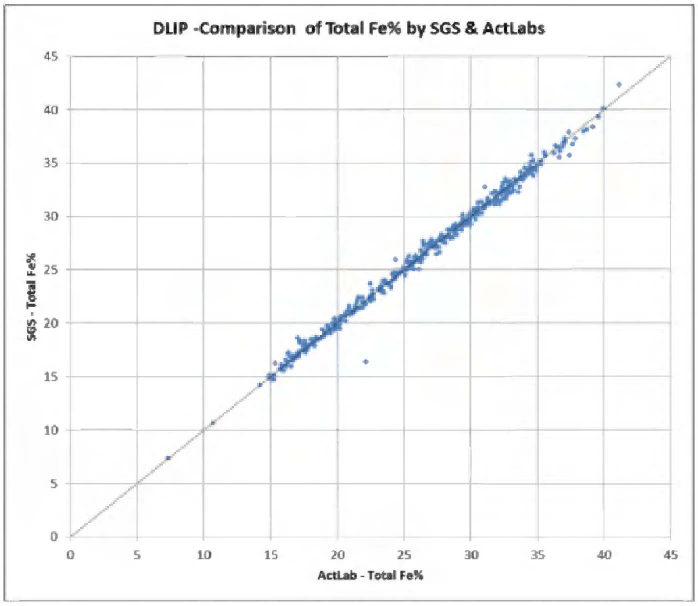
Figure 12.4 – Plot of the Fe Percent in Standard TSL-1



12.2.6 Secondary Laboratory

A total of 394 samples from Deposits 3, 4 and 6 originally analysed by Activation Laboratories Ltd. (Actlabs), Ancaster, Ontario, were re-analysed by SGS. A high correlation between the two sets of data is apparent, except for one sample. The correlation coefficient is 0.996 and the absolute difference in the individual pairs over the average was 1.01% for the median with a maximum of 8.59% (except for one value at 30.3%), which is excellent.

Figure 12.5 – Comparison of Total Fe % by SGS and Actlabs



These samples were also submitted to Davis Tube testing, but they were completely ignored since some issues as to the reliability of the results were raised, and remain unresolved.

12.2.7 Density Determinations by IOS

Met-Chem did not request a series of samples for independent density determinations, since a fairly large number of samples had been processed by IOS using two methods on each sample and validating the results by duplicate determinations on close to 2% of the samples.

A few errors were found in the IOS database of the samples used for the density determinations, and the affected samples were eliminated. Met-Chem estimates that these errors were in such low percentage as to have no impact on the validity of the results. A density factor of 3.2 g/cm^3 used in the previous resources estimates was retained in the present calculations to convert volumes into tonnes.

Met-Chem believes the density determinations were carried out by IOS with extreme care and serious validation and are reliable. Consequently, Met-Chem has used these results, which confirmed the original parameter used to convert volumes into tonnes, in the resource estimation of 2012.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

This section of the report discusses the metallurgical testing conducted at SGS Lakefield, Ontario on samples from the Duncan Lake mineralization. The DLIP consists of six (6) different bodies defined here as deposits but only three deposits (Deposit 3, Deposit 4 and Deposit 6) were investigated in the Test Program.

The material for metallurgical testwork is from two holes totaling 2,349 m of HQ core that were drilled in 2011 into each of the Deposits 3, 4 and 6.

The core of the complete holes was transported to the IOS facilities in Chicoutimi (Table 13.1). Rénald Gauthier, P. Geol., D.E.S.S., Roche Ltd Consulting Group, and Yves A. Buro, Eng., Met-Chem, selected and marked the core intervals to supply to the metallurgical laboratory during the days of January 5 to 7, 2012 in Chicoutimi. Subsequently, the core samples were shipped to SGS Lakefield, Ontario to have a sliver cut for the head analyses and the rest of the material used in the testwork. These holes were drilled in HQ-sized core in order to generate sufficient material for metallurgical testing and head assays.

Table 13.1 – Diamond Drill Holes for Metallurgical Testwork Purposes

Deposit	Drill Hole	Total Depth (m)
3	DUN-11-352	456.0
	DUN-11-237B	438.0
4	DUN-11-315	438.0
	DUN-11-324B	237.0
6	DUN-11-380	405.0
	DUN-11-382	375.0
Total		2,349

13.1 Introduction

SGS Lakefield, Ontario was mandated with the following tasks to be completed on samples from the Duncan Lake deposit:

- Task 1: Sample Preparation;
- Task 2: Head assay / Whole Rock Analysis suite (WRA);
- Task 3: JKTech Drop-Weight and SMC Tests;
- Task 4: Bond Low-energy Impact Test;
- Task 5: Bond Rod Mill Grindability Test;
- Task 6: Bond Ball Mill Grindability Test;
- Task 7: Bond Abrasion Test;
- Task 8: Coarse Cobbing;
- Task 9: Davis Tube Testing;

- Task 10: Sulfur Analysis.

The tasks were to be done to generate information necessary for the design of crushers, SAG Mills and Ball Mills. Task 8 focused on the potential of rejecting bulk of the gangue minerals at the grinding stage. Task 9 focused mainly on assessing the beneficiation characteristics of the mineralization. Task 10 focused on the analysis of the sulfur content of the concentrate and the tailings. Following the results obtained from Task 10, the Unité de recherche et de service en technologie minérale de l'Abitibi-Témiscamingue ("URSTM") was mandated with the following task to be completed on samples from the Duncan Lake deposit:

- Task 11: Acid-Base Accounting static tests ("ABA").

Task 11 focused on the potential for acid generation of the tailings and waste from core samples.

13.2 Previous Testwork Summary

Metallurgical test work was conducted at COREM, ActLab and IOS Laboratories. Results can be found in various reports submitted by the respective laboratories and listed in Section 27 of the Report.

13.3 Testwork Results

13.3.1 Sample Preparation

a) Sample Reception

A total of 27 skids consisting of multiple core boxes were received in two shipments on February 16 and on February 17 of 2012. The two shipments were given the SGS receipt numbers 0235-FEB12 (24 skids) and 0257-FEB12 (3 skids).

The core boxes contained full HQ core from six drill holes, two each from three areas designated as Deposit 3, Deposit 4 and Deposit 6. Core from the following drill holes was received:

- 237B and 352 (Deposit 3);
- 315 and 324B (Deposit 4);
- 380 and 382 (Deposit 6).

The 'as received' core had already been marked by Mr. Renald Gauthier of Roche and Mr. Yves Buro of Met-Chem, along virtually its entire length of ~2,000 meters, in order to facilitate sample preparation at the SGS facility. It was split for the most part into consecutive, 3-meter intervals with the intention of removing a thin sliver along each interval using an industrial saw setup and subsequently shipping the intervals to ALS laboratories for assays. Certain sections of the core were not marked as they were not deemed necessary to be tested.

In addition to the intervals marked for assays, smaller intervals were also marked that were distributed throughout the entire length of the core. These intervals that mostly coincided with the assay intervals, were to be saw-cut after the removal of the slivers, sorted, combined as instructed and subsequently submitted for grindability testwork at SGS. Adjacent duplicate intervals designated for future shipment to China were also marked for cutting.

The core was marked according to the following classification system:

- Deposit ID code (numbers 3, 4 or 6);
- Testwork ID code (numbers 1 for DWT, 2 for CWI, 3 for SMC, 4 for RWI, 5 for BWI). The combinations of tests 1 to 5 were predominantly marked immediately next to each other;
- A or B designation depending on whether the sample is to be tested at SGS (A) or shipped to China (B). A and B samples were adjacent;
- Lithological code (BIF, LEAN, SED, GR, VOLC);
- Suffix (Comp if material is designated to be part of a composite).

b) Core-cutting of Assay Samples

The sample preparation steps are outlined below:

- Thin slivers were first removed from the designated marked sections of the core.
- The marked sections from the different formations designated to make up each composite were collected, weighed, blended and submitted for the JK drop-weight test which requires a top size of 2.5" (63 mm). Each DWT sample weighed ~70 kg. A 5-kg sub-sample was removed during the DWT preparation for the Bond abrasion test, which requires -3/4"/+1/2" material.
- Similarly, the marked sections intended to make up each composite were collected and weighed but the CWI composites were made based on the weight proportions of their components as this test did not require the entire amount of sample collected but only ~20 rocks that satisfied the -3"/+2" requirement. All the samples were shipped to Phillips Enterprises LLC in Colorado for testing.
- The sections designated for the SMC, RWI and BWI tests were blended together. They were first crushed to nominal 1.25" and approximately 20-30 kg was removed for the SMC test (conducted on -22.4/+19.0 mm rocks). Subsequently, the sample was stage-crushed to ½" and 15-20 kg was removed for the RWI test. The sample was then stage-crushed to minus 6 mesh. One 10- kg charge was riffled out for the standard Bond ball mill grindability test and additional 5-kg or 10-kg charges were removed for the dry cobbing tests. Head assay splits (100 grams) were also riffled out during this stage.

- All the duplicate samples for shipment to China were saw-cut at the same time as the SGS samples. They were handled similarly to the SGS samples and individually stored in pails and/or drums.

c) Core-cutting for Grindability Testwork

- Deposit 3 material originated from drill holes 237B and 352;
- Deposit 4 material originated from drill holes 315 and 324B;
- Deposit 6 material originated from drill holes 380 and 382.

Samples from Deposit 3, Deposit 4 and Deposit 6 were sent for testing. The details of mass used for each tests is shown in Table 13.2. For each deposit, some samples were either not available or cancelled because selective mining will segregate these specific lithology for a particular deposit as shown in the legend. A total of seven lithologies from Deposits 3, 4 and 6 were submitted.

Table 13.2 – Core Samples for Metallurgical Tests – Quantities Required

		Block 3								
Test ID	Test Description	Lithologies							Kg Required per Test	Spec per Test
		Basalt	Granite	Lean Iron Formation	Iron Formation	Sediment	Composite	Test Qty		
1	JK Drop weight test						31A; 31B	1	80	0.5 to 2.5 inches
2	Low energy impact						32A; 32B	1		20 rocks of 2-3 inch
3	SMC test	33AVOLC; 33BVOLC	33AGR; 33BGR	33ALEAN; 33BLEAN	33ABIF; 33BBIF	33ASED; 33BSED		5	30	Core cut in 1/4
4	Bond Rod Mill WI	34AVOLC; 34BVOLC	34AGR; 34BGR	Lack of Material	34ABIF; 34BBIF	34ASED; 34BSED		5	40	Prep at Lab
5	Bond Ball Mill WI	35AVOLC; 35BVOLC	35AGR; 35BGR	35ALEAN; 35BLEAN	35ABIF; 35BBIF	35ASED; 35BSED		5	30	Prep at Lab

		Block 4								
Test ID	Test Description	Lithologies							Kg Required per Test	Spec per Test
		Basalt	Granite	Lean Iron Formation	Iron Formation	Sediment	Composite	Test Qty		
1	JK Drop weight test						41A, 41B	1	80	0.5 to 2.5 inches
2	Low energy impact						42A, 42B	1		20 rocks of 2-3 inch
3	SMC test	Lack of Material	43AGR; 43BGR	43ALEAN; 43BLEAN	43ABIF; 43BBIF	43ASED; 43BSED		5	30	Core cut in 1/4
4	Bond Rod Mill WI	Lack of Material	44AGR; 44BGR	44ALEAN; 44BLEAN	44ABIF; 44BBIF	44ASED; 44BSED		5	40	Prep at Lab
5	Bond Ball Mill WI	Lack of Material	45AGR; 45BGR	45ALEAN; 45BLEAN	45ABIF; 45BBIF	45ASED; 45BSED		5	30	Prep at Lab

		Block 6								
Test ID	Test Description	Lithologies							Kg Required per Test	Spec per Test
		Basalt	Granite	Lean Iron Formation	Iron Formation	Sediment	Composite	Test Qty		
1	JK Drop weight test						61A; 61B	1	80	0.5 to 2.5 inches
2	Low energy impact						62A; 62B	1		20 rocks of 2-3 inch
3	SMC test	Lack of Material	63AGR; 63BGR	Lack of Material	63ABIF; 63BBIF	Lack of Material		5	30	Core cut in 1/4
4	Bond Rod Mill WI	Lack of Material	64AGR; 64BGR	Lack of Material	64ABIF; 64BBIF	Lack of Material		5	40	Prep at Lab
5	Bond Ball Mill WI	Lack of Material	65AGR; 65BGR	Lack of Material	65ABIF; 65BBIF	Lack of Material		5	30	Prep at Lab

Legend	Cancelled	Lack of Material

13.3.2 Head assay / Whole Rock Analysis suite ("WRA")

The samples were subjected to WRA, which includes the analysis of the following elements: SiO_2, Al_2O_3, Fe_2O_3, MgO, CaO, Na_2O, K_2O, TiO_2, P_2O_5, MnO, Cr_2O_3, V_2O_5 and loss on ignition ("LOI") as well as Satmagan determination to establish magnetite content. In addition, a full ICP-scan was conducted to identify any potential impurities that may exist in the samples.

Table 13.3 shows the material properties of the different samples tested.

Table 13.3 – Head Assay Results

Sample Name	Fe %	S %	SiO_2 %	Al_2O_3 %	Fe_2O_3 %	MgO %	CaO %	Na_2O %	K_2O %	TiO_2 %	P_2O_5 %	MnO %	Cr_2O_3 %	V_2O_5 %	LOI %	Sum %	Fe_3O_4 %
Block 3 BIF	33.3	0.37	45.8	2.07	47.6	1.92	1.70	0.48	1.05	0.07	0.09	0.04	< 0.01	< 0.01	-0.6	100.2	42.8
Block 3 LEAN	24.2	1.35	47.8	4.53	34.6	2.57	3.19	0.65	1.63	0.17	0.17	0.09	< 0.01	0.01	2.3	97.8	16.0
Block 3 SED	6.71	0.57	59.2	14.0	9.59	3.61	4.95	3.28	2.11	0.44	0.19	0.11	0.01	0.02	1.8	99.3	0.30
Block 4 BIF	28.3	0.31	51.1	3.04	40.5	1.94	1.66	0.62	1.52	0.10	0.15	0.04	< 0.01	< 0.01	-0.2	100.5	33.3
Block 4 LEAN	11.2	0.25	54.2	9.75	16.0	5.89	4.29	2.80	2.93	0.40	0.20	0.10	0.04	0.01	2.9	99.5	5.40
Block 6 BIF	33.4	0.51	47.6	1.55	47.7	1.72	1.43	0.20	0.67	0.06	0.13	0.06	< 0.01	0.01	-0.6	100.6	42.6
Block 6 GR	4.24	0.05	59.7	15.3	6.06	3.87	5.35	5.05	1.87	0.52	0.21	0.09	0.02	0.02	1.2	99.3	0.30

13.3.3 JK Drop-weight and SMC Tests

The JK drop-weight test was performed on three composites from Deposits 3, 4 and 6. The test results are presented in Table 13.4. The SMC test is an abbreviated version of the standard JK drop-weight test performed on rocks from a single size fraction (-22.4/+19.0 mm in this case). The SMC test was performed on the seven lithologies from Deposits 3, 4 and 6. The test results are presented also in Table 13.4. For this project, the Duncan Lake SMC samples from each deposit were calibrated against the corresponding results of full drop-weight testing.

Table 13.4 – JK Drop-Weight and SMC Test Results

Sample Name	A	b	A x b	Hardness Percentile	t_a	Hardness Percentile	DWI (kWh/m^3)	M_{ia} (kWh/t)	M_{ih} (kWh/t)	M_{ic} (kWh/t)	Relative Density
Block 3 Comp	57.3	0.62	35.5	74	0.21	94	-	-	-	-	3.33
Block 3 BIF	100	0.39	39.0	66	0.29	-	9.01	19.1	15.0	7.7	3.49
Block 3 LEAN	100	0.33	33.0	80	0.25	-	10.5	21.5	17.4	9.0	3.49
Block 3 SED	78.2	0.43	33.6	78	0.31	-	8.26	22.1	17.1	8.8	2.81
Block 4 Comp	74.6	0.49	36.6	72	0.32	76	-	-	-	-	3.18
Block 4 BIF	97.4	0.43	41.9	60	0.33	-	7.95	18.3	14.0	7.2	3.30
Block 4 LEAN	71.1	0.62	44.1	56	0.40	-	6.44	17.9	13.2	6.8	2.84
Block 6 Comp	83.5	0.45	37.6	69	0.23	91	-	-	-	-	3.20
Block 6 BIF	100	0.42	42.0	60	0.31	-	8.36	17.8	13.7	7.1	3.54
Block 6 GR	100	0.38	38.0	69	0.36	-	7.29	20.1	15.1	7.8	2.80

The t_a value reported as part of the SMC procedure is an estimate

The DWT composites were generally categorized as hard in terms of resistance to impact breakage (A x b) and very hard in terms of resistance to abrasion breakage (t_a).

The SMC lithologies covered the medium to hard range of hardness, with the majority of the samples being categorized as moderately hard to hard in terms of A x b.

13.3.4 Bond Low-energy Impact Test

The Bond low-energy impact test determines the Bond Crusher Work Index ("CWI"), which can be used to calculate power requirements for crusher sizing. The rock lithologies used in each composite are presented in Table 13.5. The test results are presented in Table 13.6.

The CWI values for the three composites ranged from 8.1 kWh/t to 9.2 kWh/t, classifying all of them as medium.

Table 13.5 – Bond Low-energy Composite Lithologies

Composite Name	Litholologies (number of rocks used)				
	Volcanic	Sediment	BIF	LIF	Granite
Block 3 Comp	1	1	18	-	1
Block 4 Comp	-	1	18	1	1
Block 6 Comp	-	-	17	1	3

Table 13.6 – Bond Low-energy Impact Test Results

Sample Name	Number of Specimens	Work Index (kWh/t)	Min. (kWh/t)	Max. (kWh/t)	S.D. (kWh/t)	Relative Density	Hardness Percentile
Block 3 Comp	21	9.2	3.4	16.8	3.0	3.06	46
Block 4 Comp	21	8.1	5.9	15.3	2.0	3.21	39
Block 6 Comp	21	8.4	4.1	12.6	2.1	3.29	41

13.3.5 Bond Rod Mill Grindability Test

The Bond rod mill grindability test was performed at 14 mesh of grind (1,180 microns) on the seven lithologies from Deposits 3, 4 and 6. The test results are summarized in Table 13.7. The BIF and Lean Iron samples were categorized as moderately soft to medium in terms of their Bond Rod mill Work Index ("RWI") values. Deposit 3 Sediment was classified as hard while Deposit 6 Granite was classified as moderately soft.

Table 13.7 – Bond Rod Mill Grindability Test Results

Sample Name	Mesh of Grind	F_{80} (μm)	P_{80} (μm)	Gram per Revolution	Work Index (kWh/t)	Hardness Percentile
Block 3 BIF	14	10,809	899	10.6	12.9	34
Block 3 LEAN	14	10,090	941	9.65	14.4	52
Block 3 SED	14	9,686	949	7.45	17.2	80
Block 4 BIF	14	10,355	926	10.2	13.6	42
Block 4 LEAN	14	10,738	941	9.84	14.0	47
Block 6 BIF	14	10,342	919	10.5	13.3	38
Block 6 GR	14	11,311	881	11.1	12.3	27

13.3.6 Bond Ball Mill Grindability Test

Bond ball mill grindability tests were additionally performed at 200 mesh of grind (75 microns) on the seven lithologies from Deposits 3, 4 and 6. The test results are summarized in Table 13.8. With Bond Ball mill Work Indices ("BWI") ranging from 9.7 to 16.1 kWh/t, the samples covered the very soft to moderately hard range of hardness.

Table 13.8 – Bond Ball Mill Grindability Test Results

Sample Name	Mesh of Grind	F_{80} (µm)	P_{80} (µm)	Gram per Revolution	Work Index (kWh/t)	Hardness Percentile
Block 3 BIF	200	2,446	59	1.92	9.7	8
Block 3 LEAN	200	2,352	60	1.46	12.2	27
Block 3 SED	200	2,148	59	1.25	14.0	46
Block 4 BIF	200	2,205	60	1.83	10.3	12
Block 4 LEAN	200	2,247	56	1.36	12.5	30
Block 6 BIF	200	2,192	64	1.49	12.6	31
Block 6 GR	200	2,357	57	1.01	16.1	70

13.3.7 Bond Abrasion Test

The Deposit 3, Deposit 4 and Deposit 6 composites were submitted for the Bond abrasion test. The results are summarized in Table 13.9. The Abrasion Index ("AI") values ranged from 0.481 to 0.622 grams, classifying all three samples as abrasive.

Table 13.9 – Bond Abrasion Test Results

Sample Name	AI (g)	Hardness Percentile
Block 3	0.526	82
Block 4	0.481	79
Block 6	0.622	88

13.3.8 Coarse Cobbing

Coarse cobbing, to reject coarse gangue, was tested with a dry magnetic drum. The results are presented in Table 13.10 for tests performed on minus 6 mesh material.

Table 13.10 – Cobber Test Summary

Sample	Test	P_{100}	Head Grade (%)				Cobber Conc. Grade (%)				Cobber Conc. Recovery (%)			
		µm	Fe[1]	SiO_2	Al_2O_3	Sat	Fe[1]	SiO_2	Al_2O_3	S	Wt	Fe	SiO_2	Sat
Block 3 BIF	LIMS-1	3,350	33.3	45.8	2.07	42.8	36.7	44.0	0.97	0.21	87.4	97.4	83.2	99.8
Block 3 LEAN	LIMS-2	3,350	24.2	47.8	4.53	16.0	32.7	43.4	1.48	1.97	46.1	62.5	41.4	97.7
Block 3 SED	LIMS-3	3,350	6.71	59.2	14.0	0.30	13.4	52.2	11.1	2.83	1.57	3.16	1.37	26.0
Block 4 BIF	LIMS-4	3,350	28.3	51.1	3.04	33.3	31.4	49.5	1.88	0.31	82.4	94.4	78.2	99.5
Block 4 LEAN	LIMS-5	3,350	11.2	54.2	9.75	5.40	26.0	50.9	3.78	0.77	20.0	45.1	18.7	92.1
Block 6 BIF	LIMS-6	3,350	33.4	47.6	1.55	42.6	35.5	44.8	1.31	0.41	89.8	98.3	82.9	99.6
Block 6 GR	LIMS-7	3,350	4.24	59.7	15.3	0.30	12.4	53.9	10.5	0.45	1.42	4.13	1.27	43.5

[1] Fe grade calculated from the Fe_2O_3 WRA result

The BIF samples achieved weight rejections between 10 and 18%, with the iron recoveries varying from 94.4 to 98.3%. The magnetite recoveries, determined from the

Satmagan reading, were all higher than 99.5%. The weight rejection of the Lean Iron samples was much higher (53.9% and 80%) although the head iron and magnetite grades were much lower for these samples. The iron recoveries were only of 62.5% and 45.1% for the Deposit 3 and Deposit 4 samples respectively. The weight and iron recoveries for the Sediment and Granite samples were very low, as expected from their low magnetite head grades (0.30%).

13.3.9 Davis Tube Testing

Samples were pulverized by SGS Lakefield, Ontario to have 95% passing -325 mesh (44 microns).

A 20 g sub-sample has been introduced into the Davis Tube as the Feed and the tube has been allowed to agitate for a period of 4 minutes. The tailings have been collected in a pail. After 4 minutes, the magnets have been interrupted and the magnetic concentrate has been collected separately. The magnetic concentrate has been weighed. The magnetic concentrate and non-magnetic tailings have been assayed to whole rock analysis ("WRA").

A total of 438 samples have been assayed and a summary is presented in Table 13.11:

Table 13.11 – Davis Tube Testing Summary

	Feed Fe_2O_3 (%)	Feed Fe (%)	Weight Recovery (%)	Magnetic Concentrate SiO_2 (%)
Deposit 1 (44 samples)	36.6	25.6	28.6	11.3
Deposit 3 (191 samples)				3.7
Deposit 4 (133 samples)				6.0
Deposit 6 (70 samples)				6.5

13.3.10Sulfur analysis

Eight samples selected to represent various observed mineralization zones were used by SGS Lakefield for a QEMSCAN study. Relatively high levels of both pyrite and pyrrhotite were reported. Pyrite content dominated in five samples, with small amounts of pyrrhotite that tendered to occur as inclusions in magnetite.

From the Davis Tube testing, 100 samples were selected in order to analyze the sulfur content. 55 samples from Deposit 3, 34 samples from Deposit 4 and 11 samples from Deposit 6 were selected. Sulfur analysis was carried out on the magnetic concentrate and on the non-magnetic tailings.

Results showed that most of the sulfur was rejected in the non-magnetic tailings, i.e. on average, 87% of the sulfur from the feed was rejected in the tailings.

The remaining of the sulfur reported to the concentrate. Most of the sulfur in the concentrate (90% to 95% of the sulfur) will be removed at the Pellet Plant during the induration process. The sulphur will oxidise to SO_2 and join the exhaust gas streams.

Based on the 100 samples assayed, the sulfur content in the tailings will be above 0.3%.

13.3.11 Acid-Base Accounting static tests ("ABA")

ABA static tests were conducted by the "Unité de recherche et de service en technologie minérale de l'Abitibi-Témiscamingue" ("URSTM") on the samples to determine their potential for acid generation ("PAG"). These tests consist of determining the balance between the sample Acid Potential ("AP" – which is related to its sulphide content) and the sample Neutralization Potential ("NP" – related to carbonates and some silicates). These tests consist of:

- Total sulphur and carbon (S_{total} and C_{total}) analyses by induction furnace (Leco type instrument);
- Sulphate ($S_{sulphate}$) analysis by acid leaching (adapted from Sobek et al., 1978);
- AP calculation (AP = 31,25x%$S_{sulphides}$, where %$S_{sulphides}$ = %S_{total} - %$S_{sulphates}$);
- NP analysis by the modified Sobek method (Lawrence and Wang, 1996);
- Net neutralization potential ("NNP") and NP/AP ratio calculations.

Interpretation of the NNP and NP/AP is done in accordance with Québec's Directive 019 criteria. ABA results are presented in Table 13.12.

Table 13.12 – ABA Results (URSTM)

Sample	S_{total} (%)	$S_{sulphate}$ (%)	$S_{sulphure}$ (%)	PA (kg $CaCO_3$/t)	C_{total} (%)	$PN_{modified}$ (kg $CaCO_3$/t)	PNN (kg $CaCO_3$/t)	NPR	Acidogenic	PNC (kg $CaCO_3$/t)
Block 3 Basalt (Volcanic) (U23246)	0,021	0,043	-0,022	-0,7	0,930	46,7	47,4	-69,5	no	77,5
Block 3 Granite (U23247)	0,095	0,012	0,083	2,6	0,063	7,0	4,4	2,7	no	5,2
Block 4 Granite (U23248)	0,013	0,005	0,008	0,2	0,051	6,9	6,7	28,6	no	4,2
Block 6 Granite (U23249)	0,076	0,009	0,067	2,1	0,089	13,1	11,1	6,3	no	7,5
Block 3 lean iron formation (U23250)	1,153	0,031	1,122	35,1	0,399	31,6	-3,4	0,9	uncertain	33,2
Block 4 lean iron formation (U23251)	0,384	0,100	0,284	8,9	0,457	55,3	46,4	6,2	no	38,1
Block 6 lean iron formation (U23252)	0,620	0,009	0,611	19,1	<0,04	12,5	-6,6	0,7	uncertain	<3
Block 3 BIF (U23253)	0,499	0,013	0,486	15,2	0,118	13,4	-1,8	0,9	uncertain	9,8
Block 4 BIF (U23254)	0,429	0,008	0,421	13,1	0,079	11,9	-1,2	0,9	uncertain	6,6
Block 6 BIF (U23255)	0,647	0,009	0,638	19,9	0,041	7,5	-12,4	0,4	uncertain	3,4
Block 3 Sediment (U23256)	0,526	0,248	0,278	8,7	0,283	17,5	8,8	2,0	uncertain	23,6
Block 4 Sediment (U23257)	0,356	0,052	0,304	9,5	0,038	6,9	-2,6	0,7	uncertain	3,1
(1\4) Comp 1 (U23258) Block 1 - Tailings	0,446	0,020	0,426	13,3	0,168	9,5	-3,8	0,7	uncertain	14,0
(2\4) Comp 2 (U23259) Block 3 - Tailings	0,601	0,019	0,582	18,2	0,246	19,2	1,0	1,1	uncertain	20,5
(3\4) Comp 3 (U23260) Block 4 - Tailings	0,574	0,017	0,557	17,4	0,158	20,3	2,9	1,2	uncertain	13,2
(4\4) Comp 4 (U23261) Block 6 - Tailings	0,574	0,011	0,563	17,6	0,158	18,0	0,4	1,0	uncertain	13,2

The following samples (waste rocks) are considered to be non-acid generating because their $S_{sulphates}$ are lower than 0.3%:

- Deposit 3 Basalt (Volcanic) (U23246);
- Deposit 3 Granite (U23247);
- Deposit 4 Granite (U23248);
- Deposit 6 Granite (U23249);
- Deposit 4 lean iron formation (U23251).

All other samples have an uncertain nature with regards to acid generation, even with NPR values <1 or between 1 and 2. Their NP and AP values are too low to enable a clear prediction using only these ABA results.

Thus, it is recommended to proceed with kinetic testing in order to specify the acid-generating nature of the following samples:

- Deposit 3 lean iron formation (U23250);
- Deposit 6 lean iron formation (U23252);
- Deposit 3 BIF (U23253);
- Deposit 4 BIF (U23254);
- Deposit 6 BIF (U23255);
- Deposit 3 Sediment (U23256);
- Deposit 4 Sediment (U23257);
- (1\4) Comp 1 (U23258) Deposit 1 – Tailings;
- (2\4) Comp 2 (U23259) Deposit 3 – Tailings;
- (3\4) Comp 3 (U23260) Deposit 4 – Tailings;
- (4\4) Comp 4 (U23261) Deposit 6 – Tailings.

Finally, the CNP (carbonates NP) values were calculated from the C_{total} (CNP = 83.33 X C_{total}, in kg $CaCO_3$/t) assuming all carbon is associated with calcite. These values are generally close to the $NP_{modified}$ values and cannot be used to classify the acid-generating nature of the uncertain samples.

13.4 Conclusions and Recommendations

Preliminary metallurgical tests and Davis Tube tests results show the DLIP iron mineralization to be composed mostly of magnetite and to contain very low levels of deleterious elements, except for sulphur certainly originating from the widespread occurrence of pyrite and subordinate amounts of other sulphides. Most of the sulphur (87%) reports to the tailings in the Davis Tube tests and its acid-generating potential should be studied.

Under *Acid-Generating Tailings* section of *Directive 019 for the mining industry* from the "*Ministère du Développement durable, de l'Environnement et des Parcs*", tailings that contain more than 0.3% sulphur and whose potential for acid generation was confirmed by testing have to be treated accordingly.

At the next study stage, geochemistry study should be performed on more samples for better characterization and to determine what process conditions may be required.

Also, acid generation tests should be performed in order to know if there is a possibility of acid-generation. Static testing has been performed and dynamic characterisation tests have to be carried out on the tailings.

13.5 Future Work after PEA (Additional Testwork)

Additional work will be required to upgrade the reliability of certain parameters and of parts of the Mineral Resources base.

The main recommendations include:

- Grind size determination/optimization studies for all deposits (typical standard in taconite plant is a grind size of 44 micron (325 mesh);
- Mineralogical study should be performed on the iron mineralization to characterize the mineral species and to know the liberation size;
- For each deposit, batch bench scale test work is to be done to confirm the flow sheet for the development of an overall magnetite processing plant;
- Increase the number of Davis Tube tests to 50% of the samples to increase the confidence level of the regression model and provide a better overall estimation of the Davis Tube Weight Recovery for the deposits;
- Additional Crusher, Ball mill and Rod mill Bond work indexes (CW_i, BW_i, RW_i), to better define rocks hardness throughout the deposits;
- Determinate magnetic Fe from Davis Tube and Satmagan tests on the same samples in order to calculate a correlation between the two and later switch to Satmagan only to assess magnetic Fe, which is a cheaper method;
- Detailed Mineralogy of Feed;
- Grindability test to evaluate variability of the mineralization;
- Additional bench scale testwork;
- Pilot Plant investigation;
- Complete Waste & Tailings characterization (including leaching test and dynamic test);
- Settling & Filtration test work;
- Pulp rheology;
- Pellet feed characterization;
- A series of pot grate test must be done on representative concentrate samples to define the pellet Fe and silica content as well as the grate factor and all the other pellet quality parameters;

- Samples gathering for vendor testwork (Hydroclassifier, thickeners, filters, magnetic separators);
- Initiate a geotechnical study to assess rock quality to support final pit wall, as well as hydrogeological and hydrological studies;
- Use certified blank material to monitor the laboratory performance and commercial standards with at least one with a certified Fe values close to the cut-off grade and one to the mode;
- Additional metallurgical tests will be necessary, such as: S.G., Mineral Characterization, Size Distribution, Bulk Density Determination, Static Thickening, Dynamic Thickening, Pulp Rheology, Vacuum Filtration and Pressure Filtration.

The evaluation of using High Pressure Grinding Roll (“HPGR”) was not part of the PEA. This option would be an opportunity that should be studied during the next study phase.

To reject further portion of the non-magnetic gangue, a rougher magnetic separation stage in the ball mill grinding circuit should be explored in the next study phase.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Mineral Resource Estimates Statement

Following a drilling campaign from 2011 to 2012 Met-Chem was mandated by Century to update the resource estimate of the Duncan Lake Iron project ("DLIP"). A first resource estimate on the project was done by Met-Chem in 2010. That estimate took into account historical drill holes drilled in 1973 and drill holes drilled in the 2008-2009 drilling campaign. Subsequent to the 2010 estimate 125 newer holes were added in the database for the purpose of the present estimate. Furthermore, historical holes drilled in 1973 were not taken into account due to different issues which will be discussed in the section related to the drill hole database.

The DLIP is composed of 6 deposits named Deposit 1 to 6. A total of 177 holes were used to interpolate the different deposits in the present estimate. The estimation was performed using 3D software MineSight™ and the block modeling approach was used. All work was performed under the supervision of the qualified persons. The resource classification follows the guidelines adopted by the Council of the Canadian Institute of Mining Metallurgy and Petroleum ("CIM") through the NI 43-101 standards and guidelines. The criteria used by Met-Chem classifying the estimated resources are based on certainty and continuity of geology and grades.

The relevant definitions of the CIM standards for resource classification are provided in section 14.2. A summary of the Mineral Resource for all deposits is provided in Table 14.1.

Table 14.1 – Summary of the Mineral Resource (Cut-Off of 16% Head Fe; August 2012)

Mineral Resource Category	Metric Tonnes (Million)	Fe (%)	DTWR (%)	DT Fe (%)	DT SiO_2 (%)
Measured	405.6	23.92	26.78	67.26	5.25
Indicated	644.9	24.73	28.09	66.87	5.60
Measured + Indicated	**1,050.5**	**24.42**	**27.58**	**67.02**	**5.46**
Inferred	563.1	24.69	27.97	66.46	6.03

14.2 Definitions

According the final version of the CIM Standards/NI 43-101 which becomes effective on February 1, 2001 and was revised on June 30, 2011:

A **Mineral Resource** is a concentration or occurrence of diamonds, natural, solid, inorganic or fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth 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.

An **Inferred Mineral Resource** is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered trough appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

An **Indicated Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from location such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

A **Measured Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

14.3 Mineral Resource Estimate Estimation Procedures

The estimation of the Duncan Lake Mineral Resource includes following procedures:

- Check and validation of the drill hole database received from IOS;
- Importation of the database in MineSight v. 7.0-5 for each deposit;
- Generation of cross sections that were used for geological interpretation;
- Basic statistics to assess the behaviour of the different elements, need of grade capping and compositing length;
- Development of 3-D envelope for each deposit which shows sufficient continuity of geology and mineralization;
- Geostatistical analysis of variables constrained within the mineralized envelop to assess the spatial continuity of the mineralization and determine search ellipse parameters;
- Generation of block models for each deposit for interpolation;
- Grade interpolation and resource classification according to CIM/NI 43-101 definitions;
- Block model validation;
- Mineral Resource statement.

14.4 Drill Hole Database and Data Verification

The drill hole database used in this estimate was furnished by IOS Services Géoscientifiques Inc. ("IOS") which was in charge of managing the 2011-2012 drilling campaign.

A GeoticLog version of the database was transmitted to Met-Chem. Additional checking and validation were performed under the supervision of Mr. Yves Buro.

Three steps were used in Met-Chem validation procedure. In the first step validation modules implemented in GeoticLog™ were used. Thereafter data were exported into Excel for further verifications before importing into MineSight™. Met-Chem checked in Excel for gaps, overlaps and repeating samples and has no found major errors.

Finally further checking was completed by validation process implemented in MineSight™ when importing the data. Visual checks of assays results on cross sections were also performed in MineSight™. Prior to importing the data into MineSight, Met-Chem replaced all assays fields with missing values with -1. Values under assays detection limit were replaced by half of the same detection limit.

The entire database contains 199 drill holes that were drilled between 1973 and 2012. 22 holes were drilled in 1973, 52 holes during the 2008-2009 drilling campaign and 125 holes during the last drilling campaign of 2011-2012. The 2010 estimate has used both historical holes drilled in 1973 and holes of the 2008-2009 drilling campaign. For the purpose of the 2012 estimate only data of the drilling campaigns of 2008-2009 and 2011-2012 were used. It was decided to ignore historical data since some issues related to their confidence level were highlighted.

The main issues questioned were related to their exact location, the absence of QA/QC procedure in that moment and the missing cores to allow rechecking.

Furthermore, analytical procedure description was missing and it was assumed that soluble iron method has been used. That method differs considerably of the XRF method which was used to analyse all samples of the 2008-2009 and 2011-2012 drilling campaigns.

The content of the drill hole database imported into MineSight™ is presented in Table 14.2 while the statistics of that part of the data used for the present estimation are presented in Table 14.3.

Table 14.2 – Content of the drill hole database

File	Fields
Collar	Hole Name, Easting, Northing, Elevation, Length, Block #, Geometry Type, Sample Site Type, Unit
Assays	Hole Name, From, To, Length, Sample #, SiO_2, Al_2O_3, Fe_2O_3, Fe, Cr_2O_3, MgO, MnO, P_2O5, TiO_2, SrO, Na_2O, K_2O, BaO, CaO, S, LOI, DTWR, FeDT, SiO_2DT, DT Lab code, DT Lab name
Survey	Hole Name, Azimuth, Dip, Depth
Lithology	Hole Name, From, To, LCODE, Litho

Table 14.3 – Basic statistics on the quantity of data used for the estimate of 2012

	Holes Number	Total length drilled (m)	Samples Number	Assays length (m)	Survey Number
2008-2009 Drilling Campaign	52	10,460.25	1,489	6,484.68	392
2011-2012 Drilling Campaign	125	44,006.65	7,689	22,148.65	8,629

14.4.1 Geological Modeling Procedures

After importing the drill hole database into MineSight vertical sections were generated in order to perform geological interpretation on 2D basis prior generation of 3D envelopes by triangulation. The drilling grid spacing varied mainly from 200 m to 400 m, with a very few sections drilled at 100 m. Deposits 1, 3, 4, and 6 are those being more drilled with a 200 m spacing while Deposits 2 and 5 have been drilled on a wider spaced grid. Vertical sections were defined perpendicular to deposits elongation and looking east-northeast.

A Digital Terrain Modeling ("DTM") was created using public data source (www.geobase.ca, Federal government website). For that purpose contour lines with an interval of 2 m were used. However, that precision does not reflect the precision level of data acquisition. For holes drilled during the 2008-2009 drilling campaign collar elevation was linked on the DTM map created since elevation supplied by hand-held GPS units, had a margin of error equal 3 times the magnitude of error on X and Y direction. Holes drilled during 2011-2012 drilling campaign were surveyed with DGPS which gives coordinates with high precision level. For this reason their Z values were kept same.

The Lidar survey completed in 2012 was not used since the data was being processed at the time of writing the report of 2012.

A cut-off grade of 16% Fe was used when modeling mineralization limits on vertical sections. That cut-off was considered as a realistic number based on values being used in operating mines in Canada. The recommended cut-off value may change anytime depending iron price on the market. It should be noted that the cut-off value used to draw the iron mineralization boundaries on sections is not based on any economic study related to the DLIP. In the 2010 resource estimate a minimum thickness of 6 m was used to determine if an interpreted mineralized lens should be considered individually for mining purpose or diluted within neighboring lenses. In the 2012 estimate that limit was fixed at 10 m taking into account size of mining equipment being now used.

However, changes introduced affect only Deposits 3, 4 and 6 where newer holes were drilled in 2011-2012. No change was made in the geological interpretation of Deposits 1, 2 and 5. In addition to information displayed on vertical sections Met-Chem has also used the surface geological map produced by the Quebec Ministry of Natural Resources ("MNR") and available magnetic map to support geological interpretation. Geological contacts were snapped on individual section based on real (X, Y, Z) coordinates. 2D polygons defined on section basis were then used to produce 3D envelopes by triangulation of the contact points. 3D envelopes were thereafter used to assign a specific envelope code to all assays within each block model.

14.5 Basic Statistical Analysis on Assays

Using updated database Met-Chem performed basic statistics on assays data in order to better understand grade distribution and take necessary actions before going to next steps of resource estimate.

In Table 14.4, statistics were calculated for different elements in head and Davis Tube products. Samples used to calculate statistics referred to all deposits and were constrained within mineralized envelopes according geological interpretation.

Table 14.4 – Basic statistics on assays constrained within mineralized envelopes

	HEAD															DAVIS TUBE		
	Length	Al_2O_3%	BaO%	CaO%	Cr_2O_3%	Fe%	K_2O%	MgO%	MnO%	N_2O%	P_2O_5%	S%	SiO_2%	SrO%	TiO_2%	DTWR%	FeDT%	SiO_2%DT
Average	3.10	5.04	0.05	2.72	0.03	23.62	1.56	2.71	0.07	1.31	0.15	0.81	51.24	0.03	0.19	31.77	66.91	5.60
Median	3.00	3.83	0.04	2.22	0.01	25.52	1.44	2.23	0.06	0.67	0.15	0.52	50.78	0.02	0.13	32.60	67.57	4.74
Mode	3.00	0.82	0.04	1.43	0.01	29.15	0.87	1.60	0.04	0.19	0.14	0.42	49.87	0.01	0.07	40.80	69.94	10.10
Standard Deviation	0.93	3.97	0.03	1.69	0.14	8.74	1.00	1.69	0.12	1.51	0.12	1.64	5.17	0.03	0.25	11.67	2.99	3.64
Sample Variance	0.87	15.77	0.00	2.87	0.02	76.33	0.99	2.87	0.02	2.28	0.02	2.70	26.72	0.00	0.06	136.24	8.93	13.24
Coef. Of Variation	0.30	0.79	0.73	0.62	5.12	0.37	0.64	0.63	1.73	1.15	0.81	2.02	0.10	0.93	1.36	0.37	0.04	0.65
Range	9.08	17.65	0.59	15.96	5.90	46.99	10.22	21.31	5.91	8.45	5.91	17.25	72.45	0.43	12.59	56.12	30.98	35.49
Minimum	0.30	0.06	0.01	0.28	0.01	0.76	0.01	0.13	0.01	0.01	0.00	0.01	13.93	0.01	0.01	0.28	40.46	0.71
Maximum	9.38	17.71	0.59	16.24	5.91	47.75	10.23	21.44	5.91	8.45	5.91	17.25	86.38	0.43	12.59	56.40	71.34	36.20
Number of samples	5913	5912	4990	4990	4084	5912	4990	5911	5912	4990	5912	2657	5912	4990	5912	805	805	805

Figure 14.1 to Figure 14.5 show histograms for major head elements and Davis Tube products. The total head iron histogram (Fe %) shows two populations.

A dominant population is related to banded iron formation with a peak of frequency at about 27% Fe. Another population, less pronounced, is present between 1 and 15% Fe and is mainly related to presence of internal waste such as felsic intrusions. Such waste population (Fe < 15%) represents about 17.5% of all assays within mineralized envelopes in all deposits.

The silica oxide in head shows a pattern close to a Gaussian distribution since the mode, average and median are very close. Assays results for product elements, Davis Tube Weight Recovery, Iron and Silica in the concentrate, were only available for 805 samples. This means that not all head samples were systematically been submitted to Davis Tube analyses.

Samples submitted to such analyses were not chosen to cover entire holes but are rather randomly scattered in different holes.

Related DWTR%, FeDT% and SiO_2DT% histograms are presented in Figure 14.3 to Figure 14.5. Figure 14.6 shows a histogram of sample lengths. Sampling length is varying between 30 cm and 9.38 m with an average of 3.09 m and a mode of 3 m.

Figure 14.1 – Fe% Histogram for all Assays within Mineralized Envelopes

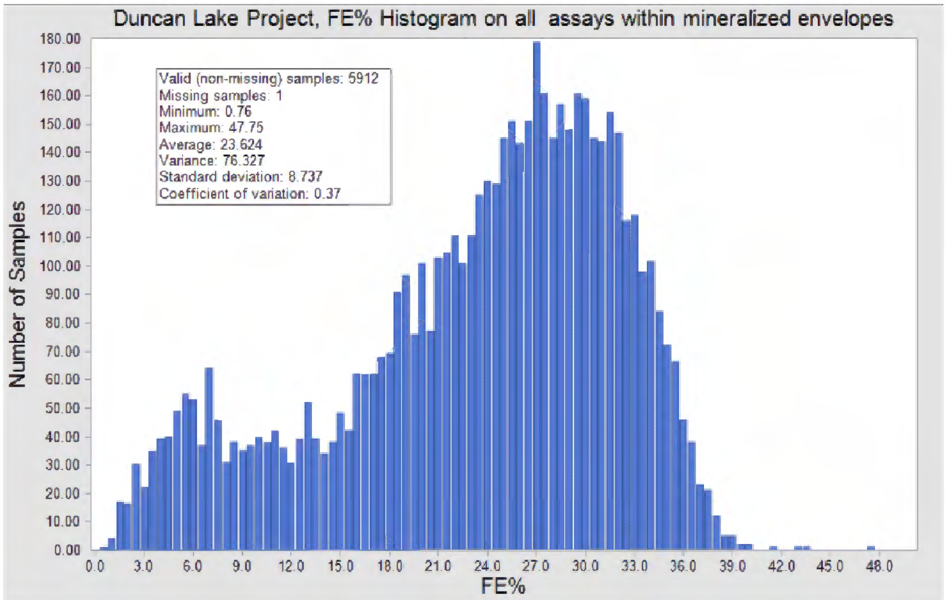


Figure 14.2 – SiO_2% Histogram for all Assays within Mineralized Envelopes

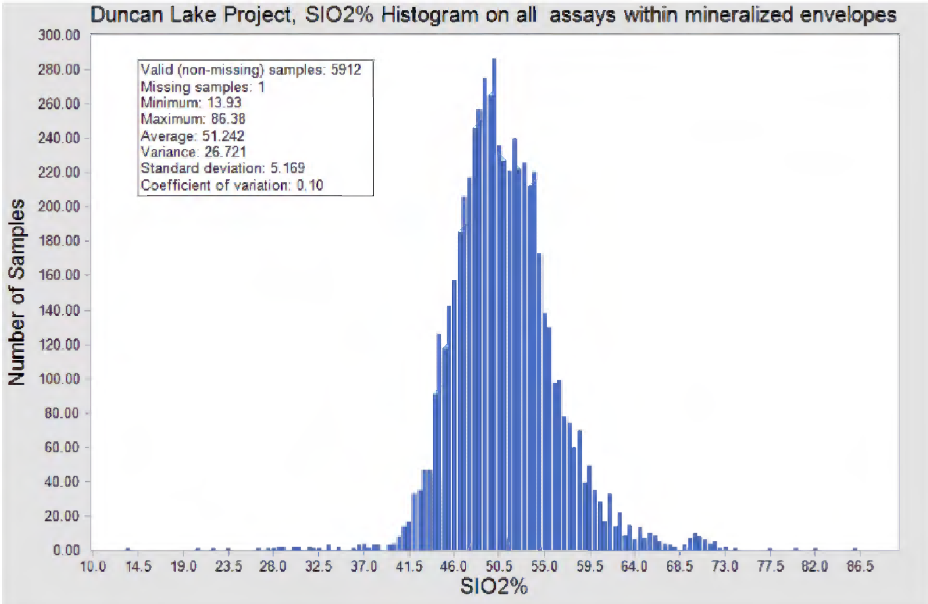


Figure 14.3 – DTWR% Histogram for all Assays within Mineralized Envelopes

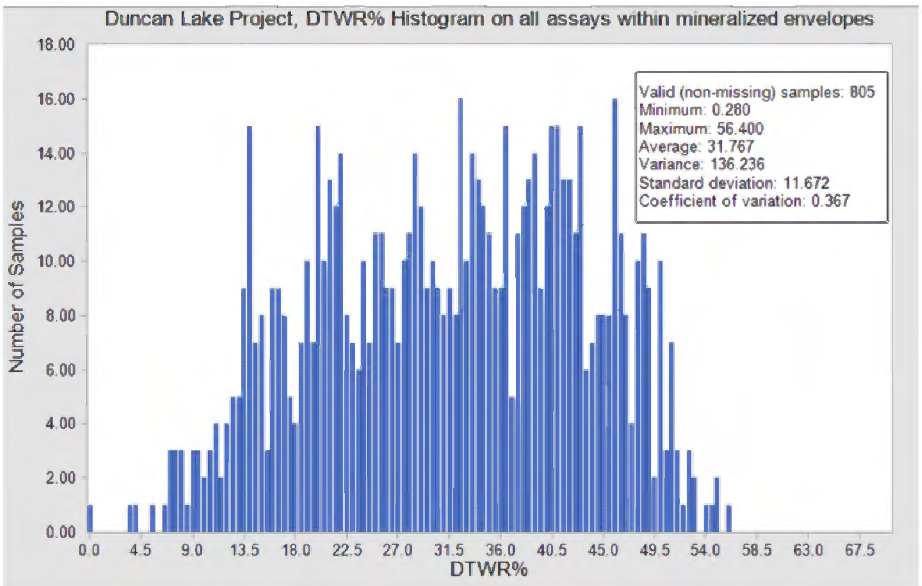


Figure 14.4 – FeDT% Histogram for all Assays within Mineralized Envelopes

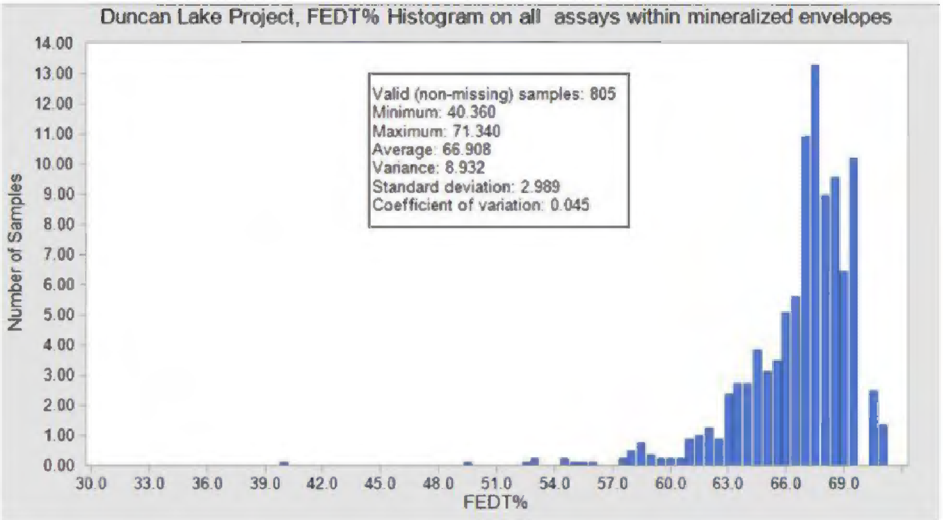


Figure 14.5 – SiO_2DT% Histogram for all Assays within Mineralized Envelopes

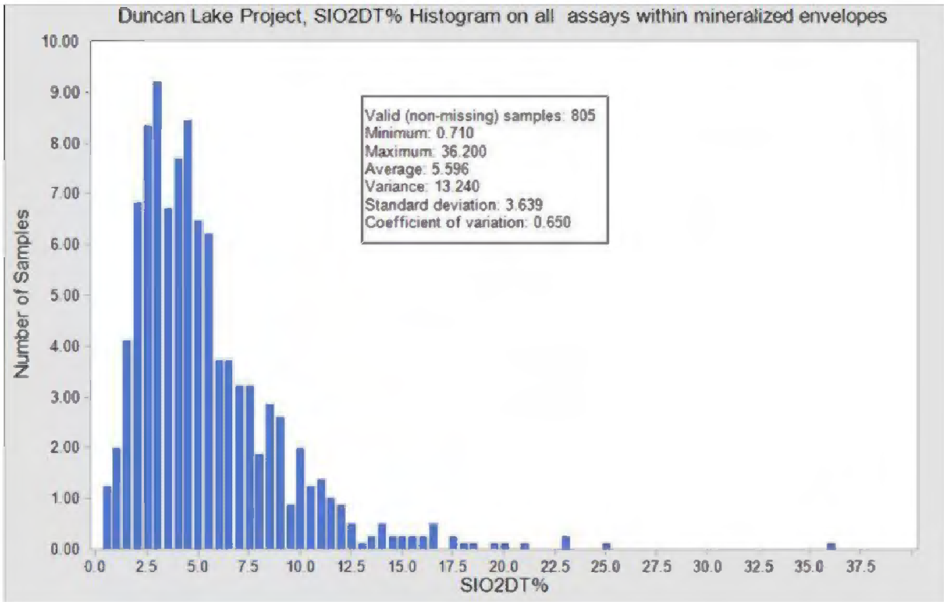
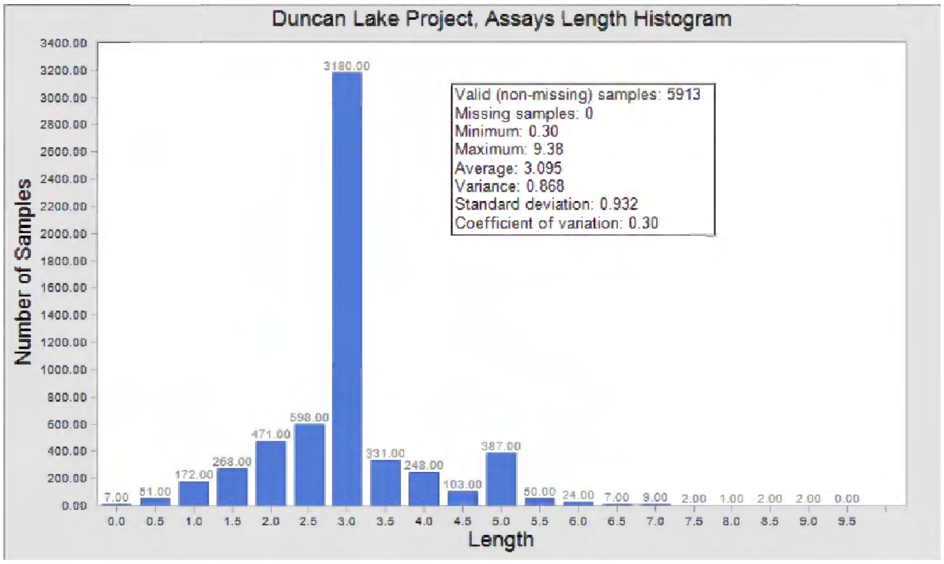


Figure 14.6 – Histogram for assays Length

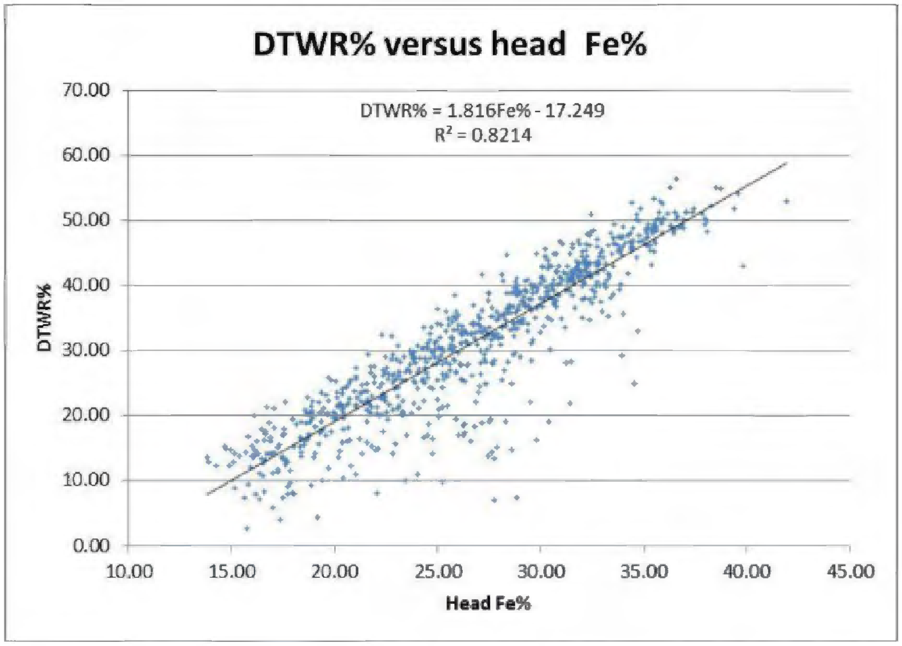


14.6 Regression Function for Missing DTWR%

Fe % and Davis Tube Weight Recovery (DTWR %) were used to generate a regression function (Figure 14.7). The correlation coefficient was relatively high (R2 = 0.82) and demonstrated a reasonably strong relationship. For this reason the regression formula was used to fill DTWR% samples where this element was not analysed. The total number of samples submitted for Davis Tube tests were 843. 841 pairs of samples were used to build the regression function. This regression function was used to fill 8,336 samples (exclusion made of 2 outliers). Among those 8,324 samples, 5,107 calculated samples were constrained within the mineralized envelopes and were involved in resource modeling. The percentage of samples used to build the regression model versus number of DTWR % being calculated is 16.47%. This percentage should be increased to 50% in next stages of the project development by analysing more DTWR %. Furthermore Met-Chem recommends performing Satmagan determinations of magnetic iron on samples already analysed for DTWR% in order to analyse the correlation relationship between both assessing methodologies. Since the Satmagan Test is cheaper than the Davis Tube Test it is recommended to later switch to this method when analysing the magnetic iron content of samples.

By this way, remaining samples not analysed with the Davis Tube method could be submitted to the Satmagan iron determination once the correlation relationship between both methods is well established.

Figure 14.7 – Regression model between DTWR% and Head Fe%



14.7 High Values Capping

For Fe% only one value corresponding to a grade of 47.75% iron could be considered as outlier since it value differ of about 9.5% of the rest of high values population. Because this difference was found to be not much pronounced not cut-off was apply. In the population of head SiO_2% there are 4 extreme values ranging from 77.76% to 86.38% while remaining high grades population ranges from 68% to 74.18% SiO_2. It was found that 3 of those extreme values are related to shorter samples. These values were kept as they will be diluted during compositing. There were not outliers associated with FeDT%. The only SiO_2DT% outlier (36% SiO_2) which is combined with the lowest FeDT% was also not removed. In conclusion no capping was applied to any element for the purpose of the present resource modeling.

14.8 Compositing

Compositing was done at the fixed length of 3 m, which represents the statistical mode of the sampling length population. Composite histograms of relevant elements are show on Figure 14.8 to Figure 14.12. A particular attention was made on DTWR% histogram since a regression model was used to calculate most of values. A peak of samples having DTWR% equal or close to zero could be seen on the histogram left corner. These correspond mainly to calculated DTWR% values. In fact, with the regression equation negative DTWR% values were calculated for all samples having Fe% less than 9.50%. Those negative values were replaced by zero. The corresponding under population (Fe < 9.50%) is easily visible on the left part of Fe% composites histogram.

Figure 14.8 – Fe% Histogram for Composites within Mineralized Envelopes

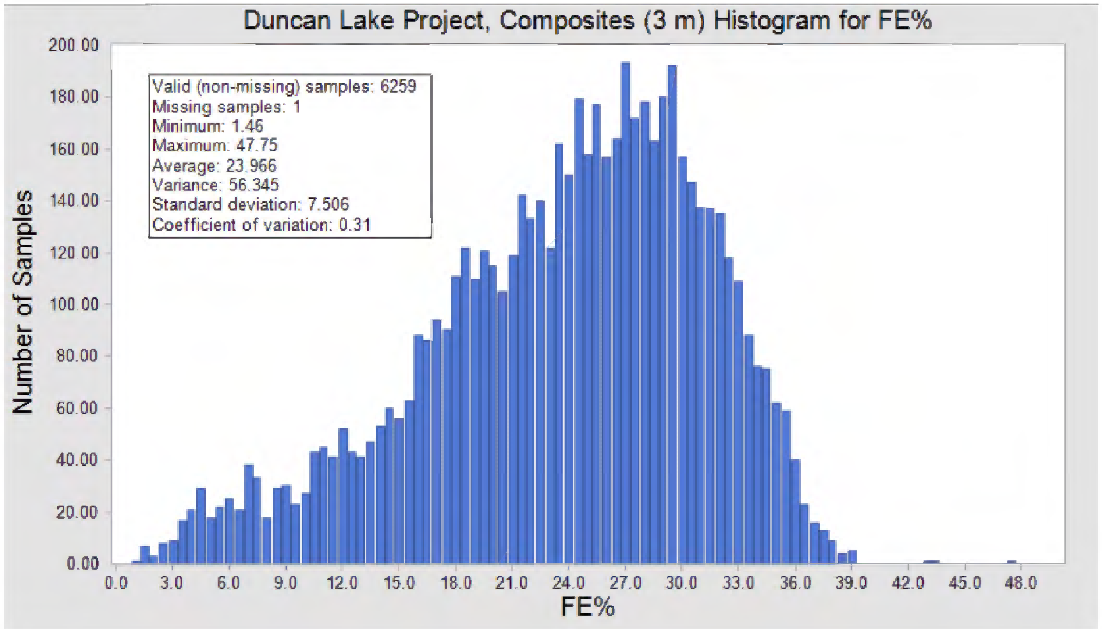


Figure 14.9 – SiO_2% Histogram for all Composites within Mineralized Envelopes

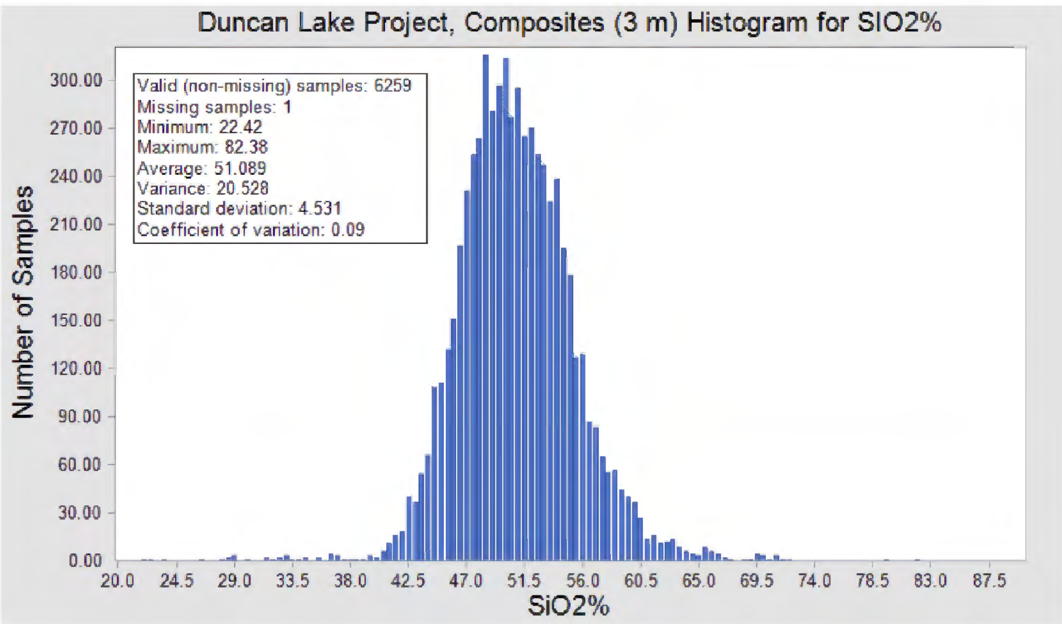


Figure 14.10 – DTWR% Histogram for all Composites within Mineralized Envelopes

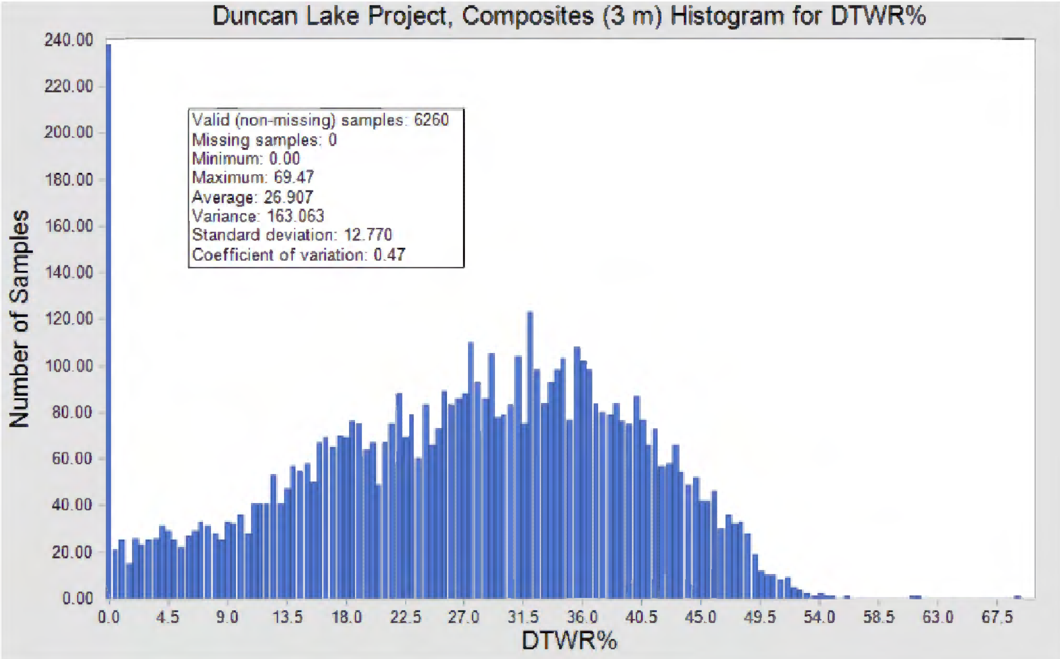


Figure 14.11 – FeDT% Histogram for all Composites (3 m) within Mineralized Envelopes

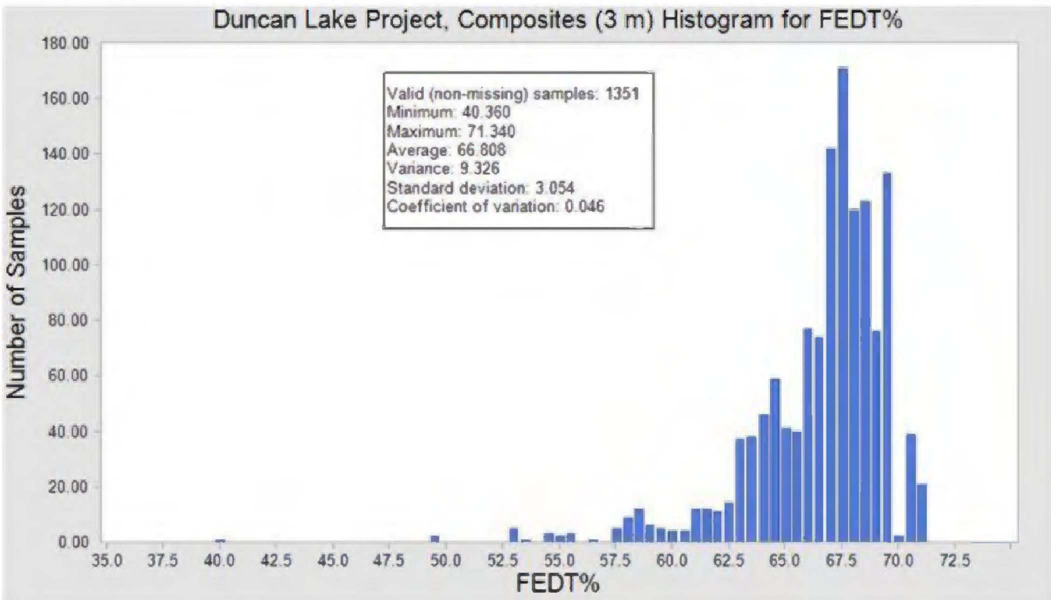
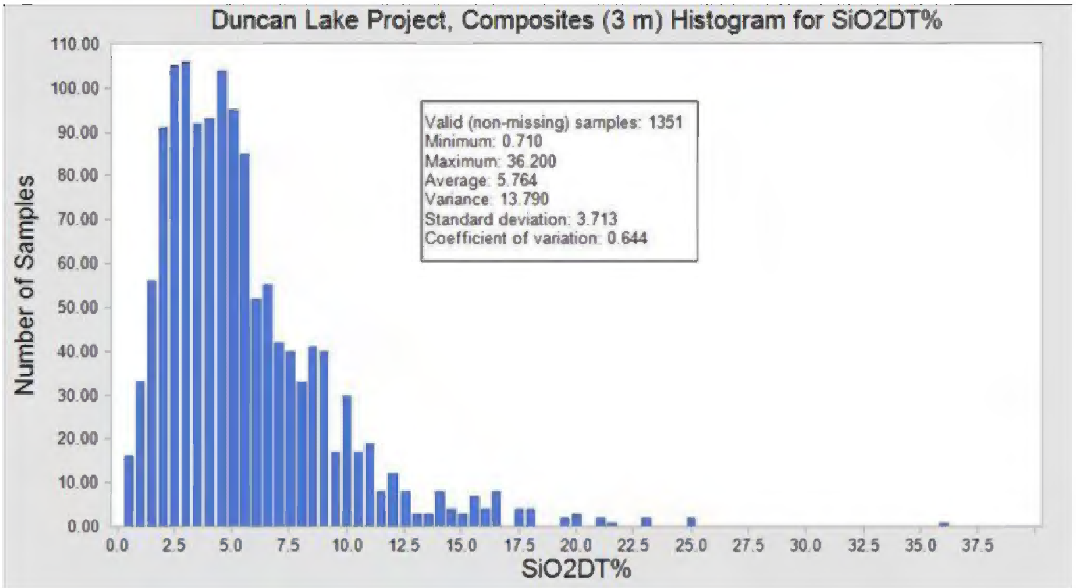


Figure 14.12 – SiO_2DT% Histogram for all Composites (3 m) within Mineralized Envelopes



14.9 Variograms Modeling

Variograms were modeled using 3 m composites population in order to analyse the spatial continuity of the mineralization and determine parameters to be used for grade interpolation. Variograms were modeled by S. Ibrango, Ph.D., P. Geo., Senior geologist at Met-Chem. In the 2010 estimate variograms were only derived for Deposit 1 which was the only non-deformed deposit having sufficient data. In the present case variograms were generated for Deposit 4 since the recent drilling program has allowed increasing number of available data. No newer drill holes were drilled on Deposit 1. Figure 14.13 and Figure 14.14 show the histogram of elements used in modeling variograms. The number of samples used is 1,606 which represent a statistically relevant population size for such analysis.

Directional variograms were generated for Fe% and DTWR% in directions corresponding to the major axis (axe of better continuity), the semi major axis (perpendicular to the major) and the minor axis (normally perpendicular to the major and semi-major axis).

Figure 14.15 and Figure 14.16 show experimental and model variograms on strike direction (Major Axis) with an azimuth of N60° and a dip of 0°. The Range in this axis is 300 m for both Fe% and DTWR%. A high nugget effect is present for DTWR% where the ratio nugget to total sill is 85.76% compared to 32.14% in the case of Fe%.

Figure 14.13 – Fe% Composites Histogram on Deposit 4 (for variograms modeling)

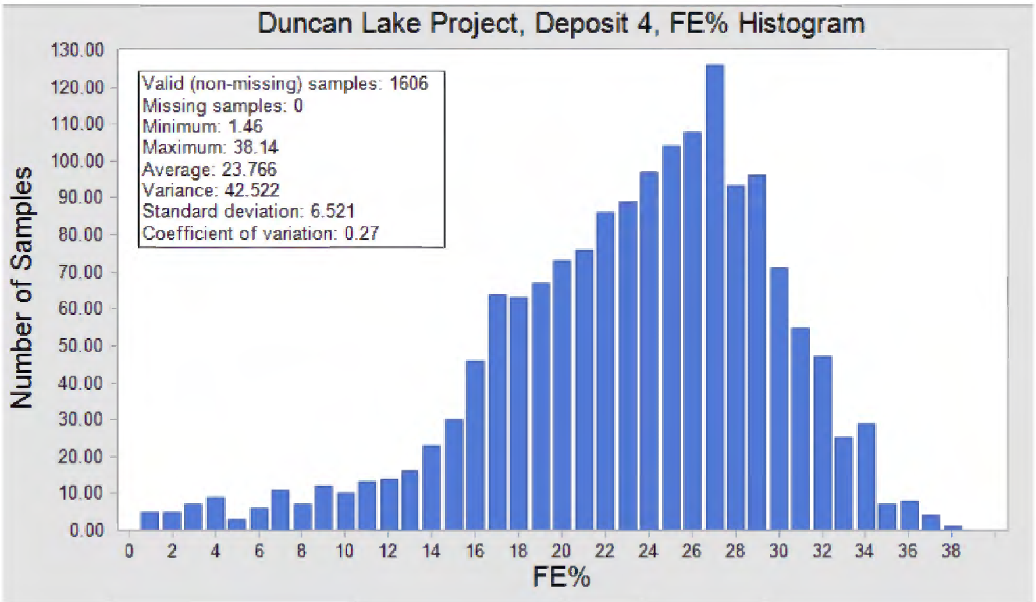


Figure 14.14 – DTWR% Composites Histogram on Deposit 4 (for variograms modeling)

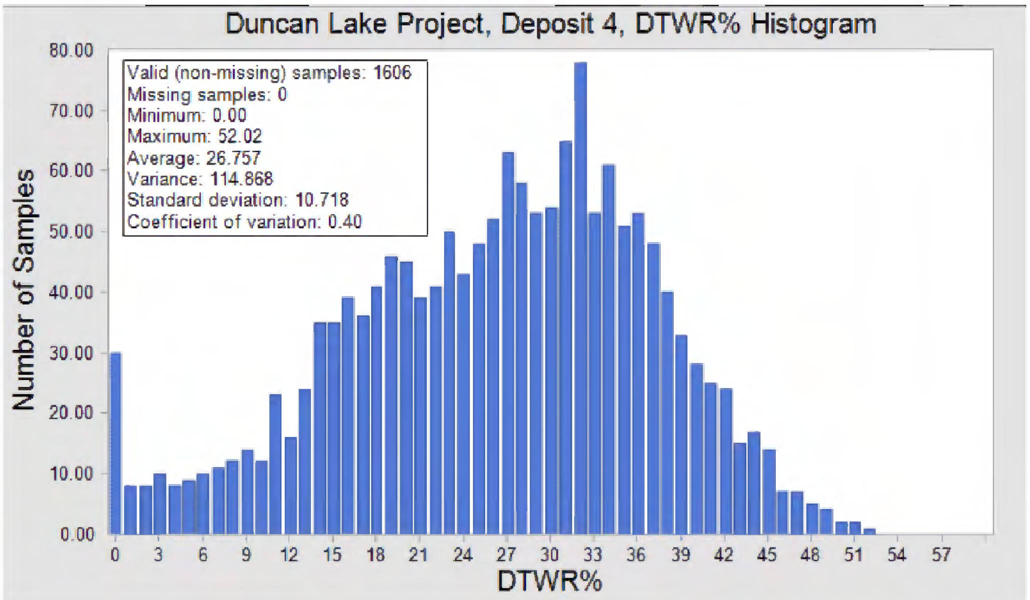


Figure 14.15 – Fe% Variograms along Strike Direction (Major Axis) for Deposit 4

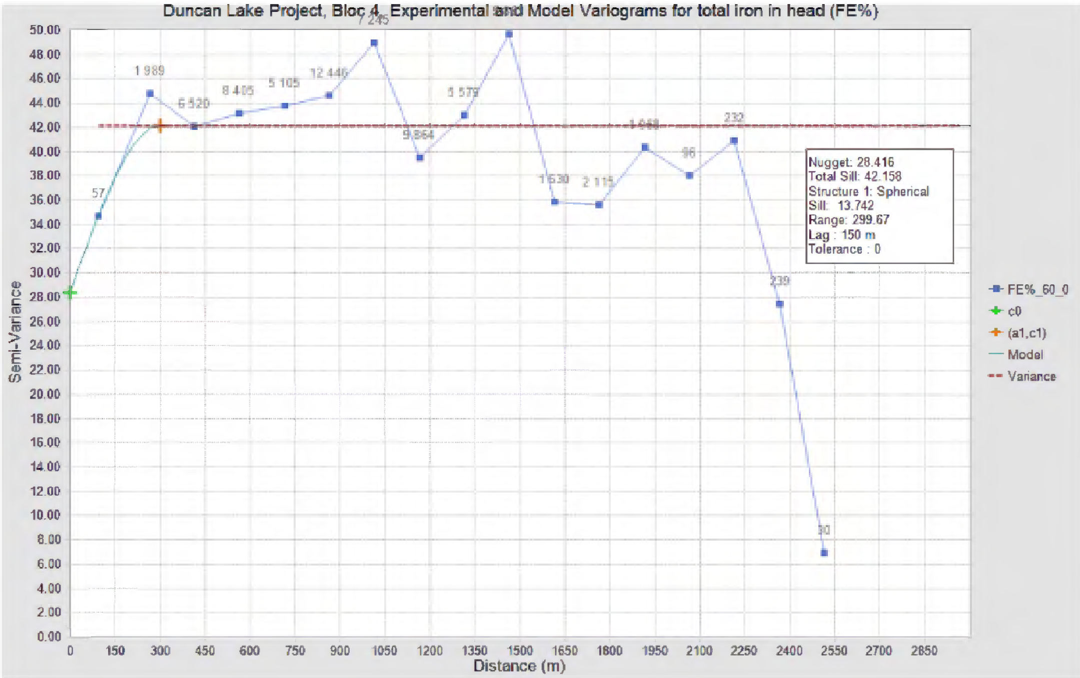
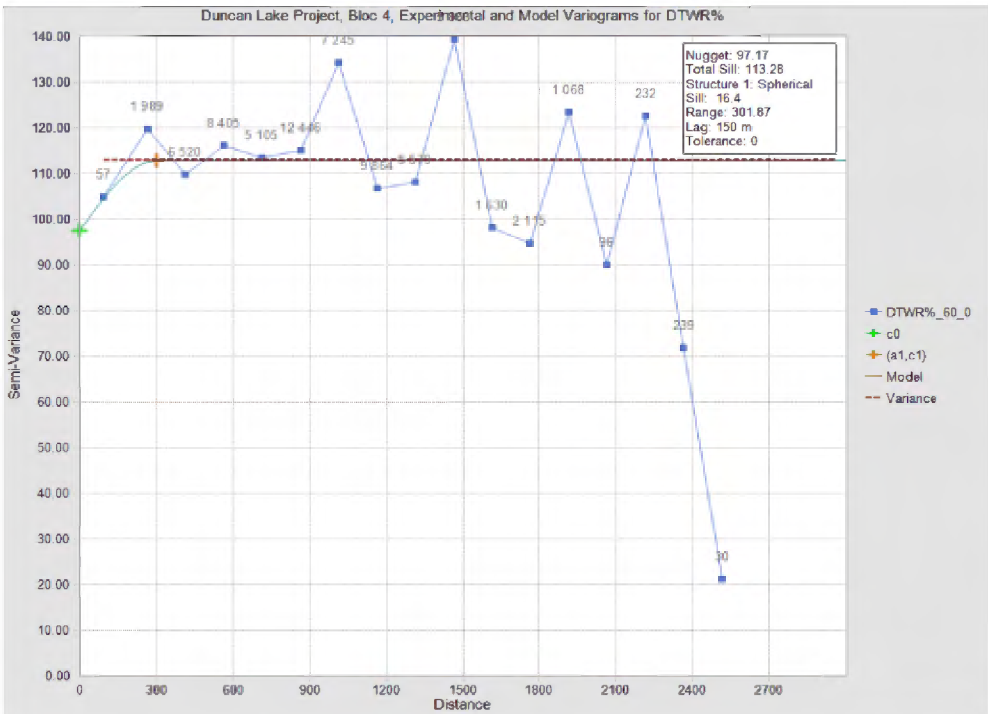


Figure 14.16 – DTWR% Variograms along Strike Direction (Major Axis) for Deposit 4



In the semi major axis (Dip direction) ranges of 150 m and 254 m were respectively founded for Fe% and DTWR% (Figure 14.17 and Figure 14.18). However, the corresponding sills were higher than the variance of pair elements. This is not the ideal situation. Furthermore, there were lesser pairs of samples available for that direction. This reflects the fact that the dip direction was lesser drilled than the strike direction. During variograms modeling of Deposit 1 in previous resource estimate it was not possible to model the dip direction due to lack of information in that direction. A chosen value was used by the QP based on his experience on such BIF deposits. For now ranges founded in the dip direction could be considered as the best information available and based on variogram modeling. Variograms analysis of the dip direction should be redone ounce that direction is better informed. For the purpose of the present estimate Met-Chem has elected to consider the more conservative lower range (150 m) for the semi major axis.

Figure 14.17 – Fe% Variograms across Strike Direction (Semi Major Axis) for Deposit 4

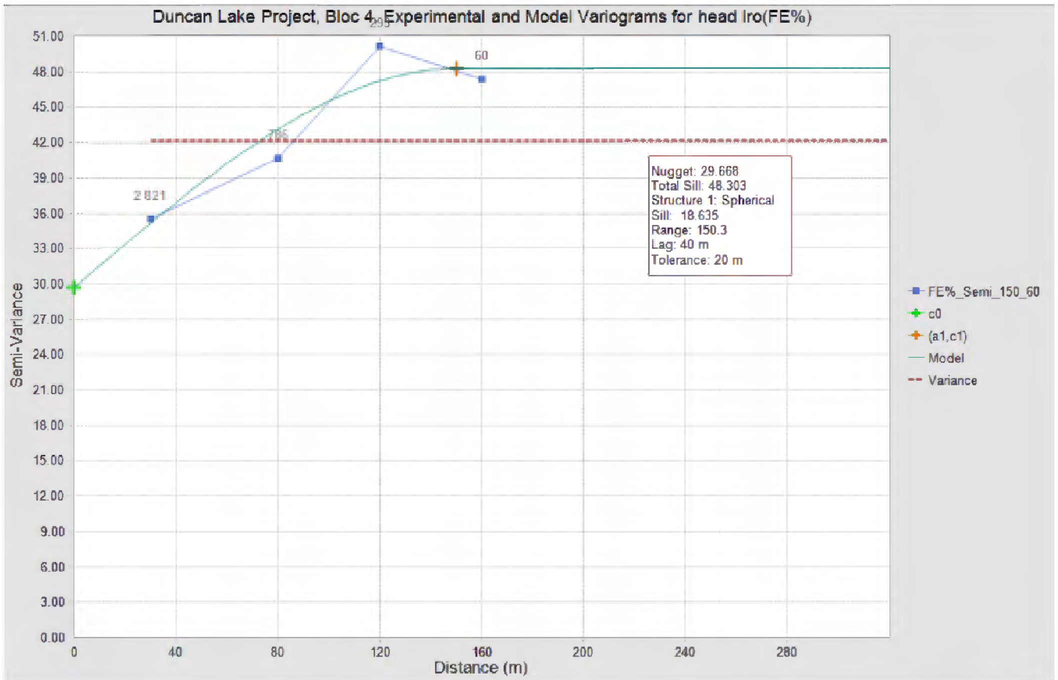
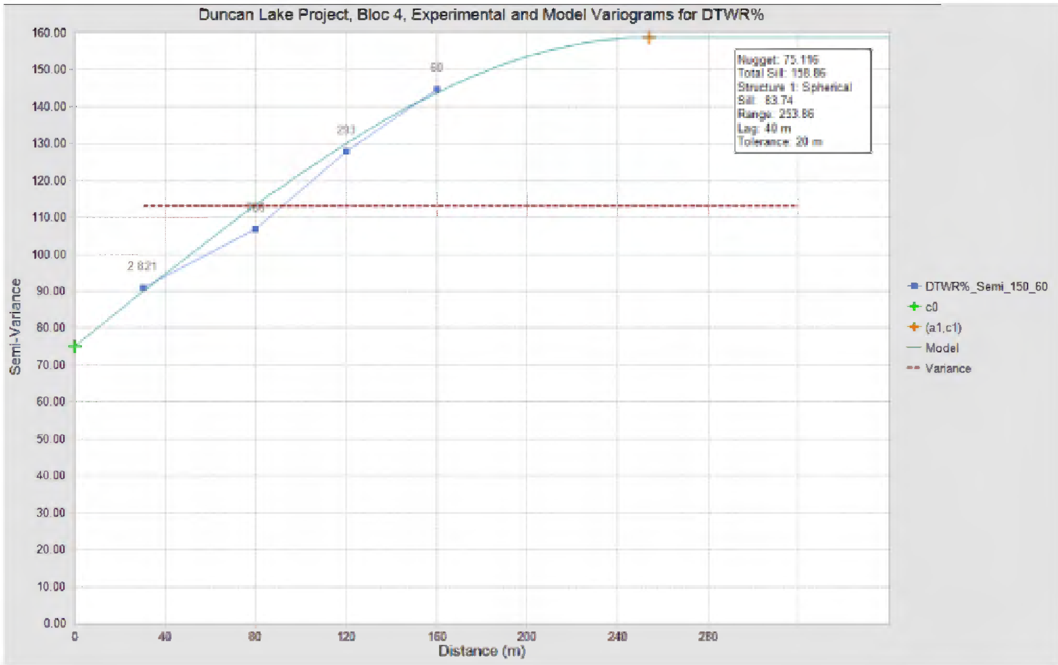


Figure 14.18 – DTWR% Variograms across Strike Direction (Semi Major Axis) for Deposit 4



Combined down-hole variograms were generated in order to provide an indication of the range of down-hole continuity as well as the background of nugget effect for adjacent samples. Figure 14.19 and Figure 14.20 display experimental and model down-hole variograms for Fe% and DTWR%. For both elements the nugget effect are respectively 39.49% and 35.88% of the total sill. The range on the down-hole direction is approximately 12 m and variograms show a Hole-Effect pattern. Since drill holes are targeted to intersect the iron formation perpendicular to the strike and dip direction, the range of the down-hole variogram could reasonably be considered as an approximation of the spatial continuity in the minor axis. In fact the minor axis, strictly considered, is generally not well define in the case of dipping mineralization (in opposition to horizontal layers) due to lack of information in this direction.

In the 2010 estimate a range of 30 m was interpreted on Deposit 1 when modeling the down-hole variogram. For the purpose of the present estimate we have elected to consider the averaged down-hole range as a range in the minor axis. For this reason a distance of 20 m was considered as reasonable for this direction.

Figure 14.19 – Fe% Combined Downhole Variograms (considered as the Minor Axis) for Deposit 4

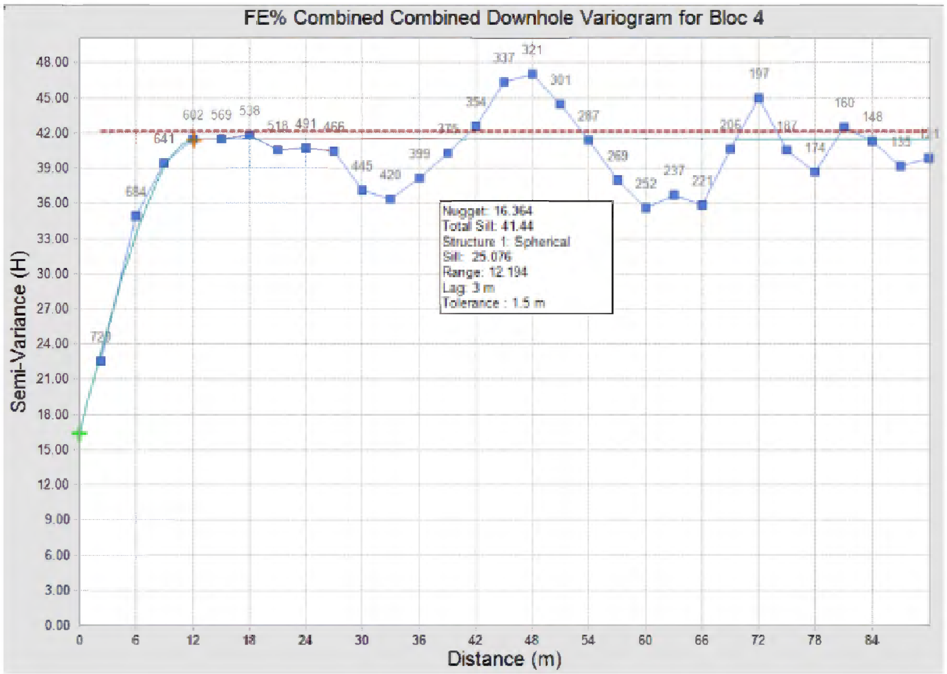
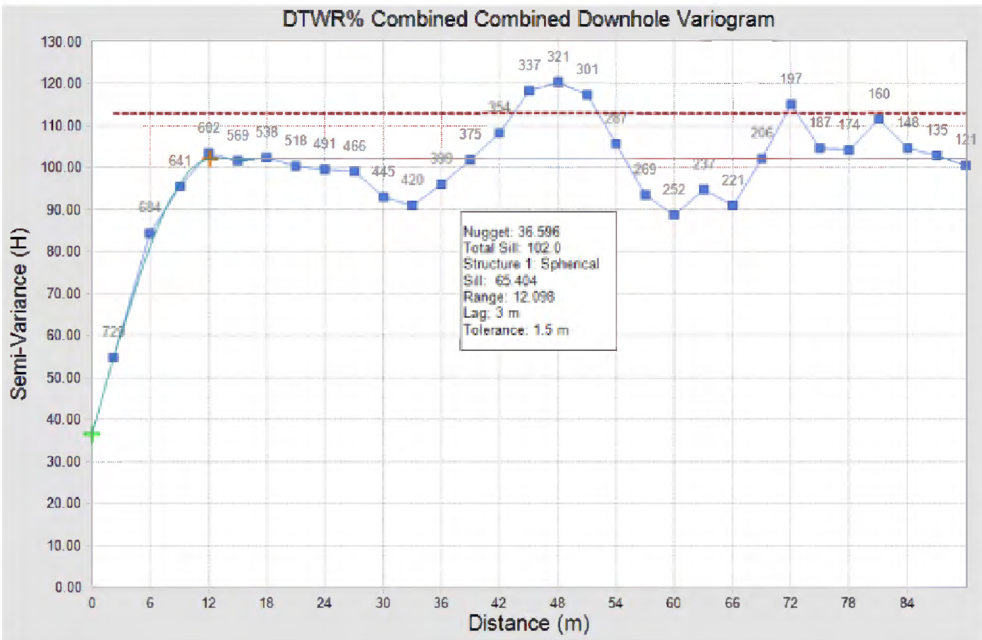


Figure 14.20 – DTWR% Combined Downhole Variogram (considered as the Minor Axis)



In conclusion, ellipse parameters considered for grade interpolation are as follows; 300 m in the major axis, 150 m in the semi major axis and 20 m in the minor axis. This basis ellipse is oriented according each deposit considered and when necessary according under domains defined in some deposits. Domain orientations are discussed in the block modeling section.

14.10 Density

A density of 3.20 g/cm^3 was used to convert the volumes into tonnes, based on a study completed by IOS. The density was determined on a total of 3,107 mineralized samples selected from 93 different holes. The water displacement method was used and the average of the density measurements was 3.179.

Consequently, Met-Chem used a rounded up factor of 3.2, which is the same Met-Chem used in the previous resources estimates (2010), and is in line with the factor used by similar iron deposits and mines. The technique used by IOS for the density determination is discussed under Section 11.4.4 of this report.

14.11 Block Model Setup/Parameters

Block model was created using MineSight™ software package to generated a grid of regular blocks for estimating tonnes and grades. Since the different deposits are very spaced it was not convenient to create a single block model as it will result on useless time and digital memory consumption during models handling. For this reason, a block model was created for each of the 6 deposits. Block sizes used was X = 20 m, Y = 5 m and Z = 10 m. It is the same that those used in the 2010 estimate. Sizes used do not respect the rule of thumb according which Block size should optimally be comprised between 1/3 to 1/5 of the averaged drilling spacing. Y and Z sizes were selected to better control mineralization contacts and conform to mining equipment size. However, those sizes appear to be small comparatively to the drilling spacing and may create an artificial precision in Blocks grade even if the average grade of all Blocks will not change. Block model parameters are labelled in Table 14.5.

Table 14.5 – Blocks Model Parameters

	Direction	Minimum (UTM)	Maximum (UTM)	Number of Blocks	Model Origin (UTM)
Deposit 1	Easting (X)	310 630 .31	316 237.69	280	312 277.88
	Northing (Y)	5 925 836	5 931 443.50	466	5 925 836
	Elevation (Z)	-400	200	60	0
	Rotation angle	N/A	N/A	N/A	315°
Deposit 2	Easting (X)	318 765.09	322 724.88	180	319 825.75
	Northing (Y)	5 932 091	5 936 050.50	400	5 932 444.50
	Elevation (Z)	-400	200	60	0
	Rotation angle	N/A	N/A		315°
Deposit 3	Easting (X)	322 557.34	326 658.56	170	324 254.41
	Northing (Y)	5 934 583	5 938 684	480	5 934 583
	Elevation (Z)	-400	200	60	0
	Rotation angle	N/A	N/A	N/A	315°
Deposit 4	Easting (X)	327 402.56	331 156.63	180	328 398.88
	Northing (Y)	5 940 175	5 943 676.50	310	5 940 175
	Elevation (Z)	-400	200	60	0
	Rotation angle	N/A	N/A	N/A	320°
Deposit 5	Easting (X)	316 430.81	322 695.75	318	318 198.56
	Northing (Y)	5 928 196.50	5 934 461.50	500	5 928 196.50
	Elevation (Z)	-400	200	60	0
	Rotation angle	N/A	N/A	N/A	315°
Deposit 6	Easting (X)	325 843.59	328 329.97	105	326 356.63
	Northing (Y)	5 935 380	5 937 508	300	5 935 380
	Elevation (Z)	-400	200	60	0
	Rotation angle	N/A	N/A	N/A	340°

14.12 Structural Domains for Interpolation

Due to the deformed nature of the mineralization in some deposits it was necessary to define structural domains in order to be able to interpolate all Blocks within mineralized envelopes. Table 14.6 summarizes for each deposit domain(s) used with their orientation in azimuth and dip.

Table 14.6 – Structural domains per deposit for interpolation

Search Ellipsoid Orientation	Domain	Azimuth	Dip
Deposit 1	1	230°	-70°
Deposit 2	1	232°	-82°
Deposit 3	1	217°	80°
	2	55°	70°
	3	235°	50°
	4	157°	65°
	5	278°	64°
Deposit 4	1	215°	65°
	2	235°	65°
	3	250°	70°
Deposit 5	1	245°	-61°
	2	231°	-59°
Depsoti 6	1	240°	85°
	2	245°	80°
	3	270°	70°

14.13 Resource Estimation Method

The Duncan Lake resources were estimated for each deposit using the Inverse Distance Squared Method ("IDW2") which Met-Chem believes is appropriate in estimating resources of deposits showing a good geological continuity along strike such as Banded Iron Formations. Three interpolation passes were used in the estimation. Each grade interpolation pass was based on varying search ranges, minimum and maximum number of composites, and a maximum number of composite per drill hole

14.14 Resource Classification

Each interpolation pass defined in resource estimate was used to classify all blocks that were interpolated. Blocks interpolated using first pass were classified as Measured Resources, blocks interpolated using second pass were classify as Indicated Resource while blocks interpolated using third pass were classify as Inferred Resource. Interpolation parameters used for those pass (Measured, Indicated and Inferred) are displayed in Table 14.7.

Table 14.7 – Interpolation and classification parameters

Interpolation method: Inverse Distance Squared (ID2)			
Description	Resource Category		
	Measured	Indicated	Inferred
Min. composites number	12	6	3
Max. Composites number	20	20	20
Max. Composites per hole	3	3	3
Min. number of hole(s)/category	4	2	1
Ellipsoid - major axis	300 m	300 m	450 m
Ellipsoid - semi-major axis	150 m	150 m	225 m
Ellipsoid - minor axis	20 m	20 m	30 m

Ellipse ranges were gathered from the variogram analysis and reflect grade continuity which is also a confidence factor like the drill hole spacing. Mineral Resource classification is based on certainty and continuity of geology and grades. In addition to quantitative confidence factors, Met-Chem has also considered qualitative confidence factors such as QA/QC results, industry practices for such commodity and Met-Chem's abundant experience with similar types of mineralization.

Mineral Resource statement is based on a cut-off of 16% head Fe. This cut-off has been determined to be appropriate at this stage of the project. In the 2010 resource estimation 3 cut-offs, 16% Fe, 18% Fe and 20% Fe, were selected to simulate tonnage and grade variation. Results show low sensitivity in tonnage and grade to varying the cut-off. This is typical for taconite deposits as they are less sensitive to cut-off variations.

Table 14.8 – Measured Resources (2012)

Deposit	Measured Resources, Cut-Off 16% Fe				
	Metric Tonnes (Million)	Fe (%)	DTWR (%)	DT SiO_2 (%)	DT Fe (%)
1	27.2	22.01	23.98	9.34	64.27
2	4.4	27.05	33.96	6.96	66.34
3	169.4	24.32	26.88	3.39	68.28
4	162.4	23.60	26.49	6.20	66.92
5	N/A	N/A	N/A	N/A	N/A
6	42.3	24.48	28.50	6.21	66.57
Total	405.6	23.92	26.78	5.25	67.26

Table 14.9 – Indicated Resources (2012)

Deposit	Indicated Resources, Cut-Off 16% Fe				
	Metric Tonnes (Million)	Fe (%)	DTWR (%)	DT SiO_2 (%)	DT Fe (%)
1	88.9	23.34	25.26	11.48	62.24
2	31.3	27.33	34.81	5.81	67.33
3	323.6	25.06	28.35	3.63	68.10
4	141.1	24.14	27.17	5.99	66.98
5	N/A	N/A	N/A	N/A	N/A
6	60.0	25.05	29.56	6.47	66.55
Total	644.9	24.73	28.09	5.60	66.87

Table 14.10 – Measured + Indicated Resources (2012)

Deposit	Measured + Indicated Resources, Cut-Off 16% Fe				
	Metric Tonnes (Million)	Fe (%)	DTWR (%)	DT SiO_2 (%)	DT Fe (%)
1	116.1	23.03	24.96	10.98	62.72
2	35.7	27.29	34.71	5.95	67.21
3	493.0	24.81	27.85	3.54	68.16
4	303.5	23.85	26.80	6.10	66.94
5	N/A	N/A	N/A	N/A	N/A
6	102.2	24.81	29.12	6.37	66.56
Total	1,050.5	24.42	27.58	5.46	67.02

Table 14.11 – Inferred Resources (2012)

Deposit	Inferred Resources, Cut-Off 16% Fe				
	Metric Tonnes (Million)	Fe (%)	DTWR (%)	DT SiO_2 (%)	DT Fe (%)
1	139.0	22.80	24.42	9.84	63.51
2	62.9	26.10	31.33	3.33	68.65
3	202.4	25.49	29.16	3.96	67.88
4	74.7	23.92	26.88	6.52	66.45
5	51.3	25.63	29.34	N/A	N/A
6	32.9	25.23	29.65	6.68	66.10
Total	563.1	24.69	27.97	6.03	66.46

Notes: - DTWR % is the Davis Tube Weight Recovery; DT Fe % is the Davis Tube Fe Concentrate Grade.
- Total tonnage may vary due to rounding.
- The effective date of the Mineral Resource estimate is August 24, 2012.
- Resource estimate is based on all six Duncan Lake Deposits.

Met-Chem is unaware of any legal, political, environmental, or other risks that could materially affect the potential development of the Mineral Resources.

Mineral Resource classification is based on the level of confidence on the continuity of geology and grades in the deposit. Areas more densely drilled are usually better known than areas with sparser drilling. Due to the uncertainty attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

15.0 MINERAL RESERVES ESTIMATE

No Mineral Reserves have been estimated for the DLIP.

16.0 MINING METHODS

Met-Chem evaluated the potential of the Duncan Lake Iron Property ("DLIP"), targeting a production rate of 12,000,000 tonnes of iron pellets (acid pellets) per year. This section of the report discusses the pit design, mine plan and fleet requirements that were estimated for the PEA which form the basis for the Mine Operating and Capital Cost estimate presented in Section 21 of this report. Figure 16.1 provides a general layout of the mine site.

16.1 Block Model

The 3-Dimensional Geological Block Models developed by Met-Chem for the Mineral Resource estimates were used as the basis for estimating the economic pit limits, pit design and mine plans. Met-Chem used MineSight® Version 7.04 mine software to create 3-Dimensional mine planning block models. MineSight® is commercially available software that has been used by Met-Chem for the past 25 years.

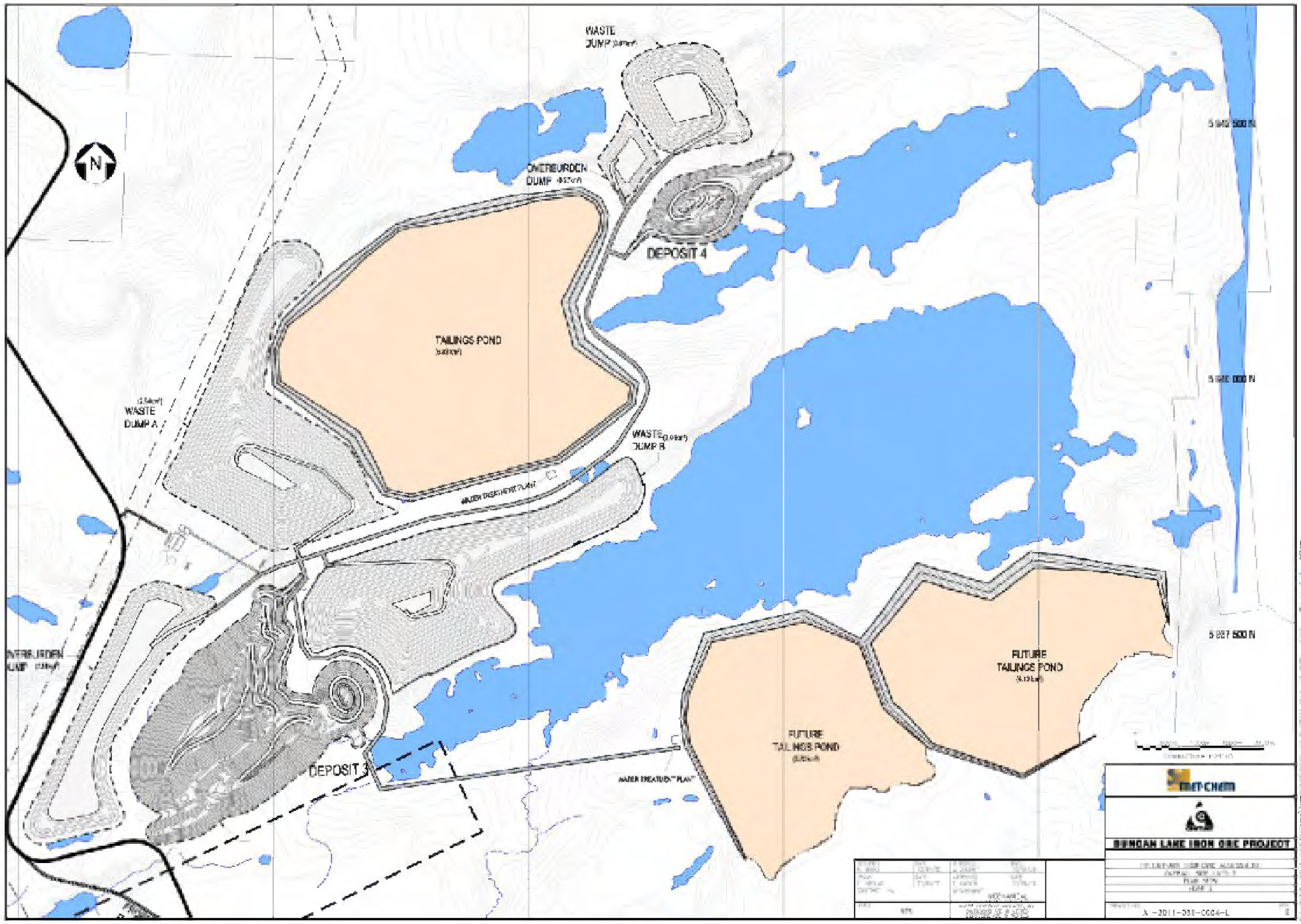
16.2 Material Properties

An in-situ dry density of 3.2 t/m^3 (3.2 g/cm^3) was used for the mineralized material. This density was supplied with the block model. A density of 2.9 t/m^3 was used for the waste rock. This lower density is reflective of the lower iron content in the barren rock.

For mine planning and equipment calculation purposes, an overburden interface surface was designed for each Block based on drill core data. A density of 2.0 t/m^3 was estimated for this material.

A swell factor of 25% was used for the design of the waste and overburden dumps and for determining the mine equipment fleet requirements.

Figure 16.1 –Mine General Layout



16.3 Selection of deposits for the PEA

To maximize the potential economics of the PEA, Met-Chem selected Deposits 3 and 4 as the basis for the PEA. These deposits have the largest tonnage and best mineralogy of the 6 DLIP deposits and can supply the concentrator and pellet plant for over 20 years of full production.

16.4 Mining Method

The mining method selected for the Project is a conventional open pit drill and blast operation with rigid frame mining haul trucks and hydraulic excavators. Pre-production stripping of waste and overburden material will be mined by a contractor.

Vegetation, 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 face shovel hydraulic excavators. The mineralized material will be hauled to a primary crusher and waste rock will be hauled to the waste dump.

To properly manage water infiltration into the pit, a sump will be established at the lowest point on the pit floor. Water collected in this sump will be pumped to a collection point at surface.

16.5 Pit Optimization

Open pit optimization was done on both Deposit 3 and 4 to derive the pit shell with the highest Project Net Present Value (“NPV”). A series of pit shells were generated using the Lerch Grossman algorithm in the Economic Planner optimizer of MineSight®. These shells were generated by varying the selling price.

The optimization was carried out during the initial stage of the PEA study using the cost, sales price and pit and plant operating parameters presented in Table 16.1. These parameters are preliminary estimates for developing the economic pit and should not be confused with the operating costs subsequently developed for the PEA and provided elsewhere in this report. A conservative pellet sales price of USD 140/t was used in the pit optimization, a value lower than the sales price used in the PEA economic evaluation presented in Section 22.0 of this report. The pit optimization was re-evaluated after a preliminary mine plan was completed and the cost, sales price and pit and plant operating parameters were better defined. The results of the second pit optimization using the updated operating costs and sales price confirmed the original optimization results.

Inferred Mineral Resources were used in the optimization and mine plan of the PEA as allowed in the NI 43-101 guidelines for such a study.

In addition, constraints were added to the optimization to ensure that the pit limit remained outside any Hydro Quebec property boundary and also a minimum of 65 meters from any surrounding major lakes, as per governmental regulations.

Table 16.1 – Pit Optimization Parameters

Item	Value	Units
Mining Cost – Mineralization	2.20	$/t (mined)
Mining Cost – Waste Rock	2.40	$/t (mined)
Mining Cost – Overburden	1.75	$/t (mined)
Processing and Pipeline Cost	18.00	$/t (pellet)
Pelletizing Cost	12.00	$/t (pellet)
Shipping Cost	37.00	$/t (pellet)
General, Admin & Infrastructure Cost	5.70	$/t (pellet)
Sales Price	140	USD/t (pellet)
In-Situ Dry Density – Overburden	2.00	t/m^3
In-Situ Dry Density – Mineralization	3.20	t/m^3
In-Situ Dry Density – Waste Rock	2.90	t/m^3
Overall Pit Slope	52	Deg

* The cost parameters are preliminary estimates for developing the economic pit and should not be confused with the operating costs subsequently developed for the PEA Study and given elsewhere in this report.

16.5.1 Pit Optimization Results

a) Deposit 3

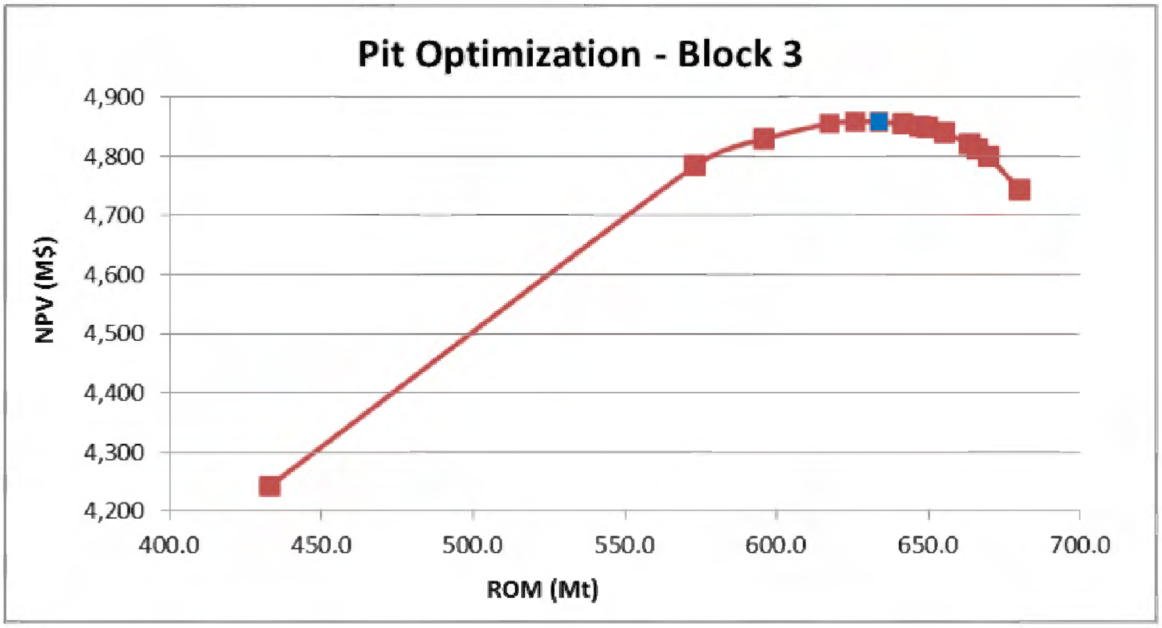
Table 16.2 presents the tonnages and grades that are associated with each of the 14 pit shells generated for the Deposit 3. The NPV was calculated for each shell based on the parameters presented in Table 16.1. Figure 16.2 is a chart showing the NPV vs. the mineralized tonnage for each shell.

Table 16.2 – Deposit 3 Pit Optimization Results

Description	Mineralization (Mt)	Fe (%)	DTWR (%)	Total Waste (Mt)	Strip Ratio	NPV (M$)
PIT01	433	24.97	28.2	497	1.15	4,243
PIT02	573	24.97	28.2	826	1.44	4,784
PIT03	596	24.98	28.2	903	1.52	4,829
PIT04	617	25.01	28.2	990	1.60	4,856
PIT05	626	25.02	28.3	1,031	1.65	4,859
PIT06	634	25.02	28.3	1,067	1.68	4,859
PIT07	642	25.02	28.3	1,107	1.72	4,855
PIT08	647	25.02	28.3	1,135	1.75	4,851
PIT09	650	25.02	28.3	1,148	1.77	4,849
PIT10	655	25.02	28.3	1,181	1.80	4,840

Description	Mineralization (Mt)	Fe (%)	DTWR (%)	Total Waste (Mt)	Strip Ratio	NPV (M$)
PIT11	664	25.02	28.3	1,235	1.86	4,820
PIT12	666	25.03	28.3	1,251	1.88	4,813
PIT13	670	25.03	28.3	1,281	1.91	4,799
PIT14	680	25.03	28.3	1,371	2.02	4,743

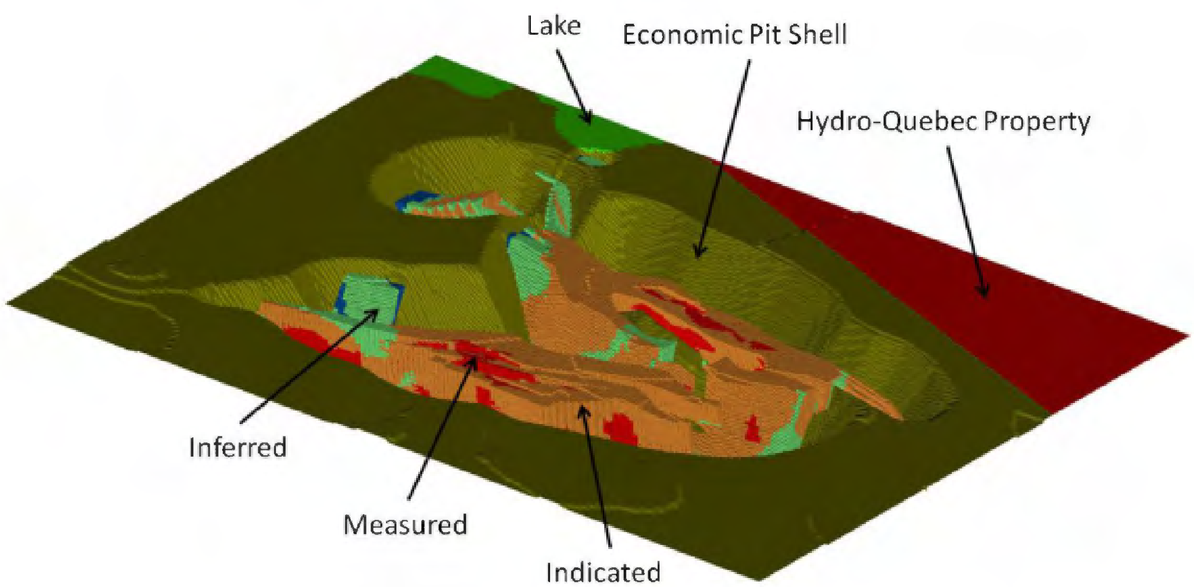
Figure 16.2 – Pit Optimization Results



The pit shell with the highest NPV is PIT06. This pit contains 634 Mt of mineralization with an associated strip ratio of 1.7:1. The optimized pit shell does not account for dilution, mining recovery and pit design. These items are discussed in the Mine Design section of this report.

Figure 16.3 presents an isometric view of the Deposit 3 showing the optimized pit shell number 6.

Figure 16.3 – 3D Isometric View of Deposit 3 - Economic PIT 06



b) Deposit 4

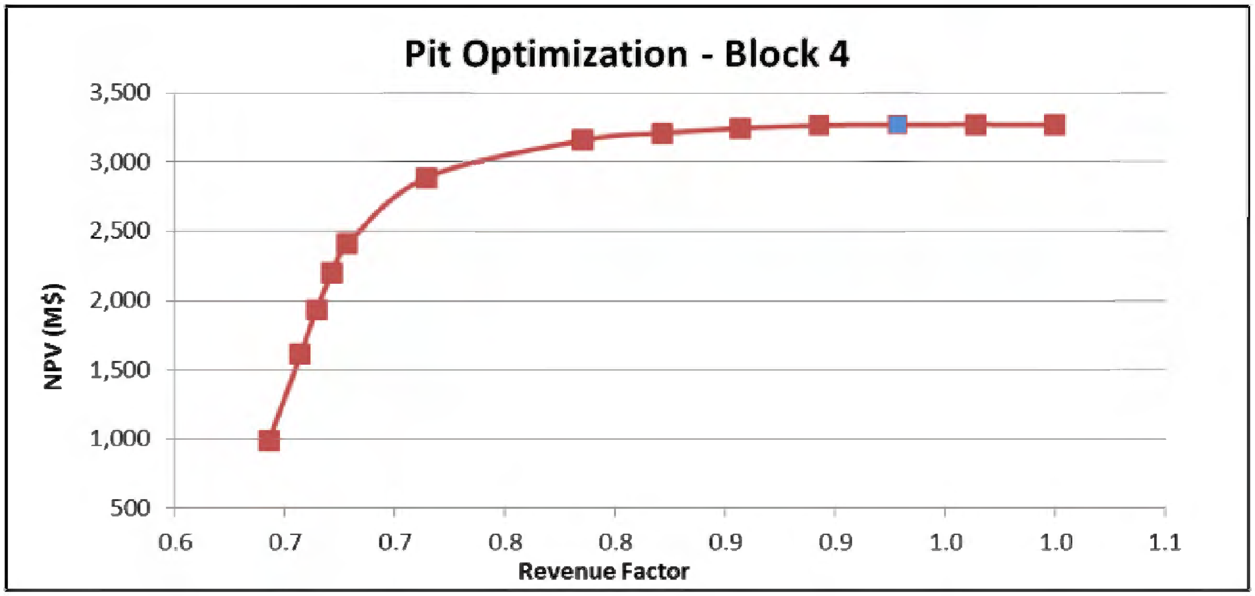
Table 16.3 presents the tonnages and grades that are associated with each of the 14 pit shells generated for Deposit 4. The NPV was calculated for each shell based on the parameters presented in Table 16.1. Figure 16.4 is a chart showing the NPV vs. the mineralized tonnage for each shell.

The pit shell with the highest NPV is PIT12. This pit contains 343 Mt of mineralization with an associated strip ratio of 1.4:1. The optimized pit shell does not account for dilution, mining recovery and pit design. These items are discussed in the Mine Design section of this report.

Table 16.3 – Pit Deposit 4 Pit Optimization Results

Description	Mineralization (Mt)	Fe (%)	DTWR (%)	Total Waste (Mt)	Strip Ratio	NPV (M$)
PIT01	65	24.7	28.5	23	0.36	1,585
PIT02	116	24.5	28.0	64	0.55	2,544
PIT03	146	24.2	27.6	90	0.62	3,041
PIT04	173	24.1	27.4	116	0.67	3,450
PIT05	196	24.1	27.2	141	0.72	3,778
PIT06	255	24.0	27.1	223	0.88	4,520
PIT07	300	23.9	27.0	323	1.08	4,944
PIT08	313	23.9	27.0	366	1.17	5,027
PIT09	325	23.9	27.0	409	1.26	5,085
PIT10	335	23.9	27.0	453	1.35	5,122
PIT11	338	23.9	27.0	466	1.38	5,131
PIT12	343	23.9	27.0	492	1.44	5,138
PIT13	346	23.9	27.0	510	1.47	5,137
PIT14	362	23.9	26.9	625	1.72	5,077

Figure 16.4 – Optimization Results



16.6 Mine Design

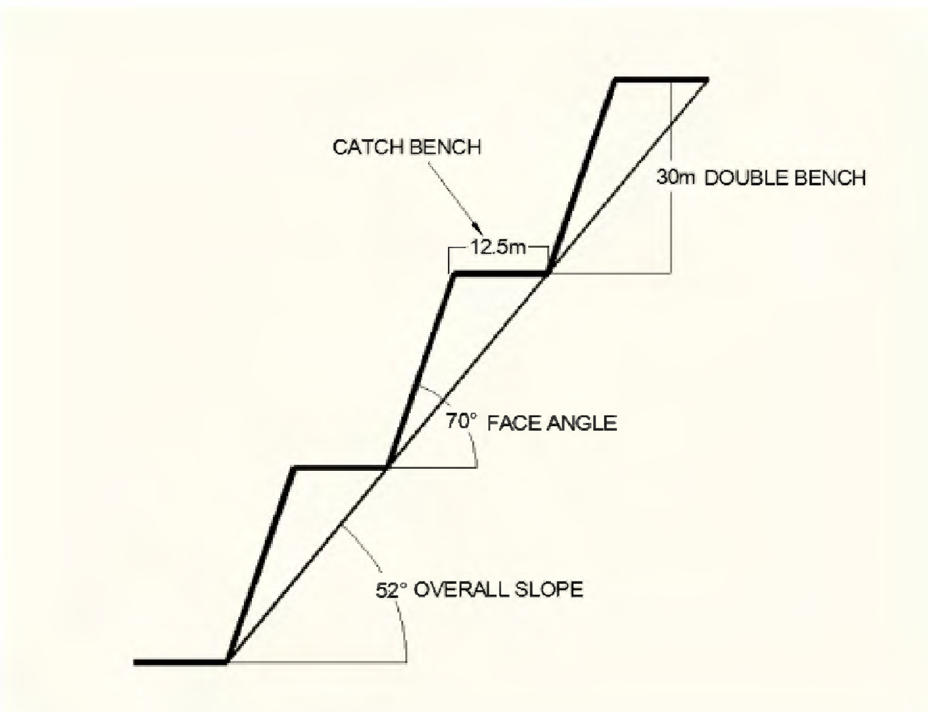
The economic pit limits derived from the pit optimization were used as a guideline for the detailed pit design. The pit design process includes smoothing the pit wall, adding ramps to access the pit bottom and ensuring that the pit can be mined using the initially selected equipment. The following section provides the parameters that were used for the detailed pit design.

16.6.1 Geotechnical Pit Slope Parameters

Met-Chem used an overall pit slope of 52° for the final pit walls. The final pit wall includes a 12.5 m catch bench for every two (2), 15 m high benches and accounts for a 70° face angle.

This design is based on Met-Chem's internal database for similar deposits in the region. Met-Chem recommends a complete pit slope analysis for the next stage of the Project. The pit wall configuration is illustrated in Figure 16.5.

Figure 16.5 – Pit Wall Configuration



16.6.2 Haul Road Design

The ramps and haul roads were designed with an overall width of 36 m. For double lane traffic, industry practice indicates the running surface width to be a minimum of 3 times the width of the largest truck. The overall width of a 400 st (363 tonne) haul truck is 9.8 m which results in a running surface of 29.3 m. The allowance for berms and ditches increases the overall haul road width to 36 m. A maximum ramp grade of 10% was used. This grade is acceptable for a 400 st tonne haul truck.

16.6.3 Mine Dilution and Mining Recovery

During the mining operation, material at the mineralization and waste rock contacts will not be separated perfectly. In order to account for this it was assumed that 2% of the mineralized material within the pit will not be recovered and sent to the plant as feed. It was also assumed that of the unrecovered material, 1% will be sent to the waste dump and 1% will remain in the ground. The waste rock quantity was therefore increased by 1%.

16.6.4 Pit Design

a) Deposit 3

The pit that has been designed for the Deposit 3 is approximately 2,600 m long and 1,500 m wide at surface with a maximum pit depth of 520 m. The total surface area of the pit is roughly 310 ha. There are two ramp accesses for Deposit 3 at the 168 m elevation on the Northeast corner of the pit. Both ramps descend down the East wall of the pit and provide access to different parts of the deposit. The lowest elevation in the pit is at -360 m.

Accounting for the mining recovery, the pit includes 163 Mt of Measured Mineral Resources, 306 Mt Indicated Mineral Resources for a total of Measured and Indicated Mineral Resources of 469 Mt and 147 Mt of Inferred Mineral Resources. The proportion of Inferred Mineral Resources that are contained within the pit shell is 23.9%. The average grade of the Deposit 3 is 25.0 % Fe, with an average DWTR of 28.2% and an average SiO_2 in concentrate of 3.9%. In order to access these in-pit resources, 117 Mt of overburden and 1,149 Mt of waste rock must be removed, resulting in a waste to mineralization stripping ratio of approximately 2:1. See Table 16.4 for In-pit Resources for Deposit 3. Figure 16.7 shows the Deposit 3 pit design.

Table 16.4 –In-pit Resources Deposit 3

Category	Mineralization (Mt)	Fe (%)	DTWR (%)	SiO_2 in concentrate (%)	Proportion (%)
Measured	163	24.3	26.9	3.9	26.5
Indicated	306	25.1	28.4	3.9	49.6
Measured + Indicated	469	24.8	27.9	3.9	76.1
Inferred	147	25.5	29.3	3.9	23.9

b) Deposit 4

The pit that has been designed for the Deposit 4 is approximately 2,000 m long and 800 m wide at surface with a maximum pit depth of 400 m. The total surface area of the pit is roughly 90 ha.

The ramp access for Deposit 4 at the 150 m elevation is located in the Southwest corner of the pit. The ramp descends down the South wall of the pit and provides access to the pit bottom. The lowest elevation in the pit is at -255 m.

Accounting for the mining recovery, the pit includes 119 Mt of Measured Mineral Resources, 72 Mt Indicated Mineral Resources for a total of Measured and Indicated Resources of 191 Mt and 10 Mt of Inferred Mineral Resources. The proportion of Inferred Mineral Resources that are contained within the pit shell is 5.0%. The average grade of the Deposit 4 deposit is 24.0 % Fe with an average DTWR of 27.2% and an average SiO_2 in concentrate of 8.2%. In order to access these in-pit resources, 13 Mt of overburden and 196 Mt of waste rock must be removed, resulting in a waste to mineralization stripping ratio of approximately 1:1. See Table 16.5 for In-pit Resources for Deposit 4. Figure 16.8 shows the Deposit 4 pit design.

Table 16.5 –In-pit Resources Deposit 4

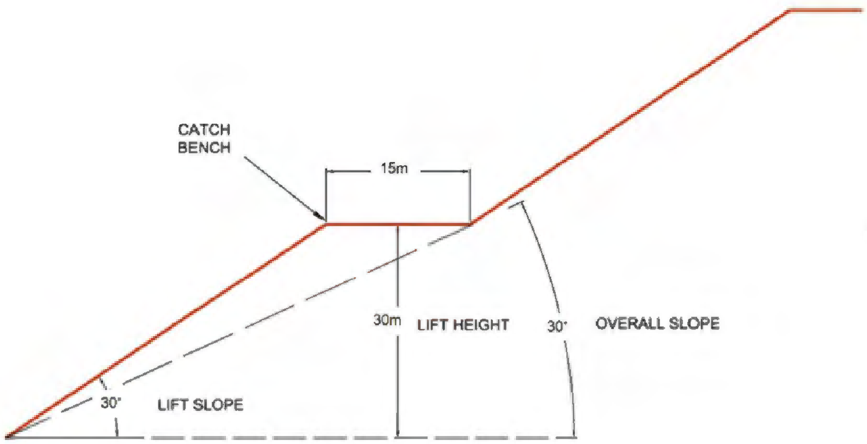
Category	Mineralization (Mt)	Fe (%)	DTWR (%)	Concentrate SiO_2 (%)	Proportion (%)
Measured	119	23.6	26.6	8.2	59.2
Indicated	72	24.6	28.1	8.2	35.8
Measured + Indicated	191	24.0	27.2	8.2	95.0
Inferred	10	25.0	28.4	8.2	5.0

16.6.5 Waste Dump and Overburden Stockpile Design

Three waste rock dumps were designed for both Deposits, 3 & 4. Each waste dump was design with an overall slope of 30° to account for the re-vegetation that is required for the closure plan. A total dump capacity of 558 million m^3 is required with a footprint area of 680 ha. The maximum height of the dump is approximately 200 m.

Figure 16.6 shows a typical section through the waste dump.

Figure 16.6 – Waste Dump Configuration



Two overburden stockpiles have been designed for the Block 3 & 4 deposits. Each overburden dump was design with an overall slope of 20° and has a joint capacity of 70 million m^3 and a combined footprint area of roughly 190 ha. These stockpiles will be 15 m high.

The dump and stockpile layouts are shown on Figure 16.1 at the beginning of this section of the report.

Figure 16.7 – Deposit 3 Pit Design

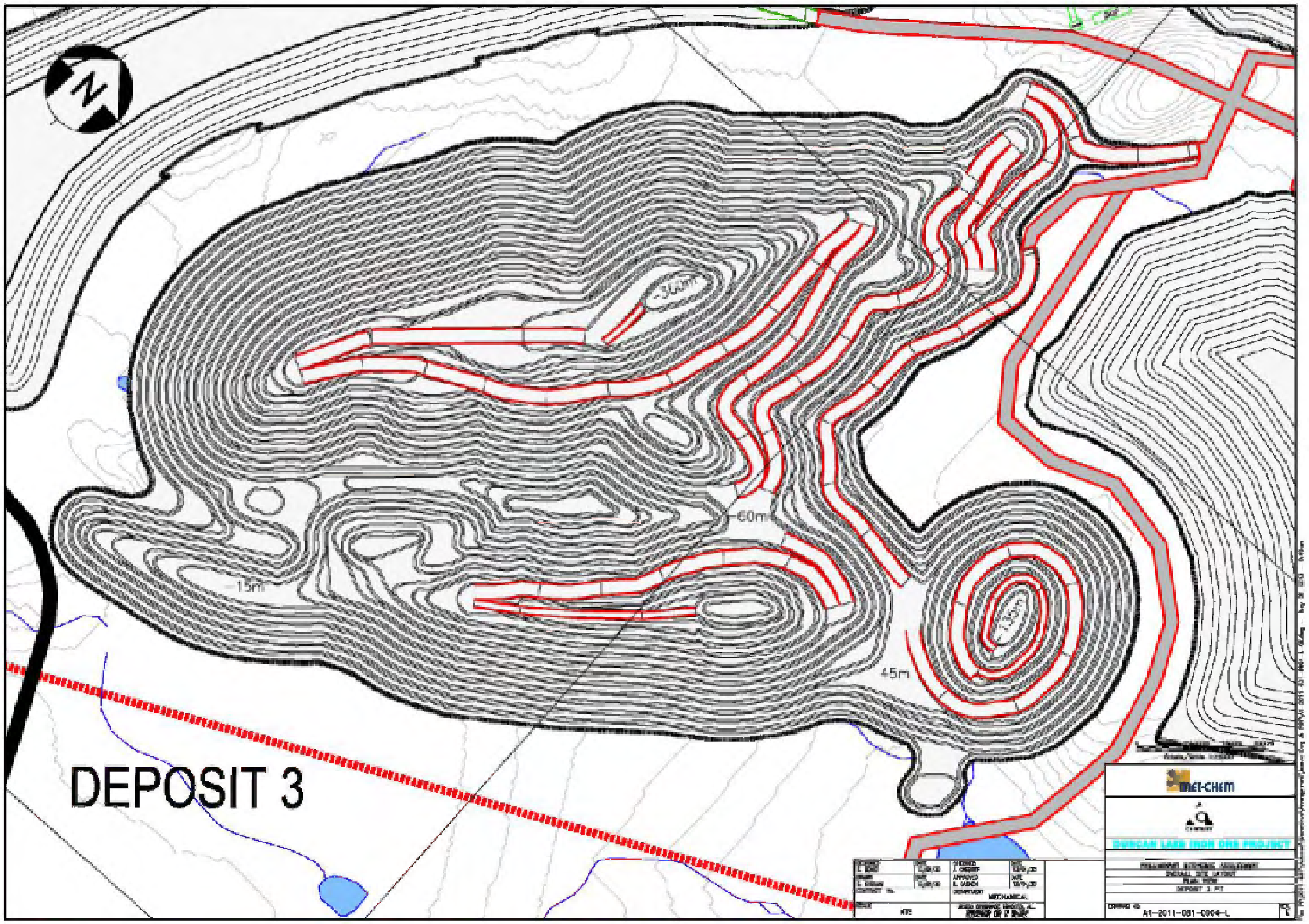
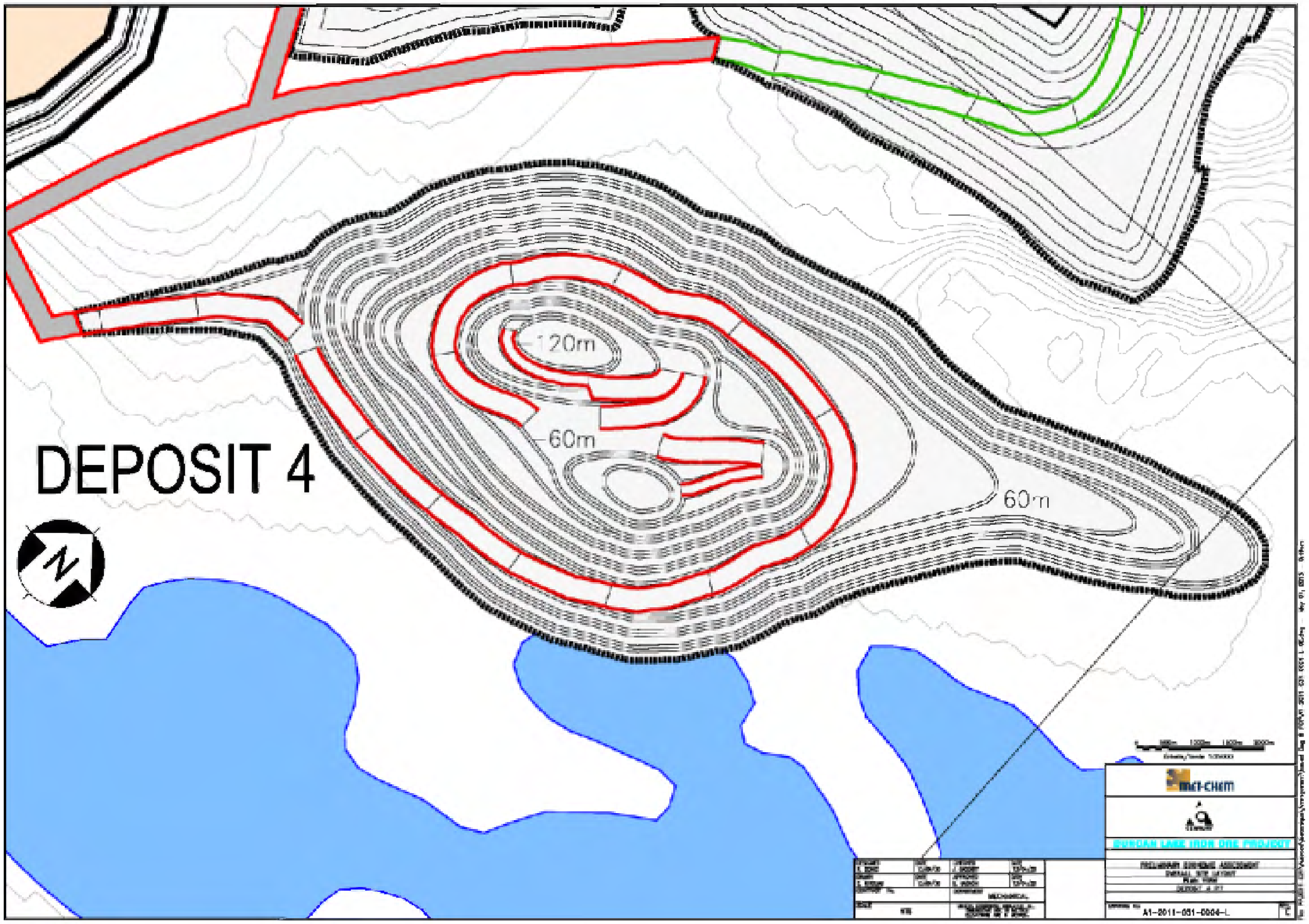


Figure 16.8 – Deposit 4 Pit Design



16.7 Mine Planning

16.7.1 Annual Production Requirements

The production target for the mine is to supply enough feed to the plant to produce a constant pellet production of 12 Mtpy with a silica content (SiO_2) of 5%. Year 1 of production is targeted at 9.6 Mtpy of pellets (80% of full production). This target is a function of the plant's ramp up before obtaining full operating capacity at 12 Mtpy in Year 2. The plant is expected to reach full capacity at the start of the second year of production. The mine is expected to supply an average of approximately 41.3 Mt of run of mine material per year (113,100 tpd).

The mine plan will attempt to supply a constant head grade close to the deposit's average along with a supply of material which has a SiO_2 in concentrate of 5%. Due to the grade variability in the deposit, achieving a constant Fe grade or constant weight recovery is not always feasible. The amount of feed to the plant is a function of the weight recovery and thus varies between 40 Mt and 43 Mt per year during full production.

Since Deposits 3 and 4 have different grade quality, they will be mined simultaneously to ensure a constant blend to the concentrator with a ratio of 74.5% coming from Deposit 3 and 25.5% from Deposit 4.

16.7.2 Work Schedule

Mining operations for the Project will be 358 days per year taken into consideration bad weather days, operating a fly-in fly-out operation around the clock on two (2), twelve (12) hour shifts. The fleet requirements and manpower are based on this work schedule.

16.7.3 Pre-Production

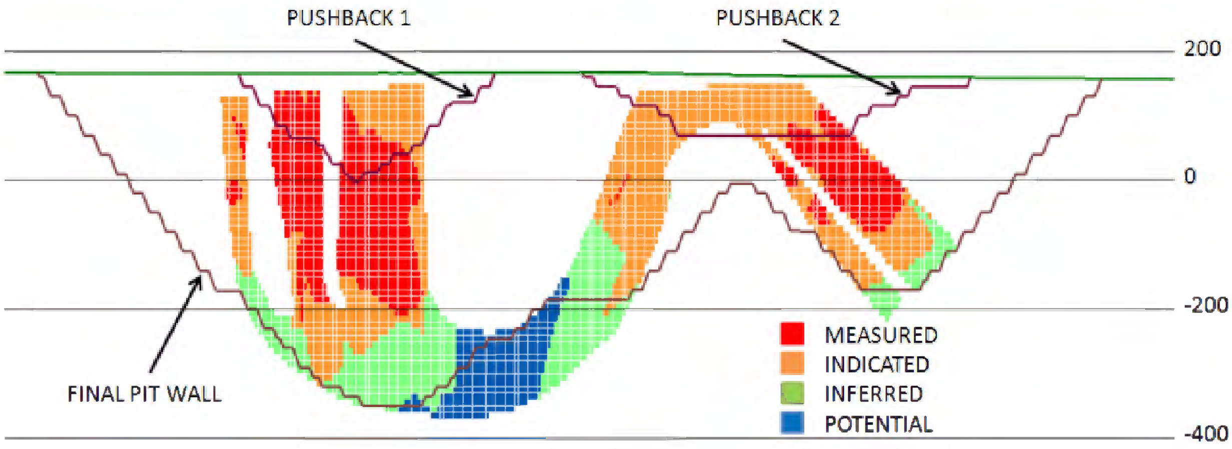
A pre-production period of two (2) years has been planned at the Deposit 3 & 4 to achieve the following objectives:

- Supply road construction material;
- Supply construction material for the tailings dyke;
- Strip overburden and waste rock to expose the mineralization of both deposits to have a blended feed to the processing plant.

16.7.4 Pushback Sequencing

In order to minimize the waste removal in the early years of production, the Deposit 3 will be mined in a series of three (3) pushbacks. Figure 16.9 shows a typical cross-section of Deposit 3, highlighting the pushback designs.

Figure 16.9 – Deposit 3 Pushback Design



16.7.5 Production Schedule (Mine Plan)

A production schedule was developed for the life of the Deposits 3 & 4. The schedule realizes the pre-production requirements and meets the annual production target.

The pre-production phase of mining will begin in Pushback 1 of the Deposit 3 and also in Deposit 4. A total of 9.0 Mt of material will be excavated in pre-production. This material can potentially be used for construction material of the tailings pond dykes. A total of approximately 7.0 km of mine road will be built during pre-production using suitable waste material from the mine. The remaining waste material will be hauled to the waste dumps.

During the first year of production, mining will progress in Pushback 1 of the Deposit 3 and also in Deposit 4. A total of 9.2 Mt of concentrate will be produced in Year 1 to meet a pellet requirement of 80% of full production (12 Mt of pellets per year), which accounts for the plant ramp-up. The stripping ratio will be 1:1 during Year 1.

Overburden stripping and waste removal will begin in Pushback 2 of Deposit 3 during the first year of production. The mineralization for Year 2 will be mined from Deposit 3 Pushbacks 1 & 2 along with Deposit 4 to reach a production target of 11.6 Mt of concentrate to meet a pellet of full production (12 Mt of pellets per year). During Year 2, Pushback 1 will supply the majority of the annual production requirements till it is mined out completely. The stripping ratio will be 1.2:1 during Year 2.

During Year 3, Pushback 2 will be supplying most of the production needs to achieve a 12 Mtpy pellet production along with feed from Deposit 4. The stripping ratio will be 1.4:1 during Year 3.

Pushback 3 of Deposit 3 (Final wall) will begin overburden and waste stripping in Years 2 and 3. Mining of Pushback 2 will end in Year 4 of the mine life. The Deposit 3

Pushback 3 and Deposit 4 will be mined until Year 20. The stripping ratio will increase to 1.7:1 in Year 5 to reach a maximum of 2.0:1 in years 6 to 10, will stay constant till year 11 to 15 and will drop off to 1.8:1 in year 16 to 20.

A summarized production schedule for the Deposits 3 & 4 is presented in Table 16.6.

Table 16.6 – Mine Production Schedule (Deposits 3 and 4 combined)

TOTAL Schedule	Units	PRE PROD	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6-10	Year 11-15	Year 16-20	Total
CONCENTRATE	Mt	0.0	9.2	11.6	11.6	11.6	11.6	57.8	57.8	57.8	228.7
Pellet	Mt	0.0	9.6	12.0	12.0	12.0	12.0	60.0	60.0	60.0	237.6
Total Mineralization to Plant	Mt	0.0	33.8	42.2	43.0	43.0	41.0	204.8	204.8	204.8	817
DT Weight Recovery	%	0.0	27.4	27.3	26.8	26.9	28.2	28.2	28.2	28.2	28.0
SiO_2 (in concentrate)	%	0.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0	5.0
Fe	%	0.0	24.4	24.4	24.3	24.3	24.9	24.9	24.9	24.9	24.8
Total Waste	Mt	9.0	30.4	52.2	57.4	65.8	69.4	409.3	409.3	372.3	1,475
Overburden	Mt	4.5	17.0	27.8	16.1	22.0	22.0	20.9	0.0	0.0	130
Waste	Mt	4.5	13.4	24.4	41.3	43.8	47.4	388.4	409.3	372.3	1,345
Total Material Moved	Mt	9.0	64.1	94.4	100.4	108.8	110.4	614.1	614.1	577.1	2,293
Stripping Ratio		n/a	0.90	1.23	1.33	1.53	1.70	2.00	2.00	1.82	1.8

16.8 Mine Equipment Fleet

The following section discusses equipment selection as well as fleet requirements in order to carry out the mine plan discussed in the previous section.

16.8.1 Haul Trucks

The haul truck selected for the Project is the Caterpillar 797. The nominal payload of the Caterpillar 797 is 363 metric tonnes, which results in a manageable fleet size for the Project. The following parameters were used to calculate the number of trucks required to carry out the mine plan.

- Mechanical Availability – 85%;
- Utilization – 95%;
- Nominal Payload – 363 tonnes (240 m^3 heaped);
- Shift Schedule – Two (2), twelve (12) hour shifts per day, seven (7) days per week;
- Weather delays – Seven (7) days per year;
- 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 ten (10) minutes for fuelling). Fuelling will be carried out once every two (2) shifts for 20 minutes;
- Job Efficiency – 91.7% (55 min/h; this represents lost time due to queuing at the shovel and dump as well as interference along the haul route);
- Rolling Resistance – 3%.

Haul routes were generated for mineralization, waste and overburden for three periods of the mine life (Year 1, Year 5 and Year 20) 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 Caterpillar 797 to complete each route.

Haul productivities (tonnes per work hour) were calculated for each haul route using the truck payload and cycle time. Table 16.7 shows the cycle time and productivity for the mineralization and waste haul routes in Year 4 as an example.

Table 16.7 – Truck Productivities (Year 4)

Destination	Cycle Times (min)					Productivity	
	Travel	Spot	Load	Dump	Total	Loads/h	t/h
mineralization	22.2	0.75	2.00	1.00	25.90	2.3	841
Waste	28.0	0.75	2.00	1.00	29.98	2.0	727

Truck hour requirements were calculated by applying the tonnages hauled to the productivity for each haul route. The number of trucks required was calculated assuming each truck has the 5,653 hours available to work in a full year. A contractor will be used during pre-production to remove waste and overburden hence trucks were not calculated for this period. In Year 1, 11 trucks are required to meet production needs for the ramp-up; 17 in Year 2; 26 in Year 5 and reach a peak of 37 in Year 11. The truck fleet required by year is presented in Table 16.9, Major Mining Equipment Fleet.

16.8.2 Shovels

The main loading machine selected for the Project is the CAT 6090FS hydraulic excavator. The CAT 6090FS is a shovel that will be suitable to handle the production requirements as well as the face heights expected.

The following parameters were used to calculate the number of excavators required to carry out the mine plan.

- Mechanical Availability – 85%;
- Utilization – 90%;
- Bucket Capacity – 90 tonnes (40 m^3);
- Bucket Fill Factor – 100% in mineralization, 100% in waste and 61% in overburden;
- Shift Schedule – Two (12), twelve (12) hour shifts per day, seven (7) days per week;
- Weather delays – Seven (7) days per year;
- Operational Delays – 90 min/shift (this includes 15 minutes for shift change, 15 minutes for equipment inspection, 40 minutes for lunch and coffee breaks and 20 minutes for fueling);
- Job Efficiency – 50% (30 min/h; this represents lost time due to waiting for trucks, cleaning up the loading area and relocating).

The CAT 6090FS excavator can load the Caterpillar 797 haul truck in four, 30 second passes for a total load time two minutes. Assuming there are trucks available to load, the excavator can load 30, 363 tonne trucks per hour for a theoretical productivity of 10,890 t/h. Accounting for mechanical availability, utilization, operational delays and job efficiency, each excavator has 2,876 available work hours in a full year. In order to mine the tonnages presented in the mine plan, three (3) shovels are required during Year 1, followed by four shovels for the remainder of the mine life.

A LeTourneau L2350 wheel loader equipped with a 68 tonne capacity bucket has been included in the fleet to be used as an alternate loading machine. This machine will be required to cover for the shovels when they are not operational. The loader will also be used for any re-handling that is required.

The shovel and loader fleets required by year are presented in Table 16.9, Major Mining Equipment Fleet.

16.8.3 Drilling and Blasting

The proposed blast patterns for mineralization and waste are presented in Table 16.8. Production drilling will be done using Caterpillar MD6640 drill(s). The number of drills required was estimated assuming an 85% mechanical availability, 90% utilization and a penetration rate of 20 m/h.

Four drills are required in Year 1, five from Year 2, three to five and seven drills are required from Years 6 to 25. The drill fleet required by year is presented in Table 16.9, Major Mining Equipment Fleet.

Blasting will be executed under contract with an explosives supplier who will store all the blasting materials and technology required by the mine.

Table 16.8 – Blasting Parameters

Parameter	Mineralization	Waste	Units
Bench Height	15	15	m
Blasthole Diameter	381	381	mm
Burden	8.1	8.1	m
Spacing	9.7	9.7	m
Subdrilling	3.8	3.8	m
Stemming	3	3	m
Explosives Density	1.26	1.26	g/cm^3
Powder Factor	0.39	0.39	kg/t

Table 16.9 – Major Mining Equipment Fleet

Description	Pre Prod	Year 01	Year 02	Year 03	Year 04	Year 05	Year 6 - 10	Year 11 - 15	Year 16 - 20
Truck - CAT 797	0	11	17	20	23	26	34	37	37
Shovel - CAT6090FS	0	3	4	4	4	4	4	4	4
Loader - L2350	0	1	1	1	1	1	1	1	1
Drill - CAT MD6640	0	4	5	6	6	6	9	9	9

16.8.4 Auxiliary Equipment

A fleet of support and service equipment was included to carry out the mine plan.

The remaining support and service equipment includes a fuel/lube truck, mechanic truck, tire handler, boom truck, tow trailer, transport bus, pickup trucks and light towers.

Table 16.10 provides a summary of the auxiliary equipment for a typical year (Year 4).

Table 16.10 – Auxiliary Equipment

Support Equipment			**# Units**
Track Dozer	Caterpillar	D11	6
Utility Excavator	Caterpillar	320D	4
Utility Excavator	Caterpillar	390D	2
Wheel Dozer	Caterpillar	854K	3
Road Grader	Caterpillar	24M	4
Secondary Drill	Caterpillar	MD5125	1
Water / Sand Truck	Caterpillar	793	3
Utility Haul Truck	Caterpillar	775	5
Utility Wheel Loader	Caterpillar	988	2
Light Tower	Magnum	MLT3080	20
Service Equipment			**# Units**
Fuel / Lube Truck	Peterbilt	365/12 kL	3
Mechanic Truck	Peterbilt	348	4
Tire Handler	Hyster	450 HDS	1
Boom Truck	Peterbilt	365	2
Tractor and Lowboy	Caterpillar	793	2
Mobile Crane (A)	RTC	80110	2
Mobile Crane (B)	RTC	8080	2
Transport Bus	Blue Bird	GMC Diesel	4
Pick-up Truck	Ford	F250	20

16.8.5 Manpower

The total mine operations workforce for the Project ranges from 251 employees in Year 1 to a maximum of 419 from Years 11 to 20. This workforce is comprised of staff as well as hourly employees. Table 16.11 shows the mine manpower requirements for Year 4.

Table 16.11 –Mine Manpower Requirements (Year 4)

Description	# Employees
Mine Superintendent	1
Maintenance Superintendent	1
Engineering Supervisor	1
Mining Engineer	4
Geologist	4
Grade Control Technician	4
Planning Technician	4
Surveyor	4
Pit Foreman	8
Drill and Blast Foreman	4
Equipment Operators	188
Fuel and Lube Truck Driver	12
Labourers	16
Dewatering Crew	8
Power Distribution Crew	8
Dispatchers	4
Trainers	4
Maintenance Foreman	8
Warehouse clerk	4
Maintenance Planner	4
Mechanics	56
Total Mine Workforce	347

All employees and contractors will be housed at a camp site located in Radisson.

16.8.6 Contract Mining

Contract mining has the effect of lowering the Project's capital expenditures but results in higher operating costs. A price for contract mining was used based on similar projects. A price of $3.00 per tonne of material mined was considered to undergo mining operations in pre-production.

17.0 RECOVERY METHOD

Based on metallurgical testing results reported in Section 13, the process design criteria were developed and process flow sheets as well as plant layouts were prepared.

The magnetite processing facilities are located between Deposits 3 and 4, closer to Deposit 3 because this deposit has better tonnage and iron grade. Processing facilities include crushing, grinding, magnetic separation and thickening of the concentrate.

The iron concentrate is then transported to the closest possible port location near Stromness Island where the pellet plant will also be located. The iron concentrate has to be transported to the pellet plant facility area via a pipeline. The pipeline will be located along the existing road and the road extension to the pellet plant.

The processing plant is designed to process an average of 113,107 tpd of Run of Mine ore ("ROM") to produce 31,644 tpd of final concentrate containing 67.6% Fe and 5% silica or less.

The ROM is trucked from the open pit to two (2) gyratory crushers and the crushed product is then sent to a 120,000 tonnes covered stock pile providing one day of storage for the processing plant.

The crushed product is then reclaimed with apron feeders and conveyed to the processing plant to feed the grinding circuit. The grinding circuit consists of three 22,000 kW SAG Mills where the material is ground and screen to a P_{100} of 3,360 µm. Screened product is then fed to cobber magnetic separators which rejects a portion of the non-magnetic product. Concentrate from these cobber magnetic separators is fed to a secondary grinding stage using six 9,300 kW ball mills operating in a closed loop with cyclones. The cyclones overflow has a P_{85} product of 75 µm (200 mesh).

A magnetic separation process is used to eliminate non-magnetic product from the cyclones overflow. The concentrate product is fed to a concentrate thickener. The non-magnetic product is thickened and pumped to the tailings pond.

The 65% solids concentrate from the concentrate thickener is then pumped to the port area via a pipeline to produce pellets.

17.1 Processing Plant Design Criteria

The processing plant is scheduled to operate 365 days per year, seven (7) day per week and 24 hours per day with 92% equipment availability.

The crushing plant is operating 16 hours per day at an average hourly rate of 7,069 t/h. An uncovered run-of-mine stock pile of 60,000 tonnes should be located near the crushing plant.

The PEA study is based on run-of-mine from Deposits 3 and 4. The processing plant is designed for a capacity of 113,107 tpd for an iron concentrate average production rate of

11,550,000 tonnes per year at a grade of 67.6% Fe. The final grind size will be 85% -75 µm (200 mesh).

A summary of design criteria is presented in Table 17.1.

Table 17.1 – Design Criteria Summary

Parameter	Unit	Value
Average processing rate	tonne / year (tpy)	41,284,059
Average processing rate	tonne / day (tpd)	113,107
Concentrate Average Production Rate	tonne / year (tpy)	11,550,000
Average head grade (Deposits 3 and 4)	% Fe	24.8
Crushing equipment availability	%	66.7%
Crushing plant operating time	h / day	16.0
Crusher average feed rate	tonne / hour (tph)	7,069
Crushed product stockpile capacity	tonne (t)	120,000
Overall processing plant availability	%	92.0
Processing plant average feed rate	tonne / hour (tph)	5,123
Concentrate Average Production	tonne / day (tpd)	31,644
Final product silica content	%	≤ 5
Overall weight recovery at 200 mesh	%	27.98
Iron Concentrate Grade	% Fe	67.6

17.2 Flow Sheets and Process Description

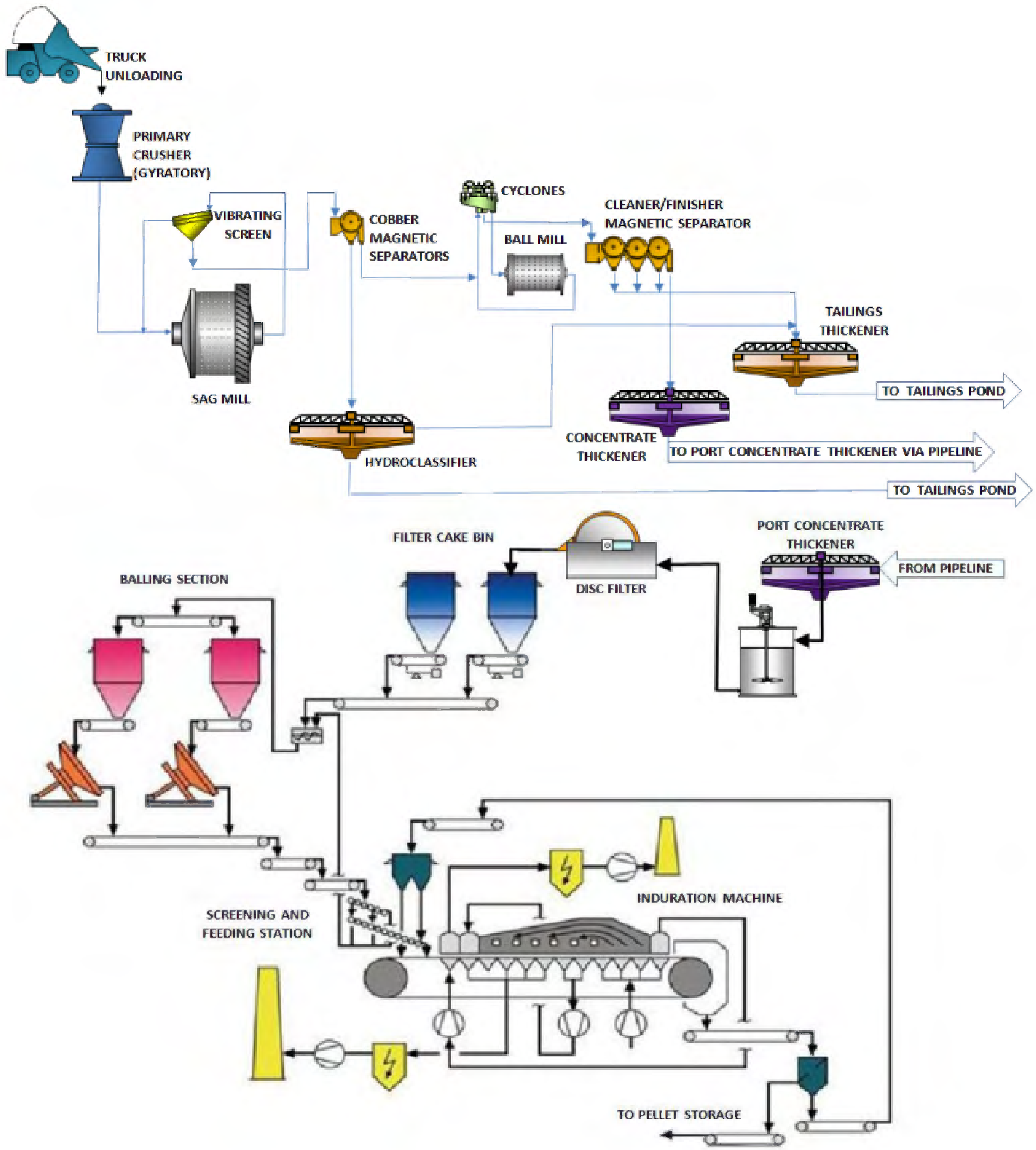
The process plant flow sheets are composed of the following areas:

- Area 100: Primary Crushing Area;
- Area 150: Stockpile Reclaiming Area;
- Area 200: Grinding Area;
- Area 300: Beneficiation Area;
- Area 350: Reagent and Air Services;
- Area 400: Port Thickening, Filtration & Pellet Plant;
- Area 450: Additives Preparation;
- Area 500: Tailings disposal and water management.

17.2.1 Simplified Flow Sheet

A simplified flow sheet of the process is presented in Figure 17.1 and summarizes different steps during beneficiation of the magnetite and pellet production. The equipment list is based on the flow sheet diagrams and equipment size is calculated based on the mass balance.

Figure 17.1 – Simplified Flow Sheet



17.2.2 Primary Crushing & Stockpile reclaiming area

It is considered at this point to use two identical gyratory crushers with hydraulic rock breaker. The crushers are housed in an enclosed building with two ROM pockets of sufficient capacity to receive the material from the mine trucks. The building includes an overhead crane, a maintenance area, a control room and electrical, sanitary and other services, and a dust collection system. The crushers' product will have a P_{80} of 198 mm. The crushed product will fall into two surge pockets from which it will be extracted by two apron feeders onto two belt conveyors to be carried out to two tripper conveyors. These tripper conveyors will discharge on a covered crushed product stockpile to be fed to the concentrator area.

The crushed product will be extracted from the stockpile by six apron feeders, located in a concrete tunnel below the stockpile. The crushed product will be carried by a conveyor system to three SAG Mills.

17.2.3 Grinding and Screening Circuit

a) Primary Grinding Circuit (SAG)

It is considered in this assessment that three SAG mills are used. Each SAG mill (12.2 m x 7.9 m) is equipped with a gearless motor of 22,000 kW and all required auxiliaries. The -198 mm crushed product will be fed directly by a conveyor into each SAG mill. Water is added to each SAG mill to control the slurry density inside the mill at 72 % solids weight.

Each SAG mill will be equipped with a 25 x 12 mm discharge grate and an 8 mm trommel screen with pocket lifter to return the oversize directly inside the mill by means of a sluice through the discharge trunnion of the SAG mill. The trommel undersize is fed to a pump box along with pulping water to maintain the solids at 65% in the pump box. From these SAG Mills discharge pump box, it is pumped onto vibrating screens with an opening of 3.36 mm.

The screens oversize is fed back into the SAG mills and the undersize flows into a pump box from where it is pumped to Cobber Magnetic Separators. The first step of magnetic separation ("cobbing") will reject about 10% of the liberated non-magnetic gangue. This gangue (final tails), will flow to a hydroclassifier. The cobber concentrate is directed to the secondary grinding circuit to further liberate the magnetite.

Based on Davis Tube Testwork ("DTT"), grind size of 75 microns was selected to process ROM from Deposit 3. It was also deemed acceptable to blend up to 29% of Deposit 4 mineralization with Deposit 3 mineralization.

b) Secondary Grinding Circuit (Ball Mills)

The secondary grinding circuit consists of six 9,300 kW ball mills operating in parallel in a closed circuit with hydrocyclones.

Each (7 m x 13.5 m) Ball Mill is equipped with all required auxiliaries. The product of the six ball mills is combined with the Cobber concentrate into six ball mills discharge pump boxes along with pulping water to maintain the solids at 47.5%.

This slurry is pumped to six hydrocyclones clusters where the underflow is returned to the ball mills to be reground.

The hydrocyclone clusters overflow will be directed to Cleaner/Finisher Low Intensity Magnetic Separators.

The grinding circuit is serviced by:

- Liner handler;
- Ball bucket;
- Ball magnet;
- Overhead cranes;
- Sump pumps.

17.2.4 Magnetic Separation

The magnetic separation circuit is composed of six trains of low intensity magnetic separation ("LIMS"). Each train has six triple drums separators. The slurry feeding the drums is at ± 25% solids from the hydrocyclones overflow.

The principal distributor is feeding each train through distributors # 1 to # 6 associated with each magnetic separation train.

The magnetic concentrate from each train is the final concentrate and goes directly to the concentrate thickener whereas the non-magnetic (magnetic separation tails) is final tails and go directly to two tailings thickeners.

17.2.5 Thickening and Process Water System

Tailings thickeners receive the pumped hydroclassifier overflow along with the tailings from the magnetic separators.

Underflow from the tailings thickeners and from the hydroclassifier is pumped to a double-compartment pump box prior to being pumped to the tailings pond at 55% solids.

The concentrate thickener which receives the final concentrate thickens the underflow to 65% solids. The underflow is pumped by positive displacement pumps via the slurry pipeline to the pellet plant.

The overflow of each thickener (almost water) is returned by gravity to the process water reservoir and back to the plant. The thickeners feed will have flocculant addition to produce clear process water. Future testwork has to be done without flocculant (see Section 13).

Water will be returned from the tailings pond to the process water reservoir to be used again in the Process Plant.

There will be one process water reservoir at the Process Plant. It will be an insulated and subterranean concrete water holder. The reason for having the reservoir below grade is to allow the large volumes from the thickener overflows to flow into it by gravity and thus avoid pumping. The reservoir will have enough capacity to supply the entire water requirement of the plant for 20 minutes. This allows for a quick restart after power failure or such other difficulties.

This complete the magnetic concentration process to recover 11.55 million tonnes per year of magnetite (Fe_3O_4) in a final concentrate containing 67.6% Fe and a maximum of 5% silica. Non-magnetic hematite iron and a minor amount of magnetite will be lost to tails along with the gangue minerals.

17.2.6 Services & Reagents

The plant will have air compressors and an instrument air compressor to provide compressed air for services and equipment. Diesel generators will be on stand-by to provide emergency power to operate critical equipment in case of power outage.

Flocculant is used to help sedimentation during slurry thickening. The selected flocculant is delivered in bags and dumped into a feed hopper and conveyed using screw feeder to an agitated mixing tank, fresh water is added to the mixing tank. The diluted flocculant is transferred to a holding tank and distributed to the thickeners feed slurries by metering pumps.

Mobile plant equipment will be provided (bobcats, lifts and front-end loader) to clean-up the diverse areas of the concentrator and to handle material as required by the operation.

17.2.7 Slurry Pipeline

The total length of the pipeline from the Processing Plant to the slurry reception point at James Bay (Stromness Island port area) is estimated at 135 km. It is assumed that the pipeline from the Processing Plant will intercept the highway going to Chisasibi and follow it to La Grande River. The annual production to be transported through the pipeline will be 11.55 Mtpy at 65% solids or about 1,064 m^3/h assuming an effective pumping time of 22.1 hours per day.

The pipeline feed is the concentrate thickener underflow at about 65% solids. It will be pumped by large, variable speed, high pressure positive displacement pumps located under the concentrate thickener.

Provision will be made to automatically add dilution water, as required, to maintain a specific density in the line. Provision will also be made to go to full water flow to push the slurry or flush the line and to add additive to prevent line corrosion. For maintenance purposes, by-passes will be provided to allow isolation of each pump without affecting the pumping capacity or the pressure requirement. Communication, instrumentation, control and protection systems, including slurry temperature measurement, customary on installations of this nature are included, and will be located in the concentrator control room. In case of power failure the pumps will also be connected to the concentrator emergency diesel generator plant.

The slurry will be received at the pellet plant complex which consists of two processing lines each of 6 Mtpy. Each line will supply pellets to the international iron market.

Test work with concentrate will be necessary to define the pumping characteristics of the slurry and allow sizing of pumps and pipeline. This should also include visit of the proposed pipeline line corridor.

17.2.8 Port thickening, filtration & pelletizing plant

a) Bentonite Grinding

Bentonite is used as the pelletizing binder.

Bentonite will be reclaimed from the bentonite storage facility and charged to a storage bin in the grinding building. It will be withdrawn by belt feeder and fed to a vertical roller mill.

Material which has passed between the mill rolls is lifted in a stream of air. A dynamic separator within the mill returns coarse particles directly to the mill. Finer particles are lifted out of the mill in a gas stream and are collected by a cyclone and bag filer and discharged through rotary valves into an aerated storage silo. Part of the cleaned air is returned to the mill.

The bentonite will be pneumatically transported to bins in the mixer area of each of the two 6 Mtpy pellet plants as required.

The bentonite grinding facility is sized to be operated on day shift (12 h) only. One bentonite grinding facility will serve both pellet plants.

b) Flux Grinding

Depending on the needs and requirements, limestone will be ground to pelletizing fineness in a wet ball mill in closed circuit with hydrocyclones.

This flux will be reclaimed from limestone stockpile and conveyed to a storage bin. The flux will be withdrawn from the bin by feeder and fed into a ball mill with an installed power of 2,600kW. Water will be added to give an in-mill density of 75% of solids by weight.

Ball mill discharge is collected in a pump box, diluted to approximately 50% solids and pumped to a hydrocyclone cluster. Coarse underflow is returned to the feed of the mill, while the overflow is pumped to the flux agitated storage tank. Limestone slurry is pumped from the storage tank to the induration machine discharge conveyor by variable speed pump.

One flux grinding facility will serve both pellet plants.

c) Concentrate Slurry Reception and Storage

The concentrate slurry will be received at the pellet plant at approximately 65% by weight of solids from a pipeline. The slurry will normally be fed directly to the slurry thanks but may be sent directly to the two thickeners depending on the incoming density and the density required for optimum filtering.

Underflow from the thickener (or direct feed from the pipeline) is fed to storage tanks. The slurry tanks have a total maximum storage of 8 hours.

Steam will be injected into the slurry storage tanks to maintain slurry temperature at approximately 45°C. The concentrate is then pumped to each filter line at a measured rate.

d) Filtering

For each of the two filter line, the concentrate is pumped to a pressure distributor and dewatered in six vacuum disc filters (and a seventh filter on standby). For each line, six vacuum pumps will be provided. Three snap blow compressors, also common to all filters, will provide air for cake release.

Another set of vacuum filters is provided for the second pellet plant.

The filter cake is transferred via conveyor to the filter cake bins in the mixing station. Filtrate and filter boot drain is pumped back to the thickener. Distributor and filter boot overflow slurry is returned by gravity to the filter feed tank.

Filtration test will need to be done at the next study stage to allow decisions on the most suitable filtration system.

e) Mixing

For each line, the bentonite will be fed to the mixer feed by screw feeders. Concentrate is withdrawn from cake bins by feeders and discharged into the mixer. The filter cake and binder are mixed in a horizontal mixer for each line. A third mixer which can serve either line is provided in case of break down. The mixed material will be transported by belt conveyor to the balling area. Reject green balls from the green ball screening system will be added to the mixed material downstream of the mixer.

f) Balling

A conveyor distributes the mixed material and green ball returns into nine balling discs feed bins. The mixed material will be continuously discharged from the balling feed bins and fed into nine balling discs.

Each disc discharges on individual belt conveyors that discharge onto a common collecting conveyor, which distributes the green pellet across a wide belt. The wide belt feeds onto a double deck roller screen whose function is to remove oversize and undersize. Green pellet fines will be recycled together with crushed oversize to the mixed material stream on the route between the mixer and the balling feed bins.

Balling test work will be required to define the balling characteristics of the concentrate and pot grate test on the produced green balls will be needed to establish the physical and metallurgical quality of produced pellets.

g) Induration

The pellets will be hardened on Straight Grate Induration machine.

Green pellets will be dried in two stages. The dried pellets will be preheated to a progressively higher temperature to calcine the flux and to initiate magnetite oxidation. The pellets will then be fired at approx. 1,270°C to provide the recrystallization and slag bonding which will give the pellets adequate strength. A short section designated as after-firing allows the heat front to completely penetrate to the bottom of the bed without the application of additional high temperature heat.

Cooling is accomplished in two stages by passage of ambient air supplied by a cooling air fan. The cooled pellets leave the induration machine at 100°C or less.

Five process fans provide process gas flow. The cooling air fan forces ambient air through the pellet bed. In the first cooling section, the air leaving the top of the bed, which contains a large amount of sensible heat from the cooling operation, is ducted through a direct recuperation header, without the use of a hot fan.

Process gas from the second cooling stage is transported by the updraft-drying fan to the updraft drying section of the grate.

Tests for the induration cycle are required (induration temperature per zone, gas flow, residence time, etc.) to define the grate factor. These tests should be done at the next study stage of the Project.

The final pellet grade will be 66.3% Fe and 5.1% SiO_2.

h) Pellet Loadout, Product Handling and Ship Loading

Product screening and chip regrind has not been foreseen in the pelletizing plant.

Product pellets discharged from the segregation bin will be transported by conveyor to the pellet stockyard.

The pellet stockyard will be located near Stromness Island port area. The stockyard provides the pellet export terminal with storage buffer for shiploading operations. The estimated stockpile area has 4 stockpiles for a total size of 300 m x 2,400 m, to handle production of 12 Mtpy and storage for 8 months.

Pellet products will be reclaimed by a slewing type bucket reclaimer with reclaiming capacity of 16,000 t/h. Belt conveyors will convey reclaimed iron pellet products from the stockyard to the shiploader at the berth for ship loading operation.

The berth at the port will have the capability to load ocean-going vessels. Two shiploaders will be standard long-travel shiploaders with slewing and luffing capability. The shiploaders loading capacity will be 12,000 t/h each.

i) Recycling Hearth and Side-Layer

Pellets discharged from each induration machine will be transported to a natural segregation bin. Larger pellets will segregate to the sides of the bin and will be allowed to overflow a side chute to return as hearth layer. Tramp material and agglomerates are prevented from being recycled in the hearth layer by the use of a static grizzly screen. The product pellets will be discharged from the bottom of the bin.

j) Induration Area Dust Collection

Process gas from the hood exhaust and windbox exhaust systems are cleaned for particulate matter by dry electrostatic precipitators. The dust collected will be slurried with water and pumped to the waste reclaim area.

k) Waste Reclamation

Dust from the electrostatic precipitators will be slurried with process water and collected together with slurry from scrubbers and washdown and is pumped to the thickeners. Thickener underflow is then sent to the filter feed tanks and thus all waste materials are returned to the process. Thickener overflow is returned to the process water tank.

l) Auxiliary Systems and Infrastructure for the Pellet Plant and Port

- Fuels

  The Induration process will operate with Bunker "C" heavy fuel oil with max 2% sulphur content.

  The induration pilot burners and the air heater of the bentonite grinding system will operate with No. 2 Fuel Oil (Diesel).

- Steam System

  Steam is required for fuel oil and slurry heating as well as building heating.

- Water System

  Water systems are provided for machinery cooling, gland & spray water, process water system and fire-fighting.

- Compressed Air Plant

A centralised system for the generation of plant compressed air and instrument air will be provided.

17.3 Mass and Water Balance

The process mass balance is calculated based on the selected flow sheets, metallurgical tests work and assay results obtained from SGS Lakefield and design criteria.

The water balance is divided into the following sections: primary grinding; secondary grinding; magnetic separation; thickening; port concentrate dewatering and pelletization plant. 5,953 m^3/d enters the process as moisture in the feed (5%), while 4,315 m^3/d leaves with the product (as product moisture and as evaporation during induration). The reclaim water pumps at the tailings pond recirculate 63,500 m^3/d of water to the process water reservoir in the Process Plant. All fresh water (14,237 m^3/d) is from a local river or lake. Water entering the tailings pond comes from slurry dewatering from the port (12,724 m^3/d) and two combined tailings slurry streams, the cobber tails and the cleaner/finisher tails, at 9,162 m^3/d and 57,490 m^3/d respectively. The tailings pond retains all the solids and 20% of the entering liquid, to have 15,875 m^3/d of water retained. Based on the process water balance, the tailings pond is not expected to show excess water but this is not taking into account precipitation.

17.4 Process Equipment

Based on the design criteria and the flow sheets, the equipment list and equipment sizes were established.

17.4.1 Long Lead Items

The estimated lead delivery time for major process plant equipment is presented Table 17.2.

Table 17.2 – Lead Delivery Time

Equipment	Source	Lead Delivery Time (weeks)
Gyratory Crushers	CITIC	52 to 54
SAG Mills	CITIC	54 to 58
Ball Mills	CITIC	50 to 56
Vibrating Screens	Weir Minerals	22
Magnetic Separators	Eriez	48
Hydrocyclones	FLSmidth-Krebs	23 to 26
Thickeners	Delkor Solid Solutions	52 to 58
Hydroclassifier	FLSmidth	28 to 36
Pellet Plant	Danieli	86 to 110

17.5 Power Requirement

Based on the installed power requirement for major equipment, the total installed power to run the process plant and off-site process is calculated and presented in Table 17.3.

Table 17.3 – Duncan Lake Project Mechanical Installed Power

Area	Installed power (kW)
Primary Crushing	5,606
Stockpile Reclaiming Area	1,896
Grinding Area	160,030
Beneficiation Area	22,725
Reagent and Air Services	491
Port Thickening, Filtration & Pellet Plant	52,115
Pellet Stockyard & Reclaiming	29,304
Tailings disposal and water management	2,237
Total	274,404

17.6 Plant Layout

The process plant facilities from crusher to thickener are composed of two main areas:

- The crusher building,
- Process plant building.

17.6.1 Crushing Building

The crusher building is located to the Southeast of the process plant building. The gyratory crusher is installed in a concrete and steel structure building.

Two (2) crusher discharge conveyors are inclined at an angle of 10^{o} and passes through 100 m long tunnels until they reach the surface.

The dump pocket is constructed of concrete and encloses a dust collector system and an electrical room. A rock breaker with control cabin is located on the surface level. Two overhead cranes are installed inside the crusher building for maintenance.

The 120,000 t live capacity crushed product stockpile is covered and three (3) concrete reclaim tunnels of 50 m long inclined at an angle of 12^{o} until they reach the surface are provided. Each has ventilation and dust collection systems.

One SAG mill feed conveyor, 150 m long with two apron feeders is installed in each tunnel. Outside of the tunnel the conveyor runs in an enclosed steel gallery supported on steel bents and towers with concrete foundations.

17.6.2 Process Plant Building

The concentrator building is conventional and is divided into four main areas:

- Primary Grinding and classification area;
- Cobber magnetic separators and Secondary Grinding;

- Cleaner/Finisher magnetic separators area;
- Tailings and Concentrate Thickening (outside the building).

The primary and secondary grinding areas have three 35 t bridge cranes and the magnetic separation has two 5 t bridge cranes.

The building is located longitudinally in a Northeast to Southwest orientation. The SAG mills feed conveyors enters the process plant building on the eastern side and the final product exits the building on the west side. The west side of the building has the large plant thickeners (tails and concentrate), the underground process water reservoir and the reagent systems. The hydroclassifier (cobber tails) is located inside the building, between the ball mills.

The main building structure consists of steel and steeped columns, roof trusses and purlins, wall struts, girts and bracings.

The electrical substation and administration building are located adjacent to the process building, on its North side.

The administration building comprises offices and lunch rooms, it is detached from the main process plant building.

The mechanical shop and laboratory are located inside the process plant building (North end).

The sample preparation and analysis areas are on the second floor and because of the proximity to the operation; it will facilitate the acquisition and delivery of samples from the operations. The sample preparation section has all facilities and equipment required such as crusher, pulverizer, oven drying, splitter, or-tap for screening, balances room, dust collection and ventilation systems.

The assay section has all facilities and equipment required for wet assay, atomic absorption and X-ray.

18.0 PROJECT INFRASTRUCTURE

18.1 Power

Electrical requirements are based on power demands, single line diagrams and electrical equipment lists.

a) Concentrator and Mine site

Based on preliminary power demand estimation, the power requirement is 217 MW at the concentrator and mine site. The 230 kV main power line will bring the power from Hydro Quebec Poste Radisson located approximately 25 km from the site. The main substation will include 4 transformers of 75/100/125 MVA, 230/34.5 kV, and will allow the continuation of operation by redundancy. Distribution on site will be provided by pole lines and required distribution equipment.

Emergency power is provided by diesel generators. Requirement is 10 MW at 4.16 kV for the process and 6 x 1 MW at 600 V for facilities including the mine, the camp, the administration building, the maintenance buildings, as well as critical heating.

b) Pellet plant and port site

Based on preliminary power demand estimation, the power requirement is 83 MW at the pellet plant and port site. The 120 kV main power line will bring the power from Hydro Quebec Poste Chisasibi located approximately 20 km from the pellet plant site. The main substation will include 3 transformers of 50/67 MVA, 120/13.8 kV, and will allow the continuation of operation by redundancy. Distribution on site will be provided by pole lines and required distribution equipment.

Emergency power is provided by diesel generators. Requirement is 3 MW at 4.16 kV for the pellet plant and 5 x 1 MW at 600 V for facilities including the port, the camp, the administration building as well as critical heating.

18.2 Port

Wastikun Island was the preferred port location initially for the on-shore features, easy access and ease of construction. Stromness Island was the second choice but proved to provide deeper water and better approach channel and allows for larger vessels to be docked. Stromness Island location became the base case for the present PEA study. In fact, maximum draft at the Wastikun Island site corresponds to a 75,000 dwt ship while draft at Stromness Island corresponds to 200,000 dwt ship. This opens the option for direct shipping to China or Europe.

The port design was based on underwater contours available from current hydrographic charts and visual observation during site visit, no geotechnical information is available. It is assumed that the bottom of the bay in the area consist of a rock surface with very little overburden. It is fairly assumed that no dredging will be required.

From the information gathered, the recommended structure for the wharf is a gravity full face type structure, sheet pile cellular design complete with intermediate arcs. The cell design is a proven construction design for harsh conditions. The access dykes will be built with rock fill available in the area and from construction.

The berths (2) will be located on the inside face of the wharf and will incorporate a breakwater on the outside. The design integrates both the wharf and the breakwater and serve as an access structure for the conveyors, pipelines and truck traffic. The impact of waves will be minimized.

Two high capacity 12,000 tph shiploaders were selected for loading the 12 Mtpy of pellets.

Additional details on the port facilities for DLIP can be found in the report titled "New port infrastructure and shipping alternatives" in Appendix D.

18.3 Concentrate Pipeline and Water Reclaim Pipeline

The concentrate slurry pipeline will be 135 km long, 20" diameter, with external pipeline corrosion coating and will be buried along the road. The pumping system is positive displacement with stand-by provision. The process plant thickening system will feed the slurry while terminal tanks and dewatering will be included at the pellet plant. Pressure monitoring stations will be installed each 30 km.

The pipeline will be operated from the concentrator pump station with SCADA and PLC allowing remote and manual control. A leak detection system will be included in the SCADA system.

The water reclaim pipeline will be following the same path from the pellet plant but will discharge to the tailings storage area, located initially at 3 km of the concentrator and progressively relocated as the tailings storage system is expanded.

18.4 Main Access Road and Site Roads

The Project will be accessible via the James Bay Road, Matagami - Radisson. This road is an all-year asphalt road.

The concentrator area will be close to the road and south of Radisson near Desaulniers Lake. The Project includes a 2.2 km deviation of the James Bay Road to be executed at Year 6 when mining of Deposit 3 will be too close to the existing road.

This James Bay Road deviation will follow the same specifications as the existing road complete with asphalt coating.

The pellet plant and port areas will be accessible from the James Bay Road heading west between Radisson and La Grande towards La Grande 1 generating station. The pellet plant will be north of the La Grande River and on the east shore of James Bay at Stromness Island.

To access the pellet plant area the existing road will have to be extended approximately 16.5 km.

The road extension to the pellet plant will be 14 m wide and constructed in layers from three types of fill:

- The first 1.25 m layer will consist of rock fill either from the quarries or blasted material from road cuts;
- The second layer, 600 mm thick, will be made up of crushed rock to comply with MG 112 (100 mm - 0);
- The top layer, 150 mm thick will be made up of crushed rock to comply with MG 20 (20 mm - 0).

Since the concentrate pipeline will be installed adjacent to the road extension to the pellet plant the road extension routing will take into consideration the concentrate pipeline design requirements for grades and turning radius.

Site roads will provide access to the following:

- The concentrator from the James Bay Road;
- The crusher from the concentrator;
- The explosive plant from the mine haul road;
- The tailings ponds from the main haul road;
- The pellet plant area permanent camp from the main access road;
- The port from the pellet stockpile area.

The site roads will be constructed for light to medium traffic and have widths of 10 m to 12 m.

18.5 Gate House and Parking

At both sites, allowance for a modular type gate house and fenced parking area is provided to control accesses to sites.

18.6 Explosive Production and Storage Facilities

It is assumed that a contractor will store and prepare the explosive for the mining activities. The contractor will be responsible for the supply and installation of the explosive production and storage facilities. Civil and concrete foundations works were included in infrastructure based on similar project requirements.

18.7 Maintenance Facilities

The maintenance facilities building will include the following:

- Four major mining equipment maintenance bays;
- Three light maintenance bays;
- One vehicle wash bay;

- One oil/water separation bay;
- A small warehouse for routine maintenance parts;
- Dry, locker room and offices;
- Lunch room;
- Restrooms.

The maintenance bays will have large garage doors (Megadoors) to accommodate mining trucks. The building will be expanded at Year 5 adding 3 large truck maintenance bays to accommodate additional mining trucks added to the mining equipment fleet.

The maintenance building is located near the crusher building.

There is no maintenance building at the pellet plant/port area. The dry and change rooms will be located within the pellet plant building.

18.8 Camp Accommodations

The principal permanent residential camp will be located at Radisson and will have capacity for 500 people. The modular type construction will require minimum foundation. It will require less labour and time to install than conventional construction. It will comprise single-occupancy bedrooms with individual shower and toilet facilities, lounges, recreational areas, a fitness area, kitchen and lunch rooms.

A 200-person permanent camp will be installed at the pellet plant/port for employees working in these areas. It will include similar sleeping quarters to those in the Radisson camp; a recreation room, a fitness area, a lunch room and a kitchen.

During construction, a 500-person construction camp will be installed near the concentrator area. The total construction work forced will be 1,000 workers at peak. The construction camp will be demobilized at the end of the construction period.

During construction, a 1,000-person construction camp will be installed near the pellet plant area. The total construction work forced will be 1,200 workers at peak. The construction camp will be demobilized at the end of the construction period.

18.9 Administration Office Buildings

Administration office buildings will be located at the concentrator and at the pellet plant. The buildings will house the area managers, personnel management, local accounting, engineering, safety, environment, conference rooms, lunch room and rest rooms.

18.10 Airstrip

No project specific airstrip is required since the project is close to La Grande Airport. The airport is located approximately 30 km south of Radisson and will accommodate fly in/fly out of employees and visitors and some of the light urgent freight.

18.11 Warehouses and Storage

A conventional structural steel building warehouse will be located at the concentrator area as well as at the pellet plant area. The buildings will have a concrete slab on grade and will be insulated and heated.

Also at both sites, there will be one cold warehouse. The buildings will not be insulated nor heated. The buildings will be fold away type. They are easy to install and do not require large concrete foundations. There will be no slab on grade. The buildings are to store equipment and material mainly to protect them from the elements.

Also at both sites, there will be laydown areas for large equipment and material storage.

18.12 Emergency Vehicle Building and First Aid

An emergency vehicle conventional building will be located at the pellet plant area. A three-door garage will be built for the fire truck, the rescue truck and the ambulance.

First aid facilities will be located in the same building and include sanitary services, one office for a nurse, and waiting, examination, recovery rooms and stock room.

At the concentrator area, a smaller emergency building will be constructed. Due to the proximity of Radisson only a rescue truck and first aid will be required, the ambulance and fire truck are assumed to come from Radisson in case of emergency.

18.13 Site Communications

Communication facilities are included in each area but are not detailed at this stage. Communication towers will be installed at the mine-concentrator site and pellet plant-port site and also along the road as required.

18.14 Assay Laboratory

The fully-equipped assay laboratory will be located in the concentrator building and in the pellet plant.

18.15 Trade Shops

Trade shops for mechanical, electrical, instrumentation, carpentry work and repairs will be located in the concentrator and pellet plant buildings.

18.16 Fire Protection

A fire loop and hydrant will be installed around the buildings for fire protection. The fire protection inside buildings is included with the buildings.

18.17 Truck Scale

A truck scale is provided at the concentrator area only to control bulk material and liquid delivery and shipping.

18.18 Water and Waste Water Management

At both sites, fresh water will come from an adjacent lake. A floating barge will house the pumps and electrical equipment and will be fitted with a de-icing pump system to keep the barge surroundings free of ice during the winter months. Water will be distributed to the different buildings and camps and will be treated for potable use. Bottled water will be distributed to remote areas when more economical.

All sanitary waste water will be collected and directed to sanitary treatment plants. These will be located at the temporary construction camps and at the port area permanent camp. Smaller units are also included at the explosive plant and airstrip. The Radisson permanent camp will be connected to the town service system.

The systems are from Seprotech and consist of standard Rotordisk® treatment plants, typically used for remote mining operations.

18.19 Waste Management

Waste will be separated into four types:

- Kitchen waste;
- Metals;
- Garbage;
- Wood and other dry construction material.

All waste is assumed to be collected separately and sent to Radisson to be disposed in town waste management controlled areas. No on-site incinerators have been provided for.

18.20 Pellet Storage Area

Near the pellet plant and ahead of the port facilities, a pellet storage area is required to store the pellet prior to loading on vessel. Due to the shipping period restricted to ice free months, the pellet storage capacity will have an 8 months pellet production capacity or 8 million tonnes of pellet storage capacity. To reduced length of stockpiles, four 2,360 m long stockpiles were designed. The storage pad required will be 300 m wide by 2,400 m long.

The pad elevation was designed to optimize the cut and fill ratio and to have enough rock available from the cut to provide for fill material.

The ocean vessels need deep water and the port was located at the end of Stromness Island which is approximately 10.7 km from the shore and the edge of the pellet storage pad.

A conveyor support and access road to the port will be constructed and it will also support the HFO and diesel pipelines coming from the wharf to the fuel storage area near the pellet plant.

18.21 Fuel Storage

The principal fuel storage facility will be located at the port area close to the pellet plant. The design criteria for storage capacity are 8 months storage capacity at the port for all fuels and 7 days capacity of diesel fuel at the concentrator area for mining equipment and services. All fuel tanks will be installed within a bermed area. The fuel farm storage areas will all be lined with geo-membrane to contain leaks if any.

The fuels stored at site will be:

- HFO for pellet plant.
- Diesel for mining equipment, mobile equipment and service vehicles.
- Gasoline for small tools and equipment, all-terrain vehicles and snowmobiles.

18.22 Tailings Storage and Management

Tailings storage and management details are presented in Section 20.

19.0 MARKET STUDIES AND CONTRACTS

The QP has relied on long term iron ore pricing and market assumptions prepared by independent consulting firm Raw Materials & Ironmaking of Bethlehem Pennsylvania, who prepared an independent marketing and sales price analysis of the Duncan Lake Iron pellets. The report, titled “Century Iron Mines Ore Marketing Study”, was prepared by Dr. Joseph J. Poveromo, a world renowned iron and steel marketing specialist and president of Raw Materials & Ironmaking. The report is dated February 25, 2013. The QP has reviewed this report and the results support the assumptions in this technical report.

Met-Chem has summarized the findings of Dr. Poveromo below:

The DLIP Project will start with the upgrading of a lower grade magnetite mineralization to produce a fine sized concentrate at 67.6% Fe and 5.0% SiO_2. This concentrate will be conveyed by slurry pipeline to a pellet plant located at a James Bay shipping point. The concentrate will be too fine sized to effectively transport it by vessel so we will consider blast furnace pellets as the only product. In any event the Atlantic Basin pellet feed market will be in oversupply, with the demand focused in China, so this absence of a pellet feed product will not be detrimental.

The pellet plant will produce a blast furnace acid pellet with 66.3% Fe and 5.1% SiO_2 with a very low Al_2O_3 level and low levels of other impurities and residual elements. Such a pellet will be well suited as a complement to high sinter burdens in steel plants in Asia (specifically China) and Europe. The very low (0.30 %) Al_2O_3 level will advantage DLIP for Asian ironmaking operations which have issues with high Al_2O_3 levels generally encountered with Australian iron ore. In Europe, the Duncan Lake acid pellet quality will be comparable to other North American produced pellets, well accepted in European blast furnaces.

The near term blast furnace pellet market globally suggests a potential oversupply, so the off take agreements by WISCO and MinMetals, along with a potential contract with one or more European customers, will be essential to guarantee the revenue stream for this project. On a longer term basis, the reduction in lump ore supply due to quality issues in Australia and virtual elimination of lump ore exports from India and Brazil will increase the demand for pellets.

The long term pellet price will follow from the long term fines price plus a pellet premium. A long term pellet premium of USD 35/t will be assumed; it is supported both by market evidence and the required price differential to justify pellet plant investment.

The consensus opinion among iron ore experts is that the so called long term equilibrium price of iron ore fines (62 % Fe, CFR China) will be driven by the costs of the higher cost Chinese production as this production would ultimately shut down if iron ore prices stay well below this level for a sustained time period. This high cost level is in the vicinity of

USD 120/t to 130/t so the choice of USD 125/t seems reasonable. However there will be periods of higher and lower prices.

The long term fines price, under a worst case scenario, could fall below USD 100.00/t with a "perfect storm" of many new merchant projects, much steel company equity iron ore investment, new steel plants in iron ore rich areas and a levelling off of global steel demand.

However, long term higher prices of USD 125/t, driven both by the costs of the higher cost producers and new iron ore projects, are also driven by:

- Grade depletion globally means that more ore is needed for the required Fe units;
- Shortages of equipment, supplies, labor and skills will not only delay new projects but impact on availability at existing operations; the tire shortage of several years ago impacted existing mines;
- Misguided government and steel industry promoted policies in restrictions of both iron ore exports and mining itself will cause India's iron ore industry to grossly underperform;
- Natural disasters, floods, typhoons, etc., could impact on both mining operations and shipping;
- Political unrest could affect some new mines being built in more unstable regions such as West Africa.

Aside from the real reasons for supply reductions, a major "contrived" reason for reduced supply could be oligopic behavior by the "Big Three" VALE, BHPB and Rio Tinto, in slowing down expansions or simply reducing production at existing less favored sites when ore prices drop too low, as a means of inducing shortages that will propel spot prices upward.

20.0 ENVIRONMENTAL CONSIDERATIONS, PERMITTING, AND SOCIAL COMMUNITY IMPACT

20.1 Objectives

Le Groupe Desfor was selected by Century Iron Mines Corporation (CIMC) as the environmental consultant for the baseline studies of the Duncan Lake Iron Project. The baseline studies are the first step of the project notification process prescribed by the Environment Quality Act ("EQA", R.S.Q., c. Q-2). The main objective of the baseline studies is the characterization of the biophysical and human environment of the project. Le Groupe Desfor has the mandate to review and complete the environmental studies in accordance with the James Bay and Northern Quebec Agreement ("JBNQA"), as well as with the EQA and other applicable acts and regulations.

20.2 Site Description

20.2.1 Duncan Lake Iron Project Area

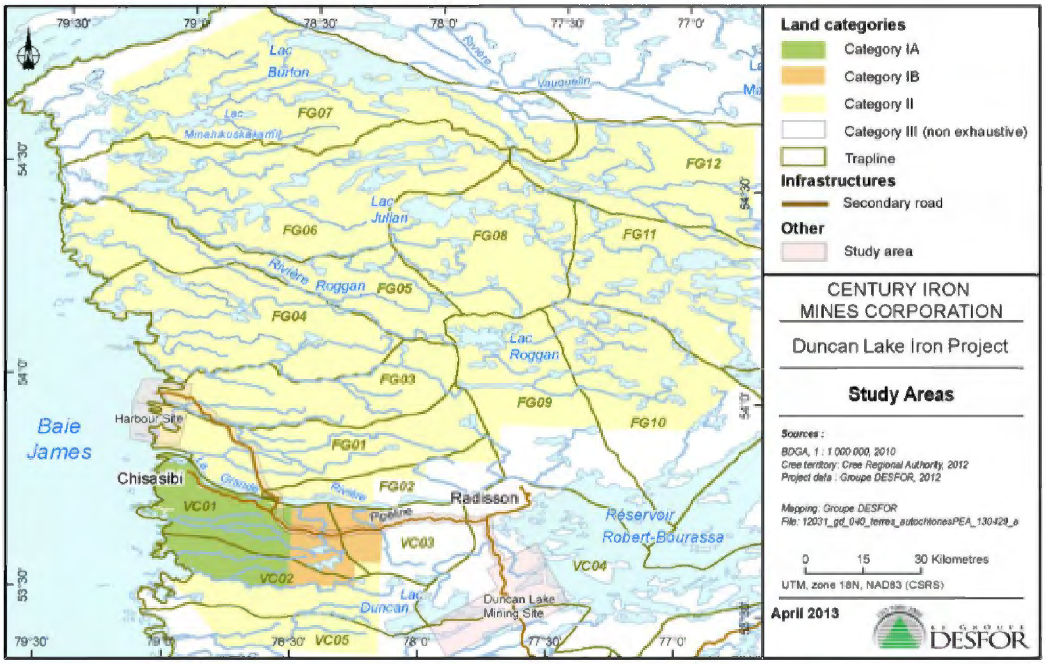
The Duncan Lake Iron Project ("DLIP") is located in Northern Quebec, on Eeyou Istchee James Bay territory. This region is governed by the Municipalité de la Baie-James, soon to be replaced by the Eeyou Istchee James Bay regional government according to the Agreement on Governance signed on July 24, 2012 by the Cree Nation and the Government of Québec (GCC 2012). The mining area planned for the project is located on category III lands, other major infrastructure of the project such as the harbor area, pellet plant and pipeline corridor are located on category I and II lands (L.R.Q., Chapter R-13.1). The overall project stands on traditional trap lines of the Chisasibi community, the nearest Cree community, located 80 km northwest of Duncan Lake. The planned mining area is located 40 km south of Radisson, 570 km north of Matagami and 10 km south of La Grande airport (Boulé et al., 2011).

The actual design of the project aims to retrieve iron from two deposits. The deposits identified are located in an area covering over 5 km^2. The mining and milling facilities will be located close to the deposits, which has the largest Measured, Indicated and Inferred Mineral Resources out of all the deposits located in the DLIP. The crusher and concentrator facilities will be built close to the deposits. Once the run of mine material is crushed, it will be concentrated to form a slurry containing about 65% of solid material. The slurry will be pumped in a pipeline, from the mill to harbour facilities required to ship the iron final product. The slurry pumped from the mining area will travel across a corridor of about 135 km to the harbour facilities, where the pellet plant will be located. With respect to the actual project description, specific environmental impacts are to be assessed and alternatives are now being studied regarding some of the infrastructure location, such as the tailings ponds, the waste deposits and other infrastructure with potentials environmental impacts.

20.2.2 Harbour Infrastructure

Stromness Island is located in the northern area of La Grande River estuary, close to the community of Chisasibi, about 10km north-west on the other side of the river (Figure 20.1). The economic and feasibility assessment study conducted by Met-Chem (2012) has identified the Wastikun Island site as the best location for the harbour infrastructure. The Stromness Island was defined as a better location during the present Study and became the base case. The Stromness harbour covers an area of 30 km^2. As described by Portha, the approach channels to Stromness Islands requires less turns, are deeper and offer a better draft for the berths than Wastikun Island.

Figure 20.1 – Study Areas - Duncan Lake Iron Project



20.2.3 Pipeline Corridor

As mentioned previously, the slurry produced by the mill will be pumped to the harbour facilities through a pipeline. The preliminary pipeline location follows the actual road network in a north-west direction toward La Grande 1 power station, between latitudes 53° 36'N and 53° 41'N and longitudes 78° 09'W and -77° 25'W (Boulé et al., 2011). Alternative locations for the pipeline installations will be studied with regards to various elements such as infrastructure encountered and land uses.

20.3 Jurisdiction, Applicable Laws and Regulations

20.3.1 James Bay and Northern Québec Agreement ("JBNQA") and Quebec-Cree Agreements

The DLIP is located on a part of the territory subject to the JBNQA (MDDEP, 2012a). Approved in 1975 by the Crees and Inuit of Northern Quebec, the governments of Quebec and Canada as well as by Hydro-Québec, the James Bay Energy Corporation and the James Bay Development Corporation, the agreement defines the rights and guarantees related to, among other issues: resources and wildlife management, environmental protection, health and social aspects and economic development.

Mining development projects on the Eeyou Istchee James Bay territory must consider and include important issues according to the specifications of the JBNQA and, in particular, sections 22 and 24 of the JBNQA require a special attention (MDDEP, 2012b). The environmental assessment process must pay a special attention to the following questions:

- The right to implement development projects in the region of James Bay and northern Quebec;
- The native nations participation in the application of the environmental protection regime;
- The protection of native nations, their social environment as well as their economies;
- The protection of hunting, fishing and trapping rights of the native nations;
- The reduction of the undesirable repercussions resulting from the development;
- The protection of wildlife resources and ecosystems in the region;
- The protection of rights and interest of native nations and non-native peoples.

In addition to the JBNQA, the Cree Nation has other agreements with the governments, such as the Québec-Cree New Relationship Agreement (Gouvernement du Québec and the Crees of Québec, 2002), also known as the "Paix des Braves" Agreement, and the Governance Agreements (Gouvernement du Québec and the Cris d'Eeyou Istchee, 2012). These agreements are also relevant for the mining industry because they guide Cree participation and collaboration during the lifetime of mining projects (Hernandez, 2012). It is now accustomed that promoters and native nations negotiate agreements regarding impacts and benefits of mining projects planned on the territory (Wilkinson, 2012). This kind of agreement is currently being discussed between the promoters and the Cree Regional Government and Chisasibi official representatives.

20.3.2 Quebec Environmental Assessment Process

Following article 22.2.3 of the JBNQA, all applicable federal and provincial laws of general application respecting environmental and social protection apply in the territory to the extent that they are in accordance to provisions of the Agreement and in particular section 22. The main politics and governmental orientations concerning environment, resources management, energy, tourism, and public security apply to mining project (MDDEP, 2012b).

The Québec Environmental Quality Act ("EQA") includes two chapters; the first gives the general provisions of the EQA and the second gives additional provisions applicable to the James Bay and Northern Quebec Region. In order to receive the certificate of authorization confirming the approval to proceed with a mining development project, the environmental and social impacts assessment and review procedure must be followed. According to Chapter 1 of the EQA, sectional certificates of authorization must be issued with regards to quarries and sand pits, water and sewer mains, camps, etc.

The environmental protection regime included in chapter 2 of the EQA aims to ensure participation of the Cree or Inuit nations in the environmental assessment process. This participation is assured by the mechanism of consultation and representation set by the EQA act. The nature of the preliminary information to be included in the baseline study is described in the Regulation respecting the environmental and social impact assessment and review procedure applicable to the territory of James Bay and Northern Québec (Q-2, r. 25). In addition to the EQA, other provincial acts must be considered in the implementation of the baseline study design.

According to the *Directive pour la réalisation d'une étude d'impact sur l'environnement d'un projet minier* (MDDEP, 2012b); the environmental assessment must include the objectives of sustainable development to the project conception. The three main objectives of sustainable development are: environmental integrity conservation, social equity enhancement and economic efficiency improvements. The Protection Policy for Lakeshores, Riverbanks, Littoral Zones and Floodplains will be respected, in particular for the harbour facilities area, located in the littoral zone, and other constructions located in riparian zones or in flood plains. Requirements for environmental and social impact assessments of mining projects are further refined in the Directive 019 (MDDEP, 2012c).

Based on the available information, planned activities for the DLIP which requires the following but non-exhaustive list of certificates of authorization according to provincial regulations include: mining activity, borrow pits, refuse material, construction of an electric transportation line, works in aquatic environment, airstrip, domestic wastewater, drinking water, atmospheric emissions (dust collector).

20.3.3 Federal Environmental Assessment Process

Native nations' rights under the JBNQA are constitutionally protected and paramount over all other laws by virtue of the terms of the JBNQA presented in the previous section, by virtue of section 35 of the Constitution Act (1982) and under federal legislation (GCC & CRA, 2000). Environmental assessment is required at the federal level according to various acts, in particular if the projects may affect fish, migratory birds, navigable waters or any other element of federal jurisdiction (Wilkinson, 2012).

Federal laws and regulations guide the design of the baseline study and impose specific elements for various permits issuance. Many elements of the study are planned as prescribed for the joint assessment by the federal and provincial authorities. Federal regulations impose a permit acquisition for various activities related to mining, including:

- Construction of a structure in a lake or river that is considered navigable;
- Engaging in an activity affecting a listed species or its habitat;
- Disturbing, destroying or taking a nest, egg, or nest shelter of a migratory bird;
- Installation of a system that operates with halocarbons;
- Explosive magazines and transporting of explosives;
- Destruction and/or harm of existing fish habitat;
- Destroy fish by any other means other than fishing;
- Discharge an effluent containing deleterious substance under the Metal Mining Effluent Regulations.

The Metal Mining Effluent Regulations (SOR/2002-222) enabled by the Fisheries Act address the establishment of a two-fold Environmental Effects Monitoring Program (Environment Canada, 2011):

- Effluent monitoring according to discharge standards and water quality monitoring studies;
- Biological follow-up studies on fish and bottom-living animal.

The regulations also require the mining facilities effluent to provide compensation for impact to be identified in the assessment process. Environmental compensation will be required, in particular for fish habitat alteration or lost. Compensation for social issues identified in the social impact study will also be required according to the Metal Mining Effluent Regulations and thus, they will be planned and realized in collaboration with the Cree nation.

20.3.4 Environmental Permitting

A preliminary list of Quebec authorizations, licenses and permits required is included in Table 20.1. Federal authorizations, licenses and permits required are listed in Table 20.2. The permit requirements will be reviewed and updated as the project advances through the environmental assessment and permitting process.

Table 20.1 –Projected Provincial Permits and Licenses for the DLIP

Permit and Licenses	Issuing Agency
Environmental assessment certificate	MDDEP - Direction des évaluations environnementales
Certificate of authorization (section 22 of the Environment Quality Act)	MDDEP - Direction régionale de l'Abitibi-Témiscamingue et Nord-du-Québec
Authorization to modify a wildlife habitat (Section 128.7 of an Act respecting the conservation and development of wildlife)	MRNF - Direction de la protection de la faune Abitibi-Témiscamingue
Permit to occupy crown land	MRNF
Forest work permit (for deforestation on crown land)	MRNF - Unité de gestion des ressources naturelles et de la Faune Abitibi-Témiscamingue
Authorization to erect or maintain a construction on the lands of the public domain	MRNF

Table 20.2 – Projected Federal Permits and Licenses for the DLIP

Permit and Licenses	Issuing Agency
CEAA Approval	Canadian Environmental Assessment Act
Metal Mining Effluent Regulations ("MMER")	Fisheries and Oceans Canada
Fish Habitat Compensation agreement	Fisheries and Oceans Canada
Section 35(2) Authorization for harmful alteration, disruption or destruction of fish habitat	Fisheries and Oceans Canada
Navigable Water: Stream Crossings Authorization	Transport Canada

20.3.5 Biophysical Environment Baseline Studies

The environmental assessment aims to characterize the biophysical elements of the project, in accordance with the Quebec-Cree Agreements, in particular the JBNQA, as well as with the provincial EQA and other applicable acts and regulations. The previous study (Boulé *et al.*, 2011, Thomassin *et al.*, 2012) has provided a scope of the biophysical environment in the mining area. The preliminary results presented in the scoping study are currently under review and will be analyzed in relation to scientific literature and reports (i.e.: Hydro-Québec, 2004; Hydro-Québec and Genivar Groupe Conseil inc., 2005). The environmental assessment process for the entire project started in July 2012, including the assessment of the human environment and the biophysical elements related to the pipeline corridor and the harbour installations. The baseline studies will give information on the actual state of the various ecosystems potentially impacted by the DLIP. This information will be useful as reference for the impact assessment for each phase of the project (development, construction, exploitation, closure and remediation).

The biophysical elements described in the three areas of the DLIP include hydrology, hydrogeology, topography, water quality, climate as well as the main vegetal and animal species composing the ecosystem, their habitat and important aspects related to their life cycle (i.e.: migration, alimentation, reproduction, protection). Each ecosystem is also described according to its ecological and social value and according to its level of vulnerability and uniqueness. The human environment is studied under a social and historical perspective, in order to understand local communities, their relation to the natural environment, the uses of the various biophysical elements and the project perceptions among communities' members (MDDEP, 2012b).

20.3.6 Study Areas Assessment Overview

The DLIP is located in a large area, within the taiga shield ecozone, at the northern limit of the boreal coniferous forest. The taiga shield ecozone is considered an ecological transition zone where characteristics of both the Boreal and the Arctic ecosystems can be observed (NRCAN, 2012), which support a wide variety of habitats and a diversity of species in the area. The landscape is constituted of bald Precambrian bedrock covered by thin acidic soils. The landscape support forest stands of variable density mixed with open zones typical of the tundra, as shrublands and sedge meadows. A short growing season and frequent forest fires are important aspects of the ecosystem dynamic (Li and Dubruc, 1999; NRCAN, 2012). A 100 years fire cycle currently prevails in this region and vast forest stands have been burned (Saucier *et al*., 2003). The territory is characterized by the abundance of lakes and wetlands scattered throughout the landscape. On average, 20% of the land in the Boreal region is covered by wetlands. Delta marshes are common around large lakes and rivers (NRCAN, 2012). In the eastern region, dams and diversions of several rivers have modified seasonal patterns of flow and have formed vast water reservoirs over large area of forest stands.

20.3.7 Meteorology and Air Quality

Local climatic conditions, in particular the seasonal regimes of precipitations, are important information for the characterization of a mining site. The meteorological monitoring program is based on automated weather stations equipped with sensors for temperature, precipitation, insulation, wind speed and direction. A weather station located close to the mining site area, at the Radisson airport (La Grande Rivière station), will be used for meteorological data collection. Long-term records from government regional weather stations and historical baseline data are included in the analysis. Atmospheric dust dispersion models will also be analyzed from literature and reports.

In northern Québec, temperatures in January can reach -40°C (mean -23°C) to 32°C (mean 14°C) in July (Environment Canada, 2012). The climate of this subarctic region is characterized by long cold winters and short summers with cool to mild temperatures, and precipitations are low to moderate (averaging 250 to 500 mm a year). Snow and freshwater ice-cover generally persist for six to eight months. Data series from 2012 have shown an average monthly temperature of -21.2°C in January and 13°C in July (Environment Canada, 2012). Precipitations measured are particularly abundant in September (100 mm) and snow accumulation reach 60 cm in November. From October to December, dominant wind generally comes from the south or southwest.

20.3.8 Hydrology

The baseline study includes the characterization of the main aspects of the hydrological system: water balance, drainage patterns, effects of diversions, discharges estimates and flow regimes. The DLIP is located within two drainage system: the Beaver river watershed oriented westward, which includes Duncan Lake and Esprit Lake sub-basins, and the Desaulniers Lake watershed which is oriented eastward (MDDEP, 2012). The Esprit Lake is close to Deposit 1 and Achan Amika Lake is located close to Deposit 2. Desaulniers Lake, with an area of 151 km^2, includes Deposits 3, 4 and 6 (Boulé *et al.*, 2011). The Desaulniers lake natural outflow has been modified following La Grande reservoir creation. The watershed discharge is now under Hydro Quebec control through the Desaulniers pump station where the discharge from the Desaulniers Lake is drained into an artificial lake and pumped into the Robert-Bourassa ("RB") reservoir (SEBJ, 1976). The RB reservoir was established in 1978 after flooding an area of 2,835 km^2 (Boulé *et al.*, 2011). Before flooding, the Desaulniers Lake sub-basin was a sub-basin of the Fort George River watershed. The annual flow of the RB reservoir effluent, which flows into La Grande River, is estimate to 3,474 $m^3 s^{-1}$ (Schetagne *et al.*, 2005).

20.3.9 Hydrogeology

The hydrogeology baseline study includes groundwater monitoring, in particular for water levels, water quality and hydraulic tests. The study areas are located in the hydrogeological regions of the Canadian Shield. In this region, the ubiquitous crystalline rocks are characterized by low storage capacity and uncertain ability to transmit water in sparse fracture networks. Fracture zones yield modest and variable quantities of potable water to depths of about 100 meters. Sediment aquifers are important, particularly in eskers and beneath clay basins and can provide drinking water from groundwater. Modest undulating relief on the Canadian Shield provides low driving force for slow groundwater movement and renewal. This affects water quality in bedrock fracture systems due to bedrock mineralization, particularly metals and uranium, the latter generating radon gas.

20.3.10Aquatic Resources and Water Quality

a) Fresh water ecosystem: lakes, rivers and wetlands

Aquatic resources included in the baseline survey are: water quality, including identification and selection of receiving lakes, streams and wetlands, physicochemical analysis, sediments quality and spatial distribution as well as taxonomic composition and abundance of benthic invertebrates.

Studies on aquatic resources in streams and rivers that could receive water from the proposed mine were carried out during summers of 2011 and 2012. The preliminary sampling design includes 8 potential receiving streams and 20 potential receiving lakes in the mining area. All water samples were analyzed for physical and chemical characteristics, including nutriments (nitrogen and phosphorus, total and dissolved metals, total cyanides and total organic carbon). A complementary water quality survey will be undertaken to further characterize spatial and temporal variation in surface waters under the baseline conditions. The Metal Mining Guiding Document for Aquatic Environmental Effects Monitoring (Environment Canada, 2012) provides guidelines for the study of sediment quality in mining projects. A wetland ecosystem survey will be undertaken in parallel with the lakes and rivers survey. Wetland-specific data will be collected as part of the 2013 surveys, including wetland type description, hydrological surveys and aquatic biological sampling.

b) Estuarine ecosystem

Near Chisasibi, on the coast of James Bay, the marine environment is shallow, with an average depth of 20 meters over half of its extent (Schetagne *et al.*, 2005). The main current pattern in the east coast of James Bay and Hudson Bay moves from the southern area to the north (Boulé *et al.*, 2011). This coastal current pattern is considered in the context of the harbour facilities assessment. Numerous marine investigations will be proposed in order to achieve an understanding of the marine and estuarine environment and the potential impacts related to the proposed harbour development. Since 1973, the hydro-electric complex has contributed to increase the flow rate and water level of La Grande Rivière (Paquet and Lévesque, 2001), which contribute to an increased sediment exportation rate toward the estuarine zone. Between 1973 until today, sediment material exported annually in the estuarine zone was estimated to 720,000 m^3 (Boulé *et al.*, 2011). A decreased of the erosion rate was associated to the construction of the LG-1 reservoir in 1993. In 2005, the annual erosion rate downstream of the La Grande-1 reservoir was associated to an annual exportation of 90,000 to 100,000 m^3 of sediments (Hydro-Québec and Genivar, 2005). This increase of sediment exportation rate will be considered as a residual impact when assessing the portrait of the environment around the harbour infrastructure.

20.3.11Fish and Fish Habitat

This section of the baseline study has for main objective to characterize fish populations and habitat use in water bodies likely to be affected by the mining project. Due to the large number of streams and waterbodies present in the three study areas, habitat encroachment is likely to be the most significant and probable biological issue on aquatic resources of this project. Pursuant to the Fisheries Act (L.R., 1985, ch. F-14), all project plans should meet the No Net Loss Guiding Principle of fish habitat.[1] Considering this Guiding principle, these fish and fish habitat studies will allow making adequate choices regarding the infrastructure and access roads locations, as well as the survey designs to limit fish habitat encroachment and therefore the future costs of fish habitat compensation measures.

A literature review in currently undertaken to find out more about fish abundance, distribution, diversity as well as fish habitat in the study area concerned by the Duncan Lake mining project. By example, fish communities in the La Grande River area have been studied since 1977 (Therrien and *al.*, 2002). Fisheries surveys have also been carried in the coastal waters of the James Bay and of the Hudson Bay (Stewart and Lockhart, 2005). Finally, preliminary studies have been conducted on the Duncan Lake mine area in 2011 (Thomassin *et al.*, 2012). Fish habitat characterization was done using standardized and recognized methods in the DLIP.

Extensive studies of fish and fish habitat throughout the three study areas will continue in 2013. As in the preliminary studies conducted in 2011, these surveys will be designed to collect the information necessary to develop fish habitat compensation plans and monitoring programs that satisfy regulatory requirements.

20.3.12Soils

A soil inventory study was started in 2011-2012 and will be completed in 2013 to satisfy the baseline study requirements. Soil and surface deposit are characterized based on permeability, slope analysis, and erosion risk. Potential borrow pits are also analyzed. Additional soils mapping and sampling will be conducted in the mine site area to achieve the level of resolution required for terrestrial ecosystem mapping of mine sites. The metal content of soil sample will be compared with the provincial and federal guidelines.

20.3.13Ecosystem Mapping

Studies on the ecosystems present in the project area are planned to be conducted during the summer of 2013. Ecosystem maps were developed to characterize the regional and local study area using two methods; predictive ecosystem mapping and terrestrial ecosystem mapping. Historical information available from field surveys conducted by Hydro-Québec projects will be analyzed.

20.3.14Vegetation

The stratification and ecological mapping of plant communities will be completed by photo interpretation of the aerial photos of the study area. The collection of data measured in a network of control points are used to guide the accuracy of the process. A field survey is planned in 2013 at the mine site area to ensure adequate coverage of the

[1] A working principle by which the department strives to balance unavoidable habitat losses with habitat replacement on a project-by-project basis, so that further reductions to Canada's fisheries resources due to habitat loss or damage may be prevented.

areas that will support the infrastructure. Field surveys will include a floristic inventory and the identification of any special status species, rare plants and sensitive ecological communities. The data will be used to refine the ecosystem map and the habitat quality index of the project area.

20.3.15Wildlife

The wildlife baseline study is based on a seasonal data collection schedule, which include description of the vertebrate fauna, a wildlife inventory and habitat suitability evaluation (modeling and mapping).

a) Mammals

About fifty species of mammals inhabit the ecozone of the taiga shield (Bernhardt, 2012). In the study area of Duncan Lake, 36 mammal species are likely to be found (Desrosiers *et al.*, 2002; CRRNTBJ, 2010). The largest mammals found in noticeable numbers are the moose, caribou, black bear and the gray wolf. A mention of a polar bear has been made at the RB reservoir, between Radisson and Sakami (CRRNTBJ, 2010), which is inside their known distribution range (MRN, 2011).

Big games such as barren-ground caribou are important for traditional hunters. Two caribou ecotypes are present in the area: the resident woodland caribou and the tundric migratory caribou. The tundric caribou population is subdivided in two large herds: the Rivière-George and the Rivière-aux-feuilles herds. The Rivière-George herd’s calve zone is located south-east of the Ungava Bay and its winter range is situated in the Taiga forest in the middle of Labrador (Taillon, 2010; Caribou Ungava, 2012). The Rivière-aux-feuilles herd migrates from its calve zone in the Ungava peninsula to its winter range in the taiga forest, reaching Chisasibi and Wemindji traditional territories (MRN, 2012a).

The resident woodland caribou population is composed of a few isolated small herds distributed south of the 52nd parallel. Assinica, Nottaway and Temiscamie are the closest known herds. Since 2000, the woodland caribou is listed as a vulnerable species in Quebec (MRN, 2011), and threatened in Canada (COSEWIC, 2012). Woodland caribou observations have been reported in the study area near “la Baie Populaire” (Yvon Labbé, Pers. Comm). The same observer also saw trails in the “Ground Road”. An investigation will be conducted to confirm their presence during the community consultation process. According to Rudolph *et al.* (2012), all classes of mining developments (active mines, developing mines and mine improvements) are considered to be pertinent anthropogenic disturbances for woodland caribou.

The actual moose density in the area is unknown since the last aerial survey was completed in 1991 (Lamontagne and Lefort, 2004). The moose population after hunting season of 1991 was evaluated to 4,680 individuals (0.26/10 km^2). The moose density is considered low compared to other regions of Québec, but it can be explained by the low carrying capacity of the land. In the "Plan de gestion de l'orignal 2012-2019", one of the objectives is to increase the moose density in this area (MRN, 2012b).

Other small and medium-sized mammals commonly present in the Duncan lake area will be considered in the wildlife baseline study, in particular: wolverine, lynx, fox, snowshoe hare, bats and rodents. The wolverine is on the endangered mammals list in Canada (COSEWIC, 2012) and on the threatened species list in Quebec (MRN, 2011). The river otter, the American marten, the fisher, the mink, the least weasel and the short-tailed weasel are also found in the area (CRRNBJ, 2010). The least weasels are, according to Québec government, susceptible of being designated vulnerable species or at risk. Many bat species have been identified near the Eastmain-1 area: little brown bat, big brown bat and hoary bat (McDuff and Brunet, 2005; Prescott and Richard, 2004). The little brown bat does not have any concerned status in Quebec but is endangered in Canada. The hoary bat and red bat are both considered susceptible in Québec and not at risk in Canada (MRN, 2011; COSEWIC, 2012).

b) Herpetofauna

The wetlands constitute quality habitats for herpetofauna. However, herpetofauna is poorly documented in James Bay region (Desroches *et al*, 2010). Six species were documented in 2002 in the Chisasibi-Radisson region: American toad, spring peeper, wood frog, mink frog, blue-spotted salamander and common gartersnake. In May 2012, all of these species but the blue-spotted salamanders were observed (Thomassin *et al*., 2012). According to previous observations, the leopard frog was observed at 120km north of Opinaca River (MacCulloch and Bider, 1975; AARQ, 2012). It might also be possible to find the two-lined salamander in the area (CRRNTBJ, 2010). Further inventories will help to verify if the species is present in the other areas of the project (areas #2 and #3).

c) Avifauna

The abundant lakes, wetlands and the proximity of the James Bay coast attracts many species of birds which rest and feed in the area before reaching their arctic breeding grounds, i.e.: geese, ducks and loons. Several other bird species use habitats located in the taiga shield ecozone, i.e.: arctic tern, common tern and white-throated sparrow (NRCAN, 2012). Attention will be consented to bird habitats that are also susceptible to be found in the study area.

The harbour proposal area is located in the Bird Conservation Region ("BCR") number 7: Taiga Shield and Hudson Plains (BSC, 2012). As seen before, the abundance of water and wetlands in this region provides important habitat for many species. The avifauna has been described for the Eastmain-1-A and Rupert Diversion area, from the RB reservoir to about 160 km south of Duncan Lake

(Benoit and Ibarzabal, 2004; Benoit and Létourneau, 2004; Mousseau, 2004; Tecsult Environment Inc., 2004). This documentation can therefore be useful to have an idea of the avifauna present in the project region.

The study area is important for bird conservation, especially the portion where the harbour activities would take place. The Northeast James Bay Coast is considered as an important bird area (IBA, 2012). This area of 3,331.24 km^2 covers a stretch of the James Bay coastline from the corner at Point Louis-XIV southward to the Rivière du Vieux-Comptoir (South of Wemindji) (IBA, 2012). Multiple types of habitats are found in the area: boreal and alpine forest, native grassland, salt and brackish marches, tidal-rivers-estuaries, saline, open see, coastal cliffs-rocky shores (marine), scree and boulders. The presence of congregatory species, different groups of birds (waterfowl, colonial birds, waterbirds, seabird, and shorebird) and threatened species were used to access this IBA. According to the CRRNTBJ (2010), the main congregatory species using the Northeast James Bay Coast in fall migration are the Canada goose, the brant, the white-rumped sandpiper and the semipalmated sandpiper. During their migration, geese have sustained traditional hunters of the Cree and Inuit nations.

20.4 Environmental survey and apprehended impacts

20.4.1 First Analysis from Previous Studies

During the site visit in August 2012, no contaminated soil by oil or gasoline was observed by the visiting CIMC team. No field camps were installed for the 2010-2011 and 2011-2012 environmental surveys conducted by Roche (Boulé *et al.*, 2011, Thomassin *et al.*, 2012). CIMC has ensured that all exploration programs on the property have and will be conducted in an environment friendly manner. CIMC is fully committed to a policy of corporate responsibility and sustainability in all aspects of its operations (CIMC, 2012). There are no environmental liabilities mentioned in GESTIM management system, and no known environmental concerns mentioned on the DLIP (GESTIM – MRN, 2012). Met-Chem is not aware of any environmental liabilities on the site (CIMC *et al.*, 2012).

20.4.2 Design of the Environmental Survey and Apprehended Impacts of the Project

The design of the environmental survey is planned in a way to achieve the collection of all technical data required to evaluate, in a summary manner, the main impacts apprehended on the environment for each phase: development, construction, exploitation, closure and remediation. A literature review on the apprehended impacts is under progress. The project notification will suggest preliminary mitigation and remediation measures according to the baseline study results. The preliminarily design of the ongoing monitoring program, to be planned for the development, construction, operation, closure and remediation periods, could include the monitoring of meteorology, air quality, hydrology, water quality, fish and fish habitat, and wildlife.

20.5 Environmental Management

20.5.1 Environmental Management System ("EMS")

Development and operation of the mine, corridor transportation, harbour facilities and associated access roads will affect a range of marine, aquatic and terrestrial habitat types and wildlife species. Mining operations might have an impact on the quality of air at the mine site and surrounding locations, depending on the mitigation measures put in place. The scope (i.e. level of detail) and nature of the EMS (i.e. standardized and non-standardized) will be related to the nature, scale and complexity of the DLIP. CIMC is to implement and adhere to an EMS that incorporates the following features:

- Definition of an environmental policy for the installation by top management. This will enable a full success of the different features in the EMS;
- Planning and establishing the necessary procedures;
- Implementation of the procedures with focus on:
  - Structure and responsibility;
  - Training, awareness and competence;
  - Communication;
  - Employee involvement;
  - Documentation;
  - Efficient process control;
  - Maintenance program;
  - Emergency preparedness and response;
  - Safeguarding compliance with environmental legislation.
- Checking performance and taking corrective action, with focus on:
  - Monitoring and measurement;
  - Corrective and preventive action;
  - Maintenance of records;
  - Independent internal auditing in order to determine whether or not the environmental management system conforms to planned arrangements and has been properly implemented and maintained.
- Review by top management.

Other features will complement the above steps as supporting measures. These include:

- Develop a management system and audit procedure examined and validated by an accredited certification body;
- Prepare and publish a regular environmental statement describing all the significant environmental aspects of the installation, allowing for year-by-year comparison against environmental objectives and targets, as well as with sector benchmarks as

appropriate; implement and adhere to an internationally accepted voluntary system such as EMAS and EN ISO 14001:1996 (EIPCCB). This voluntary step will give higher credibility to the EMS of the DLIP. In particular, EMAS which embodies all the above mentioned features.

20.6 Tailings and waste rock management

20.6.1 Tailings Management Facilities ("TMF") Design Basis

Based on current estimates provided by Met-Chem, the annual iron pellets production tonnage for the DLIP has been fixed at 12 Mtpy. The two tailing ponds areas will cover an estimate of 2,220 ha. The tailing ponds areas are designed to be expanded over the lifelong mine period (20 years) in sections. In order to ensure long-term control of the quality of surface runoff and seepage waters from the tailings, a water treatment plant will be constructed. This water treatment plant will have a minimal capacity of 7,500 m^3 per hour in order to manage average yearly precipitation. The tailings pond freeboard will be designed to contain extreme precipitation.

20.6.2 TMF Design

In consultation between Le groupe Desfor, Met-Chem and CIMC, the two locations for the TMF were determined to be: 1) between Deposit 3 and Deposit 4 (called the north zone) and 2) east of Deposit 6 (called the south zone), as shown on the site plan. To decommission tailings impoundments, a multi-layer cover system is proposed. This multi-layer cover system will be based on an engineered system that will integrate the tailing pounds into the natural evolution of the site after mine closure. The multi-layer cover system will act to retain moisture and hence provide a low diffusion barrier to atmospheric oxygen. The multi-layer cover system will consider several factors including climate conditions and climate change in the area, waste material potential reactivity (reactivity and buffering capacity), the type of waste rock (ARG or NAG), hydrogeological setting and basal inflow conditions. "*Multi-layer cover systems utilize the capillary barrier concept to keep one (or more) of its layers near saturation under all climatic conditions. This creates a blanket of water over the reactive waste material, which reduces the influx of atmospheric oxygen and subsequent production of acid drainage*", (Tremblay et Hogan). An economic and technical feasibility study of the multi-layer cover system will be done. The study will include site climate conditions, availability of cover material, cover and waste material properties and conditions, surface topography, soil and waste material evolution and finally vegetation conditions.

20.6.3 Waste Rock Management

Based on results from Section 13 (Table 13.12), most of the waste rock from Duncan Lake open pits (Deposits 3 and 4) were determined to be non-acid generating waste rock. Run-off from the waste rock disposal areas will be ditched and directed to water quality control sampling points for testing prior to release to the environment, thus mitigating any environmental impact on the surrounding Desaulniers Lake watershed area.

20.6.4 Waste Rock Disposal

Since preliminary results indicate that most of the waste rock will be non-acid generating, the waste rock piles will be capped at closure with single layer cover of overburden with vegetation.

20.6.5 Hydrogeology

Within the scoping study (Boulé *et al.*, 2011-12), no hydrogeology baseline study was undertaken. Groundwater samples have been collected at the regional level and at the mining site area. Thirty four groundwater sample sites have been identified and collected in 2011, and 12 groundwater sample sites were added in 2012 for a total of 46 groundwater sampling site in the DLIP. Based on the mining site location and considering a complete water balance evaluation, many tasks are now being conducted through the hydrogeology baseline study. Following geotechnical studies of the mining site, groundwater monitoring is planned with new wells and piezometer installation. Key hydrogeological data have been collected through constant head permeability test in the mining site area. These permeability tests have been conducted in fourteen pre-selected drill holes to acquire data on bedrock permeability and hydraulic conductivity in Deposits 1-3 and 4. Other key parameters such as groundwater flow, on site multi-parameter analysis and groundwater sampling have been collected. All these data will enable hydrogeology modeling and enhance hydrogeology understanding in the mining site area.

20.6.6 Hydrology

As mentioned in the regional study (Boulé et al., 2011-12) no hydrometric stations were established within the mining site area. Manual flow measurements were collected in three gauging stations in 2011 and 2012, providing very limited hydrology information of the area. Two limnimeter stations were also installed for the regional study in Esprit Lake and Desaulniers Lake, daily water depth measurement are collected from these stations since September 2011. Based on the mining site location and considering a complete water balance evaluation of the mining site area, many tasks are now being conducted through the hydrology baseline study. These include watershed and sub-basins limits, hydrographic network and drainage pattern. This will enable surface pollution dissemination control and ensure adequate water management in the mining areas and basins. The INRS-ETE specialist in hydrology and hydrodynamic modeling assists Desfor specialist in the study. Key hydrological parameters are to be included in the baseline survey, including actual daily runoff at different recurrence (1-5-100 and 1,000 years) of the natural system, and potential runoff on the installation site for the process water (water treatment plant). The manual flow monitoring stations conducted in the regional study in 2012 are planned to be re-mobilized in 2013 and automated real time flow monitoring stations will be added following water quality survey. Also additional manual flow measurements are to be undertaken to improve the existing rating equations (i.e. stage/discharge curves) for each sub-basins in the Desaulniers Lake watershed and in the eastern part of the Duncan lake watershed. All these data will provide key information to establish the water balance between natural system and process water for the project. The overall balance superimposes these two systems to account all water on the mining site.

20.6.7 Water Management

The water management objective for the mine is to minimize the potential for any short or long term adverse impact to the quantity and quality of surface water and groundwater, in the existing watersheds and aquifers in the area immediately surrounding the mine site by; i) defecting clean non-contact water away from the site to prevent such water from picking up contaminants; ii) by intercepting, containing and recycling water that does come in contact with mine facilities such as the open pits (4), site roads, waste rock dumps to the greatest extent possible; and iii) by treating the mill tailings slurry before it leaves the mill and by operating the tailings storage facility as a zero-discharge facility during the mine operating life.

Diversion ditches will be constructed to avoid the contact of clean runoff water with areas affected by the mine or mining activities. Contact water originating from affected areas is intercepted, collected, conveyed to central storage facilities for re-use in process, or decant to treatment (if needed) prior to release to receiving streams. The mine consists of two iron-bearing deposits that will be mined via two open pits.

20.7 Remediation and mine closure requirements

The DLIP will be developed, operated and closed with the objective of leaving the DLIP Property in a condition that will mitigate all potential environmental impacts and restore the land to its pre-mining land use and capability. Wherever possible, closure and remediation activities will be carried out while the mine is in operation at Duncan Lake, and final closure and remediation measures will be implemented at the time of mine closure.

20.7.1 Remediation Pre-Planning

A regular follow-up of the main effluent will be pursued until the DLIP phase out to post-closure phase. Following final closing of the mine activities and before remediation works, the DLIP will apply a mining and groundwater wastewater follow-up program. This follow-up program will enable the DLIP to ensure conformity as well as qualitative and quantitative evolution of the dejection discharges in the environment, within the period of transition preceding remediation of the mining site; and adapt the appropriate remediation ways to be implemented on site. Beforehand, the DLIP will have proceeded with the determination and characterization of all potential sources of contamination on site.

20.7.2 Objectives and Scope of the Remediation, and Mine Closure Plan

Additional variation of water quality emanating from the various sources is likely to occur during the closure and post-closure periods. These changes may take place over a period of as little as two years to as many as thousands of years, depending of the nature of the wastes and the proximity of water resources. Remediation and mine closure will aim to put back the site in a satisfactory state, which means eliminating all risks for health and public safety, to avoid all production and distribution of substances with potential impact on the environment, to maintain an appropriate visual landscape and finally, to transfer the infrastructure in a state that can be useful for other users or purposes.

20.7.3 Proposed Approach to Remediation and Mine Closure

The contents of the remediation plan, updated according to the requirements of the most recent version of the Guide and conditions of preparation of the plan and General requirements for restoration of mining sites in Quebec, will be used as the main reference for the authorization certificate request for the remediation works and mine closure in the DLIP.

20.7.4 Post Closure Monitoring and Treatment

During the closure, reclamation, and post-closure phases of mining at the DLIP, the following characterization methods should be employed:

- Comparison of predicted and actual water quality in lakes and rivers in the areas;
- Continued sampling of quality and quantity of water resources, including springs, leachate, surface water, and groundwater at points of compliance and other locations;
- Measurement of rate of change in groundwater levels over time after groundwater pumping has ceased;
- Monitoring of effectiveness of mitigation measures and comparison to predicted performance.

20.7.5 Estimated Closure Costs

As stipulated in the current Mining Law, a rehabilitation plan will have to be prepared. The rehabilitation and restoration plan will have to be developed in accordance with the provincial Guidelines for preparing a mining site rehabilitation plan and general mining site rehabilitation requirements (MRNF and MDDEP, 1997).

The economic analysis of a mining project will have to take into account the costs required for mine closure.

Preliminary closure plan costs have been estimated based on the rehabilitation of the tailings disposal area and the waste rock disposal area.

The preliminary cost estimate of the rehabilitation and closure plan is based on the re-sloping and re-vegetation of the tailings storage facility and the re-vegetation of the top and berms of the waste rock dumps, which usually represents the largest proportion of rehabilitation costs.

Once Deposit 3 and Deposit 4 pits will deplete, it is expected that these will fill with water through underground seepage until it eventually reaches the water table level. Whenever possible, diverted streams will recover their original flow paths.

At this time, revegetation of the berms and top of the waste rock dumps was considered. A multi-layer cover system was considered for the tailings storage facility rehabilitation.

Based on the accumulation areas identified in Table 20.3, the total cost for the rehabilitation of the tailings storage facility and waste rock dumps has been estimated at $156.4 M.

Project advances through the environmental assessment and permitting process.

Table 20.3 –Accumulation Areas for Waste Rock Dump and Tailings Storage Facility

Accumulation Areas	Unit	Area
Tailings Storage Facility North+Ext (Years 1 to 12)	m^2	6,832,000
Tailings Storage Facility South 1 (Years 13 to 15)	m^2	3,792,000
Tailings Storage Facility South 2 (Years 15 to 19.6)	m^2	4,125,000
Deposit 3 Waste Rock Dump Area	m^2	5,947,000
Deposit 4 Waste Rock Dump Area	m^2	978,500

20.8 Human Environment Baseline Studies

The baseline study planned for the human environment is designed to characterize the social and economic elements before the realization of the project. All technical data required to evaluate, in a summary manner, the main impacts apprehended on the human environment for each phase of the project (development, construction, exploitation, closure and remediation) will be included in the baseline study. The characterization of the various issues related to the human environment is under process. The presence of members from the Cree community of Chisasibi in the areas affected by the project is an important aspect that is included in the baseline study. A description of the land use and land regime will be included. The key or valued social components will be studied in regards to the socioeconomic issues. The section on the Consultation program will be further refined as a result of the literature review on apprehended impacts and the integration of the acquired information from the different actors related to the project issues.

20.8.1 Chisasibi Cree Nation

Chisasibi is a young and dynamic community located on the eastern shore of James Bay, which has continued to grow since its relocation in 1980-81 from the nearby island (Fort

George), located in the outlet of La Grande River (CNC, 2012). This island was used during centuries by the nomadic Cree nation for summertime assemblies and it was later known as Fort George at the fur trade period. As a result of negotiations between the Crees and the Québec Government over the James Bay Power Project, the Fort George Relocation Corporation was formed to manage the relocation and the construction of the present town site in Chisasibi. Many houses and community buildings have been built since Chisasibi foundation. Community services were established and economic enterprises set up in this rapidly growing community (CNC, 2012).

As described earlier, the mining project near Duncan Lake is located 80 km southeast of Chisasibi, on traditional trap lines of the community. The main potential site for harbour infrastructure is close to the community, about 10km northwest on the other side of the river, on Category II lands.

Chisasibi is the larger Cree community on the Eeyou Istchee territory, with a population reaching 3,987 persons in 2009 (MSSS, 2009). The population is composed of approximately 3,800 members of the Cree nation, about 150 Inuit and 300 non-native people (Cree Nation of Chisasibi (CNC), 2012). Chisasibi is the most populous of the nine Cree communities and has shown the highest population growth rate between 1971 and 2006 (+ 209%). As in the case of the other Cree communities, the main characteristic of the Chisasibi population is its large group of young people, with nearly half (49.5%) of its population aged under 24, while the 25-44 years group account for 30.1% of the total community members. The labour force in Chisasibi is expected to increase at a rate of about 2.4% per year, over the 2008-2013 period (Boulé et al., 2011).

20.8.2 Radisson Locality

Radisson is the closest non-Cree community to the DLIP area. It was originally founded in the 1970s, in the beginnings of La Grande hydro-electric project, and reached a population of about 2,500 residents in the main phase of the construction (Boulé et al., 2011). Radisson has the status of locality since 1994 and has the basic infrastructure and equipment to accommodate a population of about 2,500 residents. At the end of the construction phase, in 1995, the town experienced a significant decline in population reaching as low as 352 permanent residents in 2005. The low level of population is considered the minimal density for a viable community.

In addition to the Radisson permanent residents, additional temporary residents work as part of Hydro-Québec's operating maintenance staff. Temporary residents are estimated to a total of 300 persons, but only about 150 of them are on site at the same time. Radisson is connected to southern Quebec by La Grande airport. The residents of the localities in the Municipalité de la Baie-James can be identified as Jamésiens (Gouvernement du Québec and Cris d'Eeyou Istchee, 2012).

20.8.3 Consultation Program

The consultation program included in the baseline study will be conducted in consultation with community members of the Cree nation of Chisasibi. At least two rounds of community consultation meetings, which will include open house and formal presentation, will take place in 2013. Information materials will be developed and distributed throughout each of the consultation communities in the form of corporate newsletters, newspaper notices, radio public service announcements, posters and emails. A communication agreement between CIMC and the Chisasibi Cree nation will be created and made effective. This document outlines the process and components through which the two parties have agreed to operate. Additional consultation will possibly be conducted during the environmental permitting process.

Table 20.4 –Expected agreements between Chisasibi Cree nation and Proponents

Agreement	Description/ Timing
Framework Agreement	First agreement to be negotiated, and is required to provide a funding and work plan for further negotiations between the Chisasibi Cree nation and the proponent.
Communications Agreement	To define a communication protocol, methods, and financial obligations.
Site Benefit Agreement	Will be based on the negotiated needs of the Chisasibi Cree nation and the proponent, and will consider environmental conditions/reclamation, archaeology and traditional use, economics, training and financial obligations.
Bulk sample Agreement	Prior to any bulk-sampling activities.
Environmental Assessment Agreement	To be completed prior to entering the provincial Environmental Assessment process. Should consider the involvement of the Cree Nation Government ("CNG", formerly referenced as the Cree Regional Authority) in the process.
Participation Agreement	Prior to any construction or operations of the mine.

Among the aspects studied, Health Canada's determinants of health will be considered. Data have been collected through literature review and field interviews will be conducted on the following issues: Population and demographics, Governance, Economy and employment, Education, Health, Social issues, Culture, Community Services and Infrastructure. Radisson residents will also be consulted. Different fundamental components have been proposed for the consultation program: Consultation with community government and leadership; Information, distribution and access to information; Community meetings and engagement; Consultation and communication tracking; Issues identification, Issues resolution and Integration of consultation outcomes with socio-economic and cultural baseline, and effects assessment.

21.0 CAPITAL COSTS AND OPERATING COSTS

21.1 Capital Costs

21.1.1 Scope of the Estimate

The capital cost estimate includes the material, equipment, labour and freight required for the mine pre-development, mine equipment, processing facilities, tailings storage and management, concentrate transport, pellet plant facilities, pellet storage and handling facilities, ship loading and port facilities, as well as infrastructure and services necessary to support the operation.

21.1.2 Summary of the Estimate

All amounts are expressed in CAD dollars unless otherwise noted. The total life-of-mine ("LOM") capital cost for the 12 Mtpy pellet production rate scope of work is estimated at $4,546 M of which $3,881 M is initial capital and $665 M is sustaining capital as summarized in Table 21.1.

Initial capital of $3,881 M includes $3,833 M for pre-production period and $48 M for mining support and service equipment as well as mining services to be procured in the first year ramp-up period.

Table 21.1 – Summary of LOM Cost Estimate

Item Description	Total Rounded ($ Millions)
Initial Capital	
Pre-Production Capital	3,833
Ramp-Up Capital	48
Total Initial Capital	**3,881**
LOM Sustaining Capital	665
LOM Total	**4,546**

The pre-production capital of $3,833 M cost includes $2,967 M for direct costs and $866 M for indirect costs including contingency. The direct capital costs and indirect capital costs are summarized in Table 21.2.

The sustaining capital costs are detailed in section 21.1.3.

Table 21.2 – Summary of Pre-Production Cost Estimate

Item Description	**Total Rounded ($ Millions)**
Direct Cost	
Open Pit Mine	
Mine Development	53
Mine Services and Facilities	17
Mining Equipment	1
Total Open Pit Mine	**71**
Process	
Crusher Area	44
Crushed product Stockpile and Reclaim	50
Concentrator Area	524
Total Process	**618**
Concentrate and Reclaim Water Pipelines	
Concentrate Pipelines and Systems	185
Reclaim Water Pipeline	116
Road Extension	10
Total Concentrate and Reclaim Water Pipelines	**311**
Pellet Plant	
Thickening, Filtration, Pelletization	1,080
Infrastructure	27
Total Pellet Plant	**1,107**
Port	
Pellet Storage and Reclaim	304
Port Ship Loading and Wharf	250
Services	1
Camp (installation, leasing)	4
Total Port	**559**

Item Description	Total Rounded ($ Millions)
Tailings and Water Management Facilities	
Tailings Storage Facility	23
Tailings Pipelines and Spigot	5
Water Treatment Stations	4
Reclaim Water Pumping and Pipeline	8
Total Tailings and Water Management Facilities	**40**
Infrastructure at Mine Site	
Industrial Site Preparation and Drainage, Site Roads	18
Main Road (included)	00
Ancillary Buildings	6
Camp (installation, leasing)	7
Office Complex	2
Mine Vehicle Maintenance Building	23
Airstrip (not required)	00
General Services Mine Site	11
Total Infrastructure aat Mine Site	**67**
Power and Communication	
Power and Communication at Mine Site	112
Power and Communication at Pellet Plant	65
Power at Port	3
Total Power and Communication	**180**
Service Vehicles and Equipment	
Light Vehicles	2
Earthwork Vehicle	4
Material Handling Vehicles	4
Service Vehicles	2
Service Equipment	1
Emergency Vehicles / Fire, Ambulance, Rescue Truck	1
Total Service Vehicles and Equipment	**14**
Total Direct Cost	**2,966,449,000**

Item Description	Total Rounded ($ Millions)
Indirect Cost & Contingency	
Indirect Costs	363
Closure & Rehabilitation (Sustaining Capital only)	00
Contingency	503
Total Indirect Cost & Contingency	**866**
Total Pre-Production Cost	**3,833**

21.1.3 Sustaining Capital Costs

The LOM sustaining capital is estimated at $665 M. Provisions are made for mining equipment, service equipment and process equipment replacement, tailings storage relocation and progressive expansion, some additional fuel storage at the pellet plant, expansion of the mine vehicle shop and to infrastructure as well as closure and rehabilitation costs. The sustaining capital costs are summarized in Table 21.3.

Table 21.3 – Summary of Sustaining Capital Cost Estimate

Item Description	Total Rounded ($)
Open Pit Mine	48
Process	32
Concentrate and Reclaim Water Pipelines	10
Pellet Plant	6
Port	00
Tailings and Water Management Facilities	372
Infrastructure Mine Site	12
Power and Communication	00
Service Vehicles and Equipment	29
Closure and rehabilitation costs	156
Total Sustaining Capital Costs	**665**

21.1.4 Basis of Estimate – General

a) Base date, currency, escalation

The base date for the cost estimate is the first quarter of 2013.

The capital cost estimate is expressed in CAD dollars. The exchange rate used is $1.00 USD/$1.00 CAD when quotations were received in US dollars.

No allowances for escalation or currency fluctuation are included.

b) Labour, installation

Most of the installation costs are included in the unit rates or were estimated by factor. However, some installation costs are estimated by man hours, productivity loss factor and labour rate.

The man hours were established from in house database or from construction estimating standards.

The labour productivity loss for the Project was established at 1.15 considering impact of major criteria only such as working calendar, availability of skilled labour and supervision, as well as northern site conditions.

The labour rate was established as an all-inclusive, mixed crew, average hourly cost to the owner of $130, based on Commission de la Construction du Québec ("CCQ") schedule of labour cost and the hourly rates published by the Association de la Construction du Québec ("ACQ"). The working calendar is assumed 7 days per week, 10 hours per day, and 4 weeks in, 1 week out turnaround.

21.1.5 Basis of Estimate – Mining

a) Mine development cost

The mine development costs are comprised of all mine operating expenditures incurred during the pre-production phase of the project. These costs were estimated by rate ($/t, $/y, $/ha) and also by lump sum capital expenditures based on similar projects. The following items comprise the different components of the mine development cost:

- Explosives;
- Dewatering;
- Clearing;
- Topsoil Removal;
- Technical Services Equipment;
- Engineering Consulting Fees;
- Equipment Simulator;
- Manpower;
- Contract mining in pre-production;
- Operation of service equipment (Pickup Trucks).

b) Road Construction

Capital expenditure based on length of haul road designed to access deposits being mined in first year of production, estimated based on contractor rate for road contraction and preparation from similar projects.

c) Mine services and facilities

Mine change house facility cost is included in the mine vehicles building.

The explosive preparation will be sub contracted; facilities will be provided by the contractor. Provision is included for site preparation, concrete foundations and fencing. Estimation was based on similar projects.

Provisions were also made for electrical power supply to the mining area and explosive preparation facilities. Preliminary requirements were established for the electrical material, equipment and accessories. Estimation was done with unit rates, installation man hours and labour hourly rate based on in house database.

d) Mining equipment

Major mining equipment and major support equipment will be leased throughout the course of the mine life. Leasing of equipment was interpolated based on monthly rates from similar projects. See Table 21.4 for Equipment being leased.

Table 21.4 – Equipment Leasing

Description	Model
Major Equipment	
Truck	CAT 797
Hydraulic Shovel	CAT 6090FS
Loader	L2350
Drill	CAT MD6640
Support Equipment	
Track Dozer	CAT D11T
Wheel Dozer	CAT 854K
Road Grader	CAT 24M

Most of the Support Equipment and all Service equipment will be incurred as capital expenditures and will be replaced in years 10 to 15 of the mine life. Purchase price of equipment was based on similar projects.

21.1.6 Basis of Estimate – Processing Areas

a) Process Buildings

Process buildings were estimated by factors based on recent similar projects. Site preparation and ancillary buildings are included in the infrastructure section below.

b) Process Equipment

The process equipment list was derived from the flow sheets. Single source quotations from qualified suppliers were obtained for major equipment. The remaining equipment was estimated from recent in-house databases of similar projects.

Equipment installation was estimated by factor based on recent similar projects. An allowance was also provided for special lifts, sub-contracts and construction material. Freight was established at 10% of the material and equipment value.

c) Process Piping

Process piping cost was estimated by factor.

d) Electricity and Instrumentation

Electricity, automation and instrumentation for the process were estimated by factors based on recent similar projects.

e) Services and Supplies

Services and supplies for the process were estimated by factors based on recent similar projects. Services include mainly HVAC and dust collecting ducting, local fire protection as well as plant air and water services distribution. Supplies include mainly living quarter's furniture, equipment and supplies, small shops tools and storage equipment, safety and security systems as well as special coatings if required.

21.1.7 Basis of Estimate – Concentrate Pipeline and Reclaim Water Pipeline

a) Pipelines

The concentrate pipeline and reclaim water pipeline were estimated based on estimation from previous phase. The pipelines will be buried alongside the road heading to the pellet plant. The estimation was updated and benchmarked with recent similar projects and also adjusted for scope of supply and revised lengths.

b) Road extension

Provision was included for an extension to an existing road to reach the new pellet plant location. The length was derived from the site layouts; the cost was estimated with a unit rate from recent similar projects, adjusted for this project.

21.1.8 Basis of Estimate – Pellet Plant

a) Pellet plant

The pelletization facilities including thickening, filtration and induration, were estimated based on a cost per annual tonne of pellet production, developed from budget proposals from recent similar projects.

b) Infrastructure

Provisions were estimated for infrastructure, including: site preparation, site roads, administration building, permanent camp, warehouse, emergency vehicle building, metallurgical laboratory, and general services more precisely, heavy fuel oil and diesel storage and distribution facilities. These were estimated from recent similar projects and adjusted for this project. The capacity for fuel storage was established for 8.5 months of production. The diesel storage at the pellet plant considers the general diesel requirement and the mining equipment requirement. The diesel will be trucked to the mine site storage facilities.

The site preparation quantities were established from available topographic maps and preliminary layouts. The quantities were optimized to balance the cut and fill quantities.

21.1.9 Basis of Estimate – Port and Pellet Storage

a) Pellet storage and reclaim

Preliminary requirements were established for infrastructure, including site preparation and roads. The costs were estimated based on recent similar projects. The site preparation quantities were established from available topographic maps and preliminary layouts. The quantities were optimized to balance the cut and fill quantities.

The list of mechanical equipment was based on recent similar projects, benchmarked and adjusted for the project. The costs for the equipment, the freight, the installation, the piping, the electrical and instrumentation and the services and supplies were estimated as for the process area.

b) Port ship loading and wharf

The infrastructure requirements were established as per correlation and experience and verified using the United Nations monograph No.6 "Measuring and Evaluating Port Performance and Productivity".

The mathematical model is an approximate method modified to correlate to real terminals and is generally acceptable for planning purposes. The method is used to approximate waiting time under different conditions and calculate costs in order to balance port requirements versus demurrage.

Based on numbers applicable to deep water vessels, interpretation of this method, combined with previous experience, in-house statistics from bulk terminals and the results of the calculations for a four month loading season resulted in the following arrangement: two berths and two 12,000 t/h ship loaders.

Quantities were derived from conceptual drawings and cost estimation was done with unit rates and allowances based on recent similar projects.

c) Services at port

A general allowance was included for services at port.

21.1.10Basis of Estimate – Tailings

a) Tailings storage

Preliminary requirements were established for the tailings storage facilities and storage dams' quantities were estimated. A geomembrane liner was considered required along the dams only to contain potential acid generation.

Cost estimation was done with unit rates based on recent similar projects.

b) Tailings pipeline

Tailings pipeline and water reclaim pipeline were sized with preliminary data and quantities were derived from the site plans. The cost was estimated with unit rates from construction estimating standards.

c) Water treatment

Preliminary requirements were established for the water treatment and the cost estimation was based on quotations received for recent similar projects.

21.1.11Basis of Estimate – Concentrator Infrastructure and Services

a) Industrial site preparation, main road and site roads

Preliminary requirements were established for site preparation. The costs were estimated based on recent similar projects.

James Bay road will have to be deviated before year 6 when open pit becomes too close to the existing road. The length was established form the layout and cost was estimated based on similar road construction. The design criteria will match the existing James Bay road design.

Lengths for site roads were derived from layouts and the costs were estimated based on recent similar projects.

b) Ancillary buildings and facilities, laboratory equipment

Provisions were made for a gate house and parking area, a warehouse and cold warehouse storage facilities and a building for the emergency vehicles. The costs were estimated based on preliminary design and layouts and recent similar projects.

An allowance was also made for laboratory equipment.

c) Camp at mine site

The permanent camp at mine site is considered as a long term leasing. Provision was added in initial capital for the freight, installation and rental fees during construction of the camp facilities, based on quotes received for recent similar projects.

d) Office complex

Preliminary requirements were established for the office complex. The cost was estimated based on recent similar projects.

e) Mine vehicles maintenance building

Preliminary requirements were established including overhead crane, service equipment and supplies, washing facilities as well as tools and storage equipment. The cost was estimated based on recent similar projects and coordinated with the maintenance requirements of the mining equipment of this project.

The mine change house and offices are included in the mine vehicles maintenance building.

f) General services

Preliminary requirements were established for the mine site general services, including fuel storage and distribution, fresh water supply, sanitary and waste management and fire protection general systems. The costs were established based on preliminary requirements and recent similar projects.

The fuel storage requirements at mine site was established for 7 days production.

21.1.12Basis of Estimate – Power and Communication

Preliminary requirements were established for power and communication at the mine site, at the pellet plant and at the port.

For the mine site and the pellet plant, the power and communication area includes a main power line from the nearest Hydro–Québec substation, a main substation, the pole line site distribution, emergency generator sets and communication equipment and facilities while power lines only are required at the port.

Preliminary quantities were derived from the single line diagrams for the electrical material, equipment and accessories and estimations were done with unit rates, installation man hours and labour hourly rate based on in house database. Estimations for communication were based on recent similar projects.

21.1.13Service Vehicles and Equipment

Preliminary requirements were established for service vehicles and equipment and the costs were established based on budgetary quotes from recent similar projects and in-house database.

The list of Service vehicles and equipment is found in Table 21.5 below:

Table 21.5 – Services Vehicles and Equipment

Service Vehicles and Equipment	Required
Light Vehicles	
Pick-up truck	30
Bus	4
Earthwork Vehicle	
Loader with attachments	4
Grader	2
Dump truck 12 wheel	3
Material Handling Vehicles	
Mobile Crane – 100 t – for Gyratory	1
Mobile Crane – 45 t.	2
Boom Truck, 27 t.	3
Low Boy	1
Bobcat	3
Fork lifts	4
Scissor lift	2
Articulated ManLift 60 feet	2
Service Vehicles	
Diesel Tanker	6
Water Truck for Camp	2
Water Truck for road maintenance	1
Sanitary Pump Truck-Vacuum Truck	1
Service Equipment	
Fusion Machine (2) – up to 36" dia.	2
Fusion Machine 8 – 24" (2)	2
Fusion Machine 3" – 14" (2)	2
Lighting Tower	20
Emergency Vehicles / Fire, Ambulance, Rescue Truck	
Rescue Truck	2
Fire Truck	1
Ambulance	1

21.1.14 Basis of Estimate – Indirect Costs

The provision for indirect costs and contingency was established by factor applied to direct cost.

An amount of $363,187,000 is provided for indirect costs, and typically covers for the major items listed here and detailed below: Project Development, Project Implementation and Financial Costs.

- Project development owner's costs usually include: permitting process, land acquisition, administration, NSR buyout, exploration and drilling program, engineering studies (feasibility studies as well as any independent review), environmental impact assessment, metallurgical testing, geotechnical and occupational hazard studies, social impact studies and community relations, pre-production operation group and legal fees.
- Project implementation costs include but are not limited to EPCM and owner's costs including spares, first fills and commissioning and other owner's costs.
  - EPCM includes Detailed Engineering, Procurement and Construction Management as well as commissioning assistance and site assistance.
  - Spares, first fills and commissioning include capital and commissioning spare parts, capital first fills, and dry and wet commissioning that includes vendors representative on site.
  - Other owner's costs include construction indirect, owner's project team, room & board and transportation of workers to the project site.
- Financial costs included in the estimate provides for insurances. Taxes and duties are excluded from the capital cost estimate as well as from the economic analysis. Escalation and interests incurred during construction are excluded from the capital cost. Working capital is excluded from the capital costs but provision for 8 months of operation cost is considered in the Economic analysis.

An amount of $502,775,000 is provided for contingency, in consideration of the level of development of the project.

21.1.15Closure Costs

Provisions are made for closure and rehabilitation costs in the sustaining capital. Based on details given in section 20, the costs for the closure and rehabilitation of the DLIP are estimated at $156.4 M.

21.2 Operating Costs Summary

The average life of mine ("LOM") operating costs were estimated as shown on Table 21.6.

Table 21.6 – Total Operating Costs (Average life-of-mine)

Operating Costs	$/tonne of pellet
Mine production	24.02
Concentration and slurry transportation	16.86
Pellet production and handling	11.45
G&A and site services	4.84
Ship loading	2.00
Total Average LOM Operating Costs	**59.17**

Table 21.7 – Total Operating Costs (Average first 5 years)

Operating Costs	$/tonne of pellet
Mine production	18.09
Concentration and slurry transportation	17.27
Pellet production and handling	11.45
G&A and site services	4.84
Ship loading	2.00
Total Average LOM Operating Costs	**53.65**

Table 21.7 presents the average operating costs for the first 5 years of operation. The operating costs for the first 5 years are lower due to lower stripping ratio and slightly lower weight recovery

21.3 Mine Operating Costs

The mine operating cost was estimated for each period of the mine plan. This cost is based on operating the mining equipment, the manpower associated with operating the mine, the cost for explosives as well as dewatering, road maintenance, leasing costs of the mining equipment and other activities.

In order to determine the operating cost, the following assumptions were used;

- Diesel Fuel Price – $1.05/litre;
- Explosives Cost – $7,021,600/yr. + $0.29/t (based on supplier pricing for similar project);
- Clearing and Grubbing – $2,000/ha;
- Topsoil removal – $18,000/ha.

The mine operating cost was estimated to average $24.02per tonne of pellet or $2.50 per tonne mined for the life of the mine. This cost is divided into $2.44/t for mineralization, $2.61/t for waste rock and $1.70/t for overburden, as presented in Table 21.8.

Table 21.8 – Mine Operating Costs

Type of Material	Operating Cost ($/t mined)
Mineralization	2.44
Waste	2.61
Overburden	1.70

21.3.1 Mine Operating Cost Breakdown by Major Components

Table 21.9 provides a breakdown of the mine operating cost into several major components.

Table 21.9 – Operating Cost Breakdown (Activities and Manpower)

Category	Mined ($/t)	Pellet ($/t)	Total (%)
Loading	0.21	2.03	8
Hauling	1.21	11.64	48
Drilling & Blasting	0.52	4.98	21
Support & Service	0.18	1.76	7
Manpower	0.36	3.44	14
Other	0.02	0.18	1
Total	2.50	24.02	100

21.3.2 Mining Equipment

The hourly operating cost for the major mining equipment was supplied by the equipment manufacturers and suppliers. These were used to develop the operating cost estimate. For certain equipment where hourly cost estimates were not obtained, Met-Chem used its internal database.

21.3.3 Manpower

The total mine workforce for the Project ranges from 251 employees in Year 1 to a maximum of 419 from Years 11 to 20. This workforce is comprised of staff as well as hourly employees. Table 21.10 shows the mine manpower requirements for Year 4.

Table 21.10 – Mine Manpower Requirements (Year 4)

Description	# Employees
Mine Superintendent	1
Maintenance Superintendent	1
Engineering Supervisor	1
Mining Engineer	4
Geologist	4
Grade Control Technician	4
Planning Technician	4
Surveyor	4
Pit Foreman	8
Drill and Blast Foreman	4
Equipment Operators	188
Fuel and Lube Truck Driver	12
Labourers	16
Dewatering Crew	8
Power Distribution Crew	8
Dispatchers	4
Trainers	4
Maintenance Foreman	8
Warehouse clerk	4
Maintenance Planner	4
Mechanics	56
Total Mine Workforce (Year 4)	347

All employees and contractors will be housed at a camp site near the mine.

21.4 Process Operating Costs

For a typical year at a 12 Mtpy pellets production rate, the process operating costs (summarized in Table 21.11) are divided into the beneficiation plant and the port area (pellet plant and pellet handling). For the beneficiation plant, the operating costs are subdivided into these components: manpower (labour), electricity, consumables and wear parts consumption, grinding media and reagents and material handling.

These costs were derived from supplier information, Met-Chem's database or factored from similar operations.

Table 21.11 - Summary of Average Annual Process Operating Costs

Operating Cost Area	Cost ($ Millions/y[1])	Cost ($/tonne of final product[2])	Total Costs (%)
Beneficiation Plant			
Manpower	18	1.51	5.3%
Electricity[4)]	82	6.83	24.1%
Consumables and Wear Parts Consumption	15	1.24	4.4%
Grinding Media and Reagents	84	6.98	24.7%
Material Handling [5,6)]	3	0.30	1.1%
Sub-total	**202**	**16.86**	**59.6%**
Pellet Plant [5]	134	11.22	39.6%
Pellet Handling	3	0.23	0.8%
Sub-total	**137**	**11.45**	**40.4%**
Total	**339**	**28.31**	**100.0%**

1) Canadian Dollar unless noted otherwise (holds for all other tables).
2) Based on production of 12 Mtpy of acid pellets.
3) Based on an average plant throughput of 41,284,059 tonnes per year.
4) Power cost is CAD 0.09 /kWh for Primary Crushing, Primary Grinding, Cobber Separation (primary transformation). Secondary Grinding Area and all other areas (secondary transformation) are at CAD 0.045 /kWh.
5) Based on LFO price of USD 1.05 per litre and HFO price of USD 0.62 per liter.
6) Includes Slurry & Water Pipelines Operating Costs (the pumps electrical power are in Electricity costs)

Table 21.12 shows all the operating costs details of the Pellet Plant located at the port.

Table 21.12 - Summary of Pellet Plant Operating Costs

Description	Unit	Usage per Tonne of Pellet	Cost ($/t)
Limestone	kg	3	0.08
Dolomite	kg	0	0.00
Activator	kg	0.2	0.04
Bentonite	kg	4	0.60
Fuel Oil (Bunker C)	liter	6.8	4.20
Electricity – Total	kWh	35	1.58
Filter bags & sectors	pcs	0.001	0.04
Rollers	pcs	0.000015	0.04
Refractories	kg	0.015	0.03
Spares	$		2.00
Grate bars	pcs	0.0004	0.01
Other consumables	$		1.13
Labour	m-h	0.028	1.44
Total Cost			11.22

Note: 1 liter of Fuel Oil (Bunker C) will generate ≈ 42 MJ.

21.4.1 Labour Cost

In the Beneficiation Plant, it is estimated that there will be 163 employees. This includes the supervision staff for the process plant, crushing, the operation hourly employees as well as the mechanical and electrical repairmen for the same areas. The total annual cost of labour for the Beneficiation Plant is estimated at $18 M per year. This corresponds to $1.51 per tonne of pellets produced. For the Pellet Plant, it is estimated that there will be 155 employees. The total annual cost for the Pellet Plant labour is estimated at $17.3 M per year. This corresponds to $1.44 per tonne of pellets produced. For the Pellet Handling (area 425), it is estimated that there will be 16 employees. The total annual cost for the Pellet Handling labour is estimated at $1 M per year. This corresponds to $0.08 per tonne of pellets produced.

21.4.2 Electrical Power

Electrical power is required for the equipment in the process plant such as: crushers, grinding mills, conveyors, magnetic separators, pumps, services (compressed air and water), etc. The unit cost of electricity was established at $ 0.09/kWh for primary transformation (Primary Crushing, Primary Grinding & Cobber Separation). The unit cost of electricity was established at $0.045/kWh for secondary transformation: Secondary

Grinding Area and all other areas including Pelletizing and Pellet Handling. For the Beneficiation Plant, the annual estimated cost is $82 M or $6.83 per tonne of pellets produced. For the Pellet Plant, the estimated cost is $1.58 per tonne of pellets produced. For the Pellet Handling (area 425), the estimated cost is $1.5 M or $0.125 per tonne of pellets produced.

21.4.3 Consumables, Wear Parts, Grinding Media and Reagents

The consumables and reagents have been divided in three components that are described below:

- Consumables & Wear Parts;
- Grinding Media; and
- Reagents.

a) Consumables & Wear Parts

The consumption and cost for the bowls, mantles, screen decks and grinding mill liners for the different communition equipment was obtained from the equipment suppliers and from experience with similar operation. All the equipment requiring wear items having been taken into account (conveyors, magnetic separators, cyclones, thickeners, pumps, etc.). The annual cost for consumables and wear parts is estimated at $15 M or $1.24 per tonne of pellets produced.

b) Grinding Media

The grinding mills (SAG and Ball Mills) will need a regular addition of balls to replace the worn media and exercise the proper grinding action on the material. The media consumption has been estimated from the power input into the material based on steel consumption observed in similar operations. Balls will have to be added every day to maintain the steel load in the mills. The cost of grinding media for the grinding mills is estimated at $80 M per year or $6.69/tonne of pellets produced.

c) Reagents

Flocculant is required for the thickeners. The total cost of flocculant at the process plant is estimated at $4 M per year or $0.29/tonne of pellets produced.

21.4.4 Other Costs

Other costs such as site material handling for Beneficiation Plant were estimated at $4 M or $0.30/tonne of pellets produced. Material handling for Pellet Handling (area 425) was estimated at $0.2 M or $0.02/tonne of pellets produced.

21.5 General and Administration Costs

The General and Administration ("G&A") Costs for a typical year of 12 Mtpy pellets production rates are summarized in Table 21.13. The total annual G&A Operating Costs is estimated at $26 M or $2.12 per tonne of pellets.

Table 21.13 – Summary of General and Administration Costs

G&A Operating Costs	Cost ($ Millions/y)	Cost ($/tonne of Final Product)	Total G&A Costs (%)
Administration - Manpower	6	0.52	24.5
Administration - Material & Services	19	1.56	73.6
Subtotal	25	2.08	98.1
Technical services - Manpower	-	-	-
Technical services – Material & Services	1	0.04	1.9
Subtotal	1	0.04	1.9
Total	26	2.12	100

21.5.1 G&A Labour Cost

The G&A manpower is estimated at 58 employees. This includes management, finance, materials management, human resources and environmental. The total annual cost for G&A labour is estimated at $6 M per year or $0.52 per tonne of pellets.

21.5.2 Other G&A Costs

The G&A costs also covers administration material and services. This portion of the G&A costs accounts for $19 M per year or $1.56 per tonne of pellets.

This includes management and material services (security, leases, taxes, insurances, travel expenses for all employees, communication, office supplies, IT supplies and miscellaneous supplies), human resources, materials and environment supplies.

The technical services manpower is included in the mining operating costs. The technical services material and services accounts for $1 M per year or $0.04 per tonne of pellets. This includes computers (maintenance and supplies) engineering services and geology costs and laboratory consumables.

21.6 Site Services Costs

The site services ("SS") costs for a typical year of 12 Mtpy pellets production rates are summarized in Table 21.14.

Table 21.14 – Summary of Site Services Costs

SS Operating Costs	Cost ($ Millions/y)	Cost ($/tonne of Final Product)	Total SS Costs (%)
Infrastructure - Manpower	4	0.30	11.0
Infrastructure –Material & Services	29	2.42	89.0
Total	33	2.72	100

The total annual Site Services operating cost is estimated at $32.68 M or $2.72 per tonne of pellets.

21.6.1 Site Services Labour Costs

The Site Services manpower is estimated at 36 employees. This includes staff employees (superintendent and planner), hourly employees (electricians and general tradesmen) and equipment operators). The total costs for Site Services labour is estimated at $4 M per year or $0.30 per tonne of pellets.

21.6.2 Other Site Services Costs

The Site Services Costs also cover material and services. This portion of Site Services costs accounts for $29 M per year or $2.55 per tonne of pellets.

This includes but not limited to materials and services for camps room and board, potable water consumables, power for infrastructure, mobile equipment operation and maintenance (others than mining equipment) and power lines maintenance.

21.7 Shipping and ship loading costs

21.7.1 Shipping costs

Shipping costs are not part of the direct DLIP production costs but must be taken into account when estimating CFR prices to China. Shipping costs were estimated between the proposed port facilities at Stromness Island, James Bay, Quebec and Quindao, China and Rotterdam, Netherlands. Various shipping scenarios to China were evaluated for different ship size and routing options as described in the port and shipping study by Portha Inc. which can be found Appendix C. The shipping cost to Europe was evaluated in a previous shipping study done in 2010 by Mrs. Anna Klimek for Augyva.

The shipping costs take into account the following aspects:

- Period of the year (summer, winter, extender summer and extended winter);
- Distance of each travel section of each leg;
- Ship classification for Ice class to Polar class;
- Premium for Ice or Polar class over non Ice basic charter;
- Ship acquisition costs;
- Ship speed for each travel section for each season;

- Calculated ship power requirements;
- Calculated fuel consumption during travel fully loaded, at ballast and at port;
- Ship owning value including insurance, maintenance, etc.;
- Fixed time for docking and leaving berth;
- Time for loading and transfers;
- Daily charter at port, fully loaded and at ballast;
- Crew costs;
- Navigation requirements and regulations;
- Canal fees and tariffs.

The selected shipping scenario assumes the use of Capesize (185,000 dwt) and Suezmax (240,000 dwt) ships during the 4 month ice-free simmer season of James Bay. Costs are estimated at USD 35/t pellet for shipment to Quindao for 70% of the pellet production. The other 30% of the production would be shipped to Rotterdam at an estimated cost of USD 15/t. The average shipping cost taking into consideration the 70% to China and 30% to Europe averages USD 29/t.

21.7.2 Ship loading costs

Ship loading costs were evaluated by Met-Chem from similar projects at USD 2/t pellets. This costs takes into consideration the operation of the ship loaders, contract leasing and operation of tugboats during the shipping season, and unloading costs for bulk materials required for operation of the mine, concentrator and pellet plant (diesel fuel, HFO, pellet additives, grinding media, etc.).

22.0 ECONOMIC ANALYSIS

An economic/financial analysis has been carried out for the Duncan Lake Iron Project using an annual pellet production rate of 12 Mtpy.

A cash flow model is constructed on an annual basis in constant money terms first quarter 2013. No provision is made for the effects of inflation. As required in the financial assessment of investment projects, the evaluation is carried out on a so-called "100% equity" basis, i.e. the debt and equity sources of capital funds are ignored. The model reflects the base case macro-economic and technical assumptions given in this report.

22.1 Macro-Economic Assumptions

22.1.1 Main Assumptions

The main base case macro-economic assumptions used are given in Table 22.1.

A long-term 62% iron concentrate C.F.R. China price forecast of USD 125/t, based on a market study prepared by Raw Materials & Ironmaking Global Consulting Services (2013), is used to determine the F.O.B James Bay price, the location from which the pellets are to be shipped. Assuming that the pellets produced grade 66,3% Fe and are sold in both China and Europe, a weighted-average CFR price of USD 169/t of pellets is determined (using the pellet premium, transport costs and market split shown below). The sensitivity analysis examines a range of iron concentrate prices 30% above and below the base case price.

Table 22.1 – Macro-Economic Assumptions

Item	Unit	Base Case Value
62% Iron Concentrate Price (C.F.R. China)	USD/t	125
66.3% Iron Duncan Lake Concentrate	USD/t	134
Pellet Premium	USD/t	35
Shipping Cost to China	USD/t	37
Shipping Cost to Europe	USD/t	17
Market Split	China : Europe	70:30
Duncan Lake Pellet Price (C.F.R China)	USD/t	169
Production Period Exchange Rate	USD/CAD	0.95
Discount Rate 1	% per year	5.0
Discount Rate 2	% per year	8.0

22.1.2 Exchange Rate Assumptions

A short-term exchange rate of 1.00 USD/CAD is assumed during the pre-production period, decreasing to a long-term rate of 0.95 USD/CAD during the production period. It is supposed that this assumption has little effect on operating expenses and sustaining capital costs. Thus, only the price forecast responds to the decrease in exchange rate.

22.1.3 Tax Assumptions

The current Canadian tax system applicable to mining income is used to assess the project's annual tax liabilities. This consists of federal and provincial corporate taxes as well as provincial mining taxes (revised in the 2010 budget). The revisions announced in the March 21, 2013 federal budget speech concerning the reclassification of mine development expenses from Canadian Exploration Expenses ("CEE") to Canadian Development Expenses ("CDE"), and the elimination of the provision for accelerated depreciation for class 41A assets have been accounted for. Both changes will be made progressively over a period of several years. It is assumed that Quebec will follow suit with the same changes in the provincial corporate tax rules. The federal and provincial corporate tax rates currently applicable over the project's operating life are 15.0 % and 11.9 % of taxable income, respectively. The rate applicable for the purpose of assessing Quebec mining taxes is 16 % of taxable income.

22.1.4 Discount Rate Assumptions

The discount rate variants used to determine the net present value are meant to represent typical weighted-average costs of capital. Results are presented on pre-tax and post-tax bases.

22.1.5 Royalties

As mentioned in Section 4.3, DLIP is required to pay Virginia Mines a royalty of $ 0.40 per tonne of iron concentrate. This amount is taken into consideration in the pre-tax and after-tax economic evaluation.

22.2 Technical Assumptions

The key technical assumptions used in the analysis are shown in Table 22.2.

Table 22.2 – Technical Assumptions

Item	Unit	Value
Life of Mine Mill Feed (for financial analysis)	Mt	817.4
Average Grade	% Fe	24.8
Average Weight Recovery	%	27.98
Average Stripping Ratio	Waste/Mineral	1.80
Mine Life (for financial analysis)	Years	20
Concentrate Production (67% Fe)	Mt	228.7
Pellet Production (66% Fe)	Mt	237.6
Operating Costs		
Mining		
Mineralization (average based on financial analysis)	CAD/t Mineralization	2.44
Waste (average based on financial analysis)	CAD/t waste	2.61
Overburden (average based on financial analysis)	CAD/t OB	1.70
Processing		
Beneficiation & Slurry Transport	CAD/t mill feed	4.90
Pelletizing & Pellet Handling	CAD/t pellets	11.45
Other		
G&A Costs	CAD/t pellets	2.12
Site Services	CAD/t pellets	2.72
Ship Loading	CAD/t pellet	2.00
Total (average based on financial analysis)	CAD/t mill feed	17.20
	CAD/t pellets	59.17
Pre-production Capital Costs and ramp up (excluding WC)	CAD M	3,880.9
Sustaining Capital Costs	CAD M	508.9
Closure Costs	CAD M	156.4

22.3 Financial Analysis Results

The financial evaluation results based on the parameters presented in Table 22.2 and are summarized in Table 22.3.

On a pre-tax basis, the Net Present Value is CAD 6,849.3 M (at 5% discount rate) and CAD 4,144.1 M (at 8% discount rate). The project has an internal rate of return of 20.1 % and a payback period of 4.2 years. The cash flow statement and financial indicators are presented in Table 22.5.

On a post-tax basis, the Net Present Value is CAD 4,041.8 M (at 5% discount rate) and CAD 2,238.4 M (at 8% discount rate). The project has an internal rate of return of 15.9 % and a payback period of 4.8 years.

Table 22.3 – Financial Analysis Results

Item	Unit	Value
Total Revenue	CAD M	33,764.2
Total Operating Costs	CAD M	13,585.0
Total Pre-Production Capital Costs	CAD M	3,880.9
Total Sustaining Capital Costs	CAD M	508.9
Total Closure Costs	CAD M	156.4
Pre-Tax basis		
Total Cash Flow	CAD M	15,633.0
Payback Period	Years	4.2
Net Present Value @ 5%	CAD M	6,849.3
Net Present Value @ 8%	CAD M	4,144.1
Internal Rate of Return	%	20.1
Post-Tax basis		
Total Cash Flow	CAD M	9,890.6
Payback Period	Years	4.8
Net Present Value @ 5%	CAD M	4,041.8
Net Present Value @ 8%	CAD M	2,238.4
Internal Rate of Return	%	15.9

22.4 Sensitivity Analysis

A sensitivity analysis has been carried out, with the base case described above as a starting point, to assess the impact of changes in the 62% iron concentrate price (“PRICE”), total pre-production capital costs (“CAPEX”) and operating costs (“OPEX”) on the project’s NPV (@ 5% and 8%) and IRR. Each variable is examined one-at-a-time. An interval of ±30% with increments of 10% is used for all three variables, while keeping all other parameters fixed.

Figure 22.1 to Figure 22.3 show the results of the sensitivity analysis on a pre-tax basis. These indicate that the project’s viability is not significantly vulnerable to variations in capital and operating costs estimates, taken one at-a-time. The NPV is more sensitive to variations in operating expenses, as shown by the steeper curves on the NPV diagrams. However, as expected, the NPV is most sensitive to variations in price, and becomes marginal at the lowest price examined (an iron concentrate price of USD 87.50/t). Figure 22.3 indicates that the IRR is more sensitive to variations in capital costs than operating costs, as shown by the steeper slopes. Here as well, the IRR is most sensitive to variations in price, and becomes marginal at the lowest price examined.

Figure 22.4 to Figure 22.6 show the results of the sensitivity analysis on a post-tax basis. The same conclusions as those noted for the pre-tax situation can be drawn concerning the sensitivity of the post-tax financial indicators. As seen in Figure 22.5, the NPV @ 8% becomes negative at an iron concentrate price of about USD 91.00/t, i.e., the IRR drops below the level of 8 %.

The financial performance of the project given a Canadian dollar at par with the American dollar is also examined. The financial results are compared in Table 22.4. The higher long-term value of the Canadian dollar reduces the value of the project (i.e. Total Cash Flow and NPVs) by an average of about 14 percent and results in a rate of return 1.7 percentage points lower on a pre-tax basis and 1.4 percentage points lower on a post-tax basis.

Table 22.4 – Sensitivity to CAD – USD Exchange Rate Assumption

Item	CAD at Par	Base Case
Pre-Tax Basis		
Total Cash Flow (CAD M)	13,944.8	15,633.0
Payback Period (years)	4.6	4.2
Net Present Value @ 5% (CAD M)	5,945.4	6,849.3
Net Present Value @ 8% (CAD M)	3,492.1	4,144.1
Internal Rate of Return (%)	18.4	20.1
Post Tax Basis		
Total Cash Flow CAD M)	8,847.0	9,890.6
Payback Period (years)	5.2	4.8
Net Present Value @ 5% (CAD M)	3,473.8	4,041.8
Net Present Value @ 8% (CAD M)	1,823.5	2,238.4
Internal Rate of Return (%)	14.5	15.9

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA based on these Mineral Resources will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 22.5 – Cash Flow Statement

Year	PP -3	PP -2	PP -1	1	2	3	4	5	6	7
CONCENTRATE PRODUCTION (Mt)				9.24	11.55	11.55	11.55	11.55	11.55	11.55
PELLET PRODUCTION (Mt)– 66.3% Fe				9.60	12.00	12.00	12.00	12.00	12.00	12.00
PELLET SALES PRICE – FOB James Bay (USD/t)				135.465	135.465	135.465	135.465	135.465	135.465	135.465
PELLET SALES PRICE – including Exchange Rate (0.95 USD/CAD)				142.59	142.59	142.59	142.59	142.59	142.59	142.59
TOTAL ANNUAL REVENUE (M$)				**1,368.91**	**1,711.14**	**1,711.14**	**1,711.14**	**1,711.14**	**1,711.14**	**1,711.14**
OPERATING EXPENSES										
MINING COSTS				144.76	195.75	218.89	234.82	247.59	300.15	300.15
BENEFICATION PLANT & SLURRY TRANSPORT				165.53	207.06	210.85	210.68	200.76	200.76	200.76
PELLET PLANT & LOADING				109.93	137.41	137.41	137.41	137.41	137.41	137.41
G&A COSTS				20.35	25.44	25.44	25.44	25.44	25.44	25.44
SITE SERVICES				26.11	32.64	32.64	32.64	32.64	32.64	32.64
ANNUAL OPERATING COSTS(M$)				**466.69**	**598.30**	**625.23**	**640.99**	**643.84**	**696.40**	**696.40**
ROYALTY				3.70	4.62	4.62	4.62	4.62	4.62	4.62
ANNUAL OPERATING COSTS (Royalty Payment) (M$)				**470.38**	**602.92**	**629.85**	**645.61**	**648.46**	**701.02**	**701.02**
CAPITAL EXPENSES - PRE PRODUCTION										
MINE DEVELOPMENT		29.87	29.87							
OPEN-PIT MINE		15.78	15.78	48.47						
ORE PROCESS	119.83	319.54	359.48							
CONCENTRATE AND RECLAIM WATER PIPELINES	60.16	160.44	180.49							
PELLET PLANT	214.45	571.86	643.35							
PORT	108.24	288.64	324.72							
TAILINGS AND WATER MANAGEMENT FACILITIES	7.83	20.89	23.50							
INFRASTRUCTURE – MINE SITE	12.95	34.54	38.85							
POWER AND COMMUNICATIONS	34.91	93.09	104.72							
SERVICE VEHICLES AND EQUIPMENT	2.80	7.46	8.39							
WORKING CAPITAL (8 Months)			155.56	243.31	17.95	10.51	1.90	35.04		
CAPITAL EXPENSES - SUSTAINING										
OPEN-PIT MINE					5.54	5.45				
ORE PROCESS					1.78	1.78	1.78	1.78	1.78	1.78
CONCENTRATE AND RECLAIM WATER PIPELINES										
PELLET PLANT								5.60		
PORT										
TAILINGS AND WATER MANAGEMENT FACILITIES						26.18	24.69	24.69	24.69	24.69
INFRASTRUCTURE – MINE SITE								9.20	3.30	
POWER AND COMMUNICATIONS										
SERVICE VEHICLES AND EQUIPMENT									4.81	4.81
CLOSURE COSTS						0.00	1.25	3.91	6.41	9.07
TOTAL ANNUAL CAPITAL EXPENSES (M$)	**561.17**	**1,542.10**	**1,884.71**	**291.77**	**25.27**	**43.92**	**29.62**	**80.22**	**40.99**	**40.35**
TAX										
FEDERAL CORPORATE TAXES							90.64	122.77	117.95	119.42
QUEBEC CORPORATE TAXES							71.91	97.40	93.57	94.74
QUEBEC MINING TAXES					6.79	51.83	77.60	95.66	100.33	109.35
TOTAL ANNUAL TAX PAYMENTS (M$)					**6.79**	**51.83**	**240.14**	**315.83**	**311.85**	**323.52**
CASH FLOW										
ANNUAL B-T CASH FLOW (M$)	**-561.17**	**-1,542.10**	**-1,884.71**	**606.75**	**1,082.95**	**1,037.37**	**1,035.91**	**982.46**	**969.12**	**969.76**
CUMULATIVE B-T CASH FLOW (M$)	**-561.17**	**-2,103.26**	**-3,987.97**	**-3,381.22**	**-2,298.27**	**-1,260.90**	**-225.00**	**757.46**	**1,726.58**	**2,696.35**
PAYBACK PERIOD WORK AREA			1.00	1.00	1.00	1.00	1.00	0.23		
ANNUAL A-T CASH FLOW (M$)	**-561.17**	**-1,542.10**	**-1,884.71**	**606.75**	**1,076.16**	**985.54**	**795.76**	**666.62**	**657.27**	**646.25**
CUMULATIVE A-T CASH FLOW (M$)	**-561.17**	**-2,103.26**	**-3,987.97**	**-3,381.22**	**-2,305.06**	**-1,319.52**	**-523.76**	**142.87**	**800.14**	**1,446.39**
PAYBACK PERIOD WORK AREA			1.00	1.00	1.00	1.00	1.00	0.79		

Year	8	9	10	11	12	13	14	15	16	17	18	19	20	TOTAL
CONCENTRATE PRODUCTION (Mt)	11.55	11.55	11.55	11.55	11.55	11.55	11.55	11.55	11.55	11.55	11.55	11.55	11.55	228.69
PELLET PRODUCTION (Mt)– 66.3% Fe	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	12.00	237.6
PELLET SALES PRICE – FOB James Bay (USD/t)	135.465	135.465	135.465	135.465	135.465	135.465	135.465	135.465	135.465	135.465	135.465	135.465	135.465	
PELLET SALES PRICE – including Exchange Rate (0.95 USD/CAD)	142.59	142.59	142.59	142.59	142.59	142.59	142.59	142.59	142.59	142.59	142.59	142.59	142.59	
TOTAL ANNUAL REVENUE (M$)	**1,711.14**	**1,711.14**	**1,711.14**	**1,711.14**	**1,711.14**	**1,711.14**	**1,711.14**	**1,711.14**	**1,711.14**	**1,711.14**	**1,711.14**	**1,711.14**	**1,711.14**	**33,880.51**
OPERATING EXPENSES														
MINING COSTS	300.15	300.15	300.15	317.37	317.37	317.37	317.37	317.37	315.73	315.73	315.73	315.73	315.73	5,708.08
BENEFICATION PLANT & SLURRY TRANSPORT	200.76	200.76	200.76	200.76	200.76	200.76	200.76	200.76	200.76	200.76	200.76	200.76	200.76	4,006.20
PELLET PLANT & LOADING	137.41	137.41	137.41	137.41	137.41	137.41	137.41	137.41	137.41	137.41	137.41	137.41	137.41	2,720.76
G&A COSTS	25.44	25.44	25.44	25.44	25.44	25.44	25.44	25.44	25.44	25.44	25.44	25.44	25.44	503.71
SITE SERVICES	32.64	32.64	32.64	32.64	32.64	32.64	32.64	32.64	32.64	32.64	32.64	32.64	32.64	646.27
ANNUAL OPERATING COSTS(M$)	**696.40**	**696.40**	**696.40**	**713.62**	**713.62**	**713.62**	**713.62**	**713.62**	**711.98**	**711.98**	**711.98**	**711.98**	**711.98**	**13,585.02**
ROYALTY	4.62	4.62	4.62	4.62	4.62	4.62	4.62	4.62	4.62	4.62	4.62	4.62	4.62	91.48
ANNUAL OPERATING COSTS (Royalty Payment) (M$)	**701.02**	**701.02**	**701.02**	**718.24**	**718.24**	**718.24**	**718.24**	**718.24**	**716.60**	**716.60**	**716.60**	**716.60**	**716.60**	**13,676.50**
CAPITAL EXPENSES - PRE PRODUCTION														
MINE DEVELOPMENT														59.74
OPEN-PIT MINE														80.04
ORE PROCESS														798.84
CONCENTRATE AND RECLAIM WATER PIPELINES														401.09
PELLET PLANT														1,429.66
PORT														721.60
TAILINGS AND WATER MANAGEMENT FACILITIES														52.22
INFRASTRUCTURE – MINE SITE														86.34
POWER AND COMMUNICATIONS														232.72
SERVICE VEHICLES AND EQUIPMENT														18.64
WORKING CAPITAL (8 Months)			11.48					-1.09					-474.65	0.00
CAPITAL EXPENSES - SUSTAINING														
OPEN-PIT MINE				12.35	12.35	12.35								48.06
ORE PROCESS	1.78	1.78	1.78	1.78	1.78	1.78	1.78	1.78	1.78	1.78	1.78	1.78		32.03
CONCENTRATE AND RECLAIM WATER PIPELINES			7.63				2.10							9.73
PELLET PLANT														5.60
PORT														0.00
TAILINGS AND WATER MANAGEMENT FACILITIES	24.69		18.61	17.35	58.68	41.89	40.41	45.53						372.10
INFRASTRUCTURE – MINE SITE														12.50
POWER AND COMMUNICATIONS														0.00
SERVICE VEHICLES AND EQUIPMENT	4.81					4.81	4.81	4.81						26.86
CLOSURE COSTS	11.57	14.23	16.74	19.40	22.05	24.56	27.22							156.41
TOTAL ANNUAL CAPITAL EXPENSES (M$)	**42.85**	**16.01**	**56.23**	**50.88**	**94.86**	**85.39**	**76.31**	**51.03**	**1.78**	**1.78**	**1.78**	**1.78**	**-474.65**	**4,546.16**
TAX														
FEDERAL CORPORATE TAXES	120.54	122.25	122.73	120.76	119.60	119.04	118.89	121.68	123.42	124.59	125.48	126.16	122.98	2,038.91
QUEBEC CORPORATE TAXES	95.63	96.99	97.37	95.80	94.88	94.44	94.32	96.53	97.91	98.84	99.55	100.09	97.56	1,617.53
QUEBEC MINING TAXES	115.46	120.87	123.19	121.60	120.17	119.55	119.38	128.43	131.34	133.20	134.49	135.40	161.15	2,105.79
TOTAL ANNUAL TAX PAYMENTS (M$)	**331.64**	**340.11**	**343.28**	**338.16**	**334.66**	**333.02**	**332.60**	**346.64**	**352.67**	**356.63**	**359.52**	**361.65**	**381.69**	**5,762.23**
CASH FLOW														
ANNUAL B-T CASH FLOW (M$)	**967.26**	**994.10**	**953.88**	**942.02**	**898.03**	**907.50**	**916.59**	**941.87**	**992.76**	**992.76**	**992.76**	**992.76**	**1,469.19**	**15,657.85**
CUMULATIVE B-T CASH FLOW (M$)	**3,663.61**	**4,657.71**	**5,611.60**	**6,553.62**	**7,451.65**	**8,359.16**	**9,275.75**	**10,217.62**	**11,210.38**	**12,203.14**	**13,195.90**	**14,188.66**	**15,657.85**	
PAYBACK PERIOD WORK AREA														
ANNUAL A-T CASH FLOW (M$)	**635.62**	**654.00**	**610.60**	**603.86**	**563.38**	**574.48**	**583.99**	**595.23**	**640.09**	**636.13**	**633.24**	**631.11**	**1,087.50**	**9,895.62**
CUMULATIVE A-T CASH FLOW (M$)	**2,082.01**	**2,736.00**	**3,346.60**	**3,950.47**	**4,513.84**	**5,088.32**	**5,672.31**	**6,267.55**	**6,907.63**	**7,543.77**	**8,177.01**	**8,808.12**	**9,895.62**	
PAYBACK PERIOD WORK AREA														

FINANCIAL INDICATORS

BEFORE TAX	
PAYBACK PERIOD (years)	4.23
NPV @ 5 % ($)	6,862,559,578
NPV @ 8 % ($)	4,153,694,834
INTERNAL RATE OF RETURN	20.11%
AFTER TAX	
PAYBACK PERIOD (years)	4.79
NPV @ 5 % ($)	4,044,764,046
NPV @ 8 % ($)	2,240,628,832
INTERNAL RATE OF RETURN	15.86%

F.O.B. Price Forecast	Conc./Pellet Grad	Price (USD/t)
Price CFR China (USD/mt)	62.0%	125 $
Adjustment for pellet grade	66.3%	134 $
Pellet Premium		35 $
Total		**169 $**
Shipping J-Bay to China (USD/t)	(70% of producti	35.00 $
Ship loading costs (USD/t)		2.00 $
FOB James Bay for China pellets (USD/t)		**132.00 $**
European Price discount	5%	8.45 $
Shipping J-Bay to Europe (USD/t)	(30% of producti	15.00 $
Ship loading costs (USD/t)		2.00 $
FOB James Bay for Europe pellets (USD/t)		**143.55 $**
OVERALL Average FOB James Bay (USD/t)		**135.47 $**

Figure 22.1 – Pre-tax $NPV_{5\%}$: Sensitivity to Pre-production Capital Cost, Operating Cost and Price

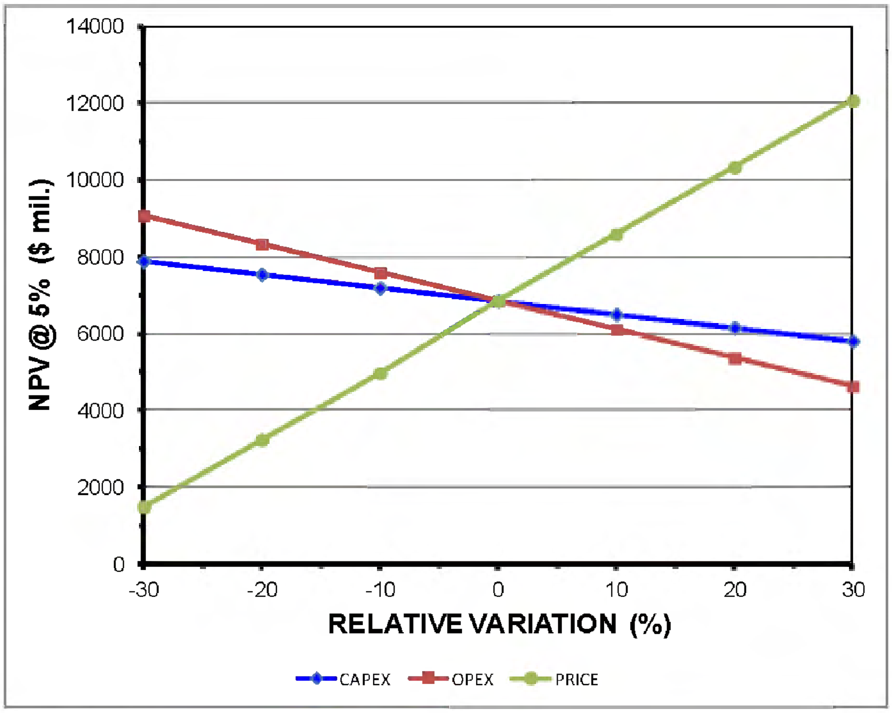


Figure 22.2 – Pre-tax $NPV_{8\%}$: Sensitivity to Pre-production Capital Cost, Operating Cost and Price

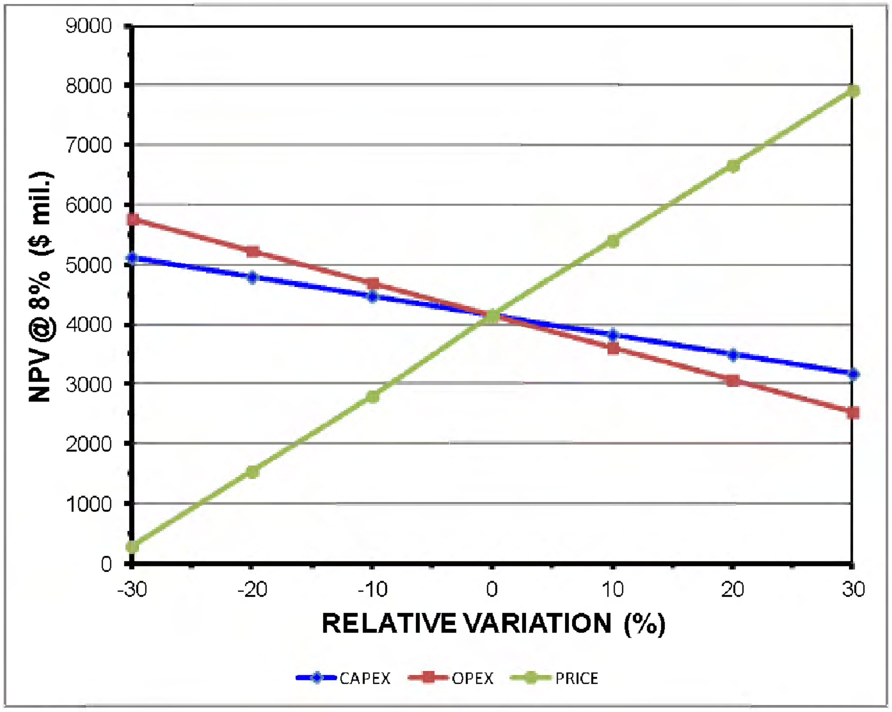


Figure 22.3 – Pre-tax IRR: Sensitivity to Pre-production Capital Cost, Operating Cost and Price

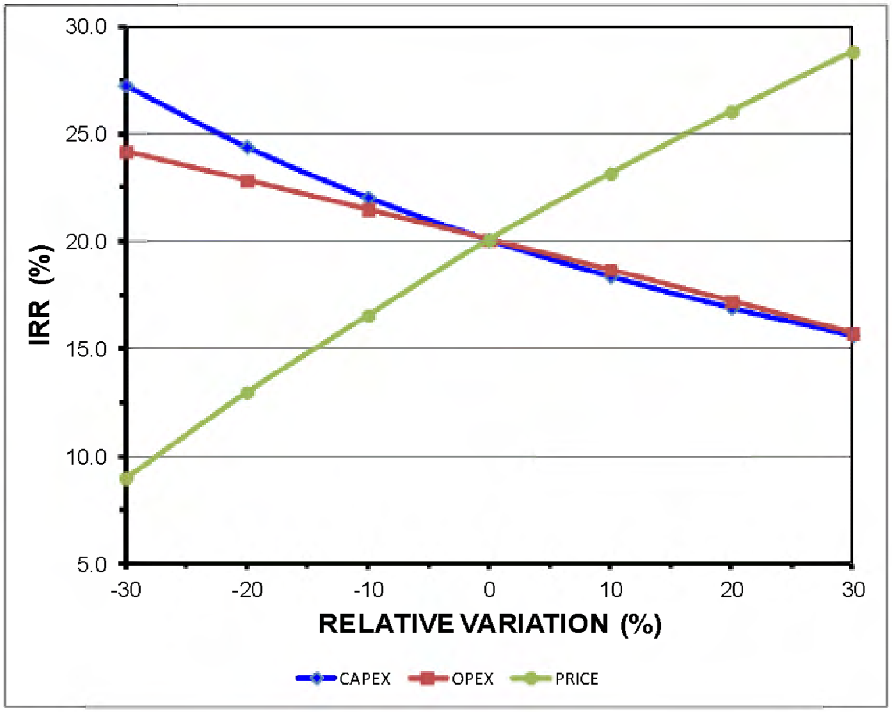


Figure 22.4 – Post-tax NPV$_{5\%}$: Sensitivity to Pre-production Capital Cost, Operating Cost and Price

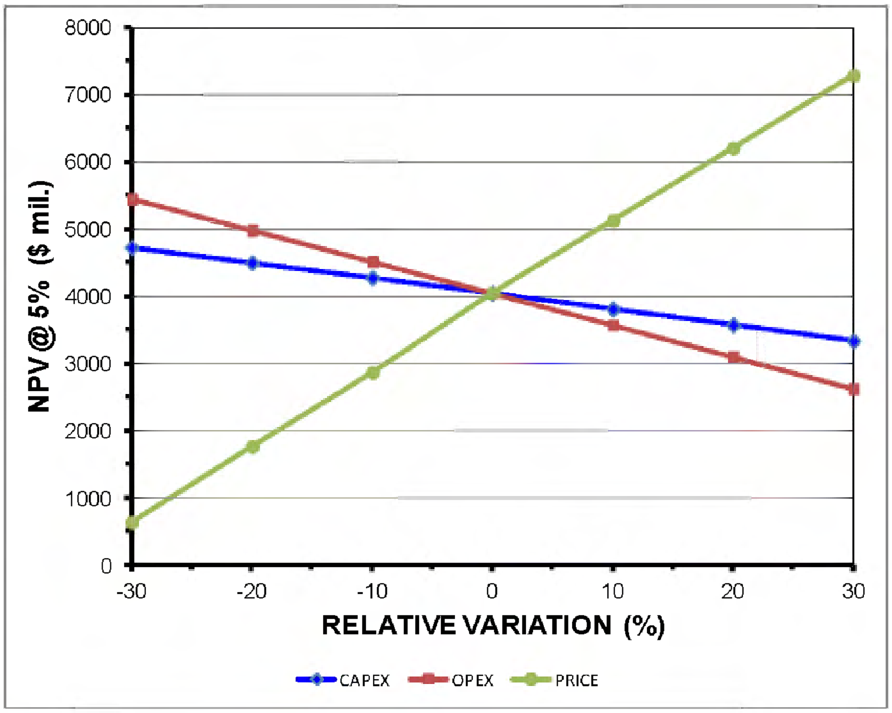


Figure 22.5 – Post-tax $NPV_{8\%}$: Sensitivity to Pre-production Capital Cost, Operating Cost and Price

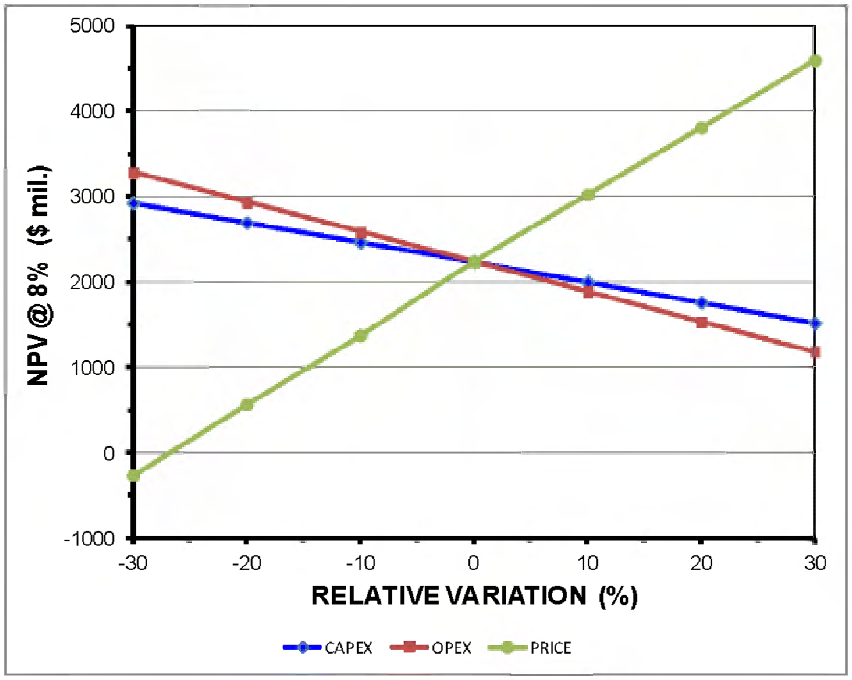
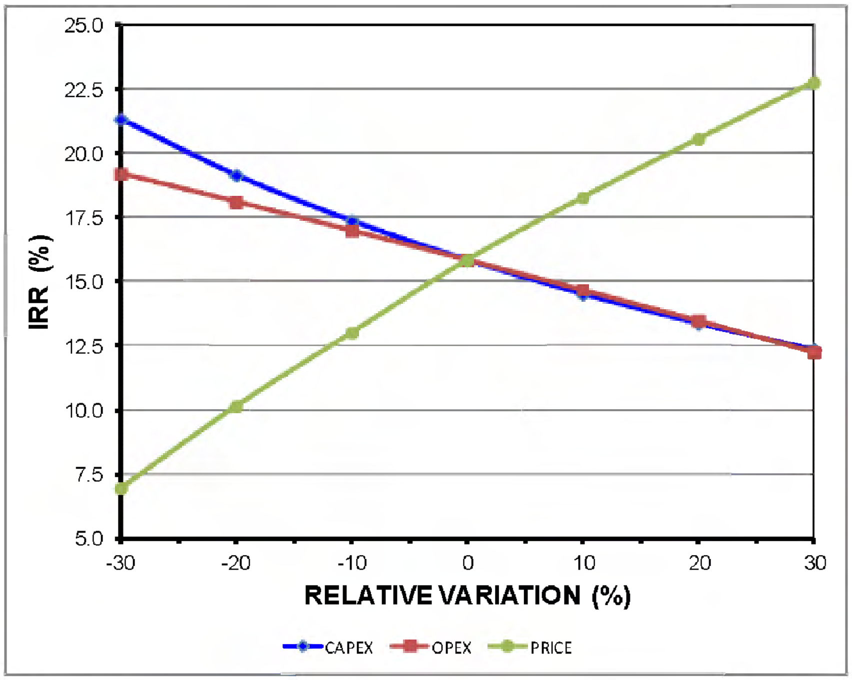


Figure 22.6 – Post-tax IRR: Sensitivity to Pre-production Capital Cost, Operating Cost and Price



22.5 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 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.

The results of the economic analysis are forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here, including :

- Drill core assays (reliability of Davis Tube results);
- Mineral processing (reliability of mineral test results);
- Mining (pit slope assumptions and ground water inflows);
- Transport (pipeline routing, bathymetry at port location);
- Commercial (iron ore price fluctuations, shipping costs);
- Environmental (ARG);
- Social and community issues;
- Governmental and Political (royalties, taxes, regulations);

These risks will be subject to further definition and mitigation in the next phase of the project to eliminate or minimize their potential impact on DLIP.

23.0 ADJACENT PROPERTIES

There is no mining activity in the area according to MNR GESTIM claim management system. There is no mining property or claims adjacent to the DLIP. However, an exclusion Block controlled by Hydro-Québec extends across part of the DLIP group of claims and may affect potential development of Deposits 1, 2 and 6.

The closest mineral property to the DLIP claims is the Duncan Property owned by NQ Resources Inc. with offices in Laval and Rouyn-Noranda, Quebec. This property is made up of two Blocks totalling 139 map designated claims covering 7,117 ha. It is located to the northwest of the DLIP claims and has no common boundary with the DLIP. The NQ Resources' property is described in their website as an Au, Ag, Cu and Zn prospect within sericite schists. The QP has been unable to verify this information considering this information is not indicative of the mineralization on the DLIP as NQ Resources' target on this property is not described as an iron deposit.

24.0 OTHER RELEVANT DATA AND INFORMATION

No other relevant information.

25.0 INTERPRETATION AND CONCLUSIONS

The DLIP is planned as a 20 year operation producing 12 Mtpy of acid pellets, with its mine and concentrator situated close to the town of Radisson, in northern Québec, and its pellet plant and port would be located near the town of Chisasibi on the shores of James Bay, some 135 km away from the mine. The port would ship the pellets on ocean-going vessels during the 4 month ice-free shipping period. The project is also in very close proximity to Hydro Quebec's La Grande hydroelectric complex. The drilling program of 2011-2012 and the data from the 2008-2009 holes allowed defining ~75% of the Mineral Resources in Deposits 3, 4 and 6 in the Measured and Indicated categories. The two drill programs have been successful in providing sufficient data on all six DLIP Deposits to produce in 2012, new or updated Mineral Resource estimates totalling 1,051 Mt of Measured and Indicated resources grading 24.42% Fe and 563 Mt of Inferred resources grading 24.69% Fe. The DLIP deposits that were considered for the PEA (Deposits 3 and 4) contain an estimated total Measured and Indicated Resources of 797 Mt at 24.44% Fe, and 277 Mt of Inferred Resources grading 25.07% Fe.

The present mineral resource estimation is compliant with the CIM Definitions, in accordance with NI 43-101 and Met-Chem believes to be a sound foundation for the PEA.

In-Pit resources used for the mine plan and the economic evaluation were estimated by designing a pit around an optimal economic pit defined by the Lerch Grossman method. An estimated 660 Mt of Measured and Indicated resources and 157 Mt of Inferred resources would produce 12 Mtpy of pellets over 20 years with an average stripping ratio of 1.8:1.

The PEA's economic evaluations shows that, using an 8% discount rate and an initial investment of $ 3.8 billion, Century would obtain a potential positive return based on a pre-tax scenario of NPV of $ 4.1 billion, 20.1% IRR and 4.2 year payback, An after-tax scenario shows an NPV of $ 2.2 billion, 15.9% IRR and 4.8 year payback. The accuracy of the cost estimates is ± 35%.

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. The results of the economic analysis are forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

Based on the results of the PEA, Met-Chem recommends that Century continues to the next phase of project development.

26.0 RECOMMENDATIONS

Considering the positive results of DLIP PEA and discussions with Century, Met-Chem recommends that the project continues to the next phase of DLIP development, the Feasibility Study.

To establish a good base for the feasibility study and minimize the risks, Met-Chem recommends a series of studies and tests which are listed below: The main recommendations include:

- Increase the percentage of Measured and Indicated category relative to the Inferred Resources within Deposits 1 and 3 by additional diamond drilling;
- Firm up the definition of the geometry of Deposit 3, particularly the SE limb and the contact at depth of the synform;
- Investigate by a first pass of drilling some of the magnetic anomalies near the main deposits, such as the N-S trending anomalies of Deposits 3 and 6, or the anomaly branching off the north of Deposit 4;
- Increase the number of Davis Tube tests to 50% of the samples to improve the confidence level of the regression model and provide a better overall estimation of the Davis Tube Weight Recovery for the deposits;
- Determine the magnetic Fe content from Davis Tube and Satmagan tests on the same samples in order to calculate a correlation between the two;
- Use certified blank material and commercial standards, with certified Fe values close to the cut-off grade to the mode to monitor the laboratory performance;
- Perform a geotechnical analysis to increase pit wall slope and angle of repose of waste and overburden material, as well as hydrogeological and hydrological studies;
- Revisit the sequencing of Pushbacks for the Deposit 3 to maximize the project's NPV;
- Explore the potential of stockpiling and mining within Hydro-Québec property to be able to increase in-pit resources and shorten haul distances;
- Consider in-pit dumping to reduce environmental footprint and shorten haulage distances;
- Perform geochemistry study on more samples for better characterization and to confirm process conditions;
- Acid generation tests should be performed in order to know if there is a possibility of acid-generation on tailings and waste rock. Static testing has been performed and dynamic characterisation tests have to be carried out on the tailings;
- Perform grind size determination/optimization studies for all deposits (typical standard in taconite plant is a grind size of 44 micron (325 mesh);

- Perform mineralogical study on the iron mineralization to characterize the mineral species and to know the liberation size;
- Perform for each deposit, batch bench scale test work to confirm the flow sheet for the development of an overall magnetite processing plant;
- Obtain additional crusher, ball mill and rod mill bond work indexes (CW_i, BW_i, RW_i), to better define rocks hardness throughout the deposits;
- Determine detailed mineralogy of feed;
- Perform grindability test to evaluate variability of the mineralization;
- Perform additional bench scale testwork;
- Perform Pilot Plant investigation;
- Complete waste & tailings characterization (including leaching test and dynamic test);
- Confirm pellet feed characterization;
- Perform a series of balling and pot grate test on representative concentrate samples to define the pellet Fe and silica content as well as the grate factor temperature profile and all the other pellet quality parameters;
- Collect samples for vendor testwork (hydroclassifier, thickeners, filters, magnetic separators);
- Additional metallurgical tests will be necessary, such as: SG, mineral characterization, size distribution, bulk density determination, static thickening, dynamic thickening, pulp rheology, vacuum filtration, and pressure filtration.
- Explore a rougher magnetic separation stage in the ball mill grinding circuit to reject further portion of the non-magnetic gangue;
- Evaluate High Pressure Grinding Roll ("HPGR");
- Evaluate a second stage of crushing with cone crushers as an alternative to SAG mills;
- Perform test work with concentrate (from pilot plant) to define the pumping characteristics of the concentrate slurry and allow sizing of pumps and pipeline complete with a site visit to confirm pipeline routing and topography;
- Perform survey and geotechnical investigation at process plant buildings and infrastructure to provide soil and bedrock bearings elevation, depths and bearing capacities and provide information for more detailed quantity estimations;
- Explore transportation study to determine optimum shipping route and ship size;
- Confirm ice-free shipping season;
- Initiate an ice measurement program;
- Initiate a geotechnical investigation to collect design parameters for dredging and wharf design;
- Initiate bathymetric investigation to confirm bottom contours.

The estimated cost for the next study phase has been estimated and is provided in Table 26.1.

Table 26.1 – Estimated Cost for Next Study Phase

Study Phase	**Cost Estimate ($ M)**
Exploration Drilling Program	3.0
Feasibility Study	7.0
Metallurgical Testwork	2.0
Port	1.5
Geotech and Pit Slope	2.0
Other Site Studies	1.0
Environmental Studies	9.0
Total	**25.5**

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28.0 CERTIFICATES OF QUALIFICATION

CONSENT OF AUTHOR

TO: **Alberta Securities Commission**
Autorité des marchés financiers
British Columbia Securities Commission
Ontario Securities Commission
Securities Commission of Newfoundland and Labrador

I, Daniel M. Gagnon, Eng. do hereby consent to the public filing of the written disclosure of the technical report titled "*NI 43-101 Technical Report Preliminary Economical Assessment of the Duncan Lake Iron Property, James Bay area, Quebec-Canada*" dated May 6th, 2013 with effective date of March 22nd, 2013, for the public filing by Century Iron Mines Corporation.

I also certify that I have read the written disclosure being filed and I do not have any reason to believe that there are any misrepresentations in the information derived from the technical report.

Dated this 6th day of May, 2013.

Daniel M. Gagnon, Eng.

CONSENT OF AUTHOR

TO: **Alberta Securities Commission**
Autorité des marchés financiers
British Columbia Securities Commission
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I, Yves A. Buro, Eng. do hereby consent to the public filing of the written disclosure of the technical report titled "*NI 43-101 Technical Report Preliminary Economical Assessment of the Duncan Lake Iron Property, James Bay area, Quebec-Canada*" dated May 6th, 2013 with effective date of March 22nd, 2013, for the public filing by Century Iron Mines Corporation.

I also certify that I have read the written disclosure being filed and I do not have any reason to believe that there are any misrepresentations in the information derived from the technical report.

Dated this 6th day of May, 2013.

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Yves A. Buro, Eng.

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I, Schadrac Ibrango, P.Geo., Ph.D. do hereby consent to the public filing of the written disclosure of the technical report titled "*NI 43-101 Technical Report Preliminary Economical Assessment of the Duncan Lake Iron Property, James Bay area, Quebec-Canada*" dated May 6th, 2013 with effective date of March 22nd, 2013, for the public filing by Century Iron Mines Corporation.

I also certify that I have read the written disclosure being filed and I do not have any reason to believe that there are any misrepresentations in the information derived from the technical report.

Dated this 6th day of May, 2013

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I, Stéphane Rivard, Eng. do hereby consent to the public filing of the written disclosure of the technical report titled "*NI 43-101 Technical Report Preliminary Economical Assessment of the Duncan Lake Iron Property, James Bay area, Quebec-Canada*" dated May 6th, 2013 with effective date of March 22nd, 2013, for the public filing by Century Iron Mines Corporation.

I also certify that I have read the written disclosure being filed and I do not have any reason to believe that there are any misrepresentations in the information derived from the technical report.

Dated this 6th day of May, 2013.

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I, Charles H. Cauchon, Eng. do hereby consent to the public filing of the written disclosure of the technical report titled "*NI 43-101 Technical Report Preliminary Economical Assessment of the Duncan Lake Iron Property, James Bay area, Quebec-Canada*" dated May 6th, 2013 with effective date of March 22nd, 2013, for the public filing by Century Iron Mines Corporation.

I also certify that I have read the written disclosure being filed and I do not have any reason to believe that there are any misrepresentations in the information derived from the technical report.

Dated this 6th day of May, 2013.

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I, Daniel Houde, Eng. do hereby consent to the public filing of the written disclosure of the technical report titled "*NI 43-101 Technical Report Preliminary Economical Assessment of the Duncan Lake Iron Property, James Bay area, Quebec-Canada*" dated May 6th, 2013 with effective date of March 22nd, 2013, for the public filing by Century Iron Mines Corporation.

I also certify that I have read the written disclosure being filed and I do not have any reason to believe that there are any misrepresentations in the information derived from the technical report.

Dated this 6th day of May, 2013.

Daniel Houde, Eng.

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I, Mary Jean Buchanan, Eng., M.Env. do hereby consent to the public filing of the written disclosure of the technical report titled "*NI 43-101 Technical Report Preliminary Economical Assessment of the Duncan Lake Iron Property, James Bay area, Quebec-Canada*" dated May 6th, 2013 with effective date of March 22nd, 2013, for the public filing by Century Iron Mines Corporation.

I also certify that I have read the written disclosure being filed and I do not have any reason to believe that there are any misrepresentations in the information derived from the technical report.

Dated this 6th day of May, 2013.

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I, Michel L. Bilodeau, Eng., M.Sc. (App.), Ph.D. do hereby consent to the public filing of the written disclosure of the technical report titled "*NI 43-101 Technical Report Preliminary Economical Assessment of the Duncan Lake Iron Property, James Bay area, Quebec-Canada*" dated May 6th, 2013 with effective date of March 22nd, 2013, for the public filing by Century Iron Mines Corporation.

I also certify that I have read the written disclosure being filed and I do not have any reason to believe that there are any misrepresentations in the information derived from the technical report.

Dated this 6th day of May, 2013.

Michel Bilodeau

Michel L. Bilodeau, Eng., M.Sc. (App.), Ph.D.

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Appendix A – Land Holding - Mining Title

Appendix B – QP Samples – Head Assay Results

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September 11, 2012

Date Rec. : 26 June 2012
LR Report : **CA03456-JUN12**
Client Ref : Duncan Lake

CERTIFICATE OF ANALYSIS

Final Report - Revised

Sample ID	SiO2 %	Al2O3 %	Fe2O3 %	MgO %	CaO %	Na2O %	K2O %	TiO2 %	P2O5 %
1: 82810611	52.7	2.38	35.9	2.35	2.27	0.23	0.93	0.08	0.21
2: 82810400	42.7	0.52	53.8	1.31	0.96	0.60	0.45	0.02	0.06
3: 82810411	41.0	0.27	56.0	1.25	1.14	0.49	0.24	< 0.01	0.07
4: 82810423	43.8	0.27	53.9	1.46	1.07	0.66	0.16	< 0.01	0.08
5: 82810438	42.5	0.42	54.0	1.51	1.36	0.71	0.38	0.01	0.09
6: 82810588	50.3	3.45	36.0	3.24	2.63	0.40	1.67	0.16	0.17
7: 82810593	46.9	3.74	41.4	2.64	2.55	1.49	1.08	0.18	0.16
8: 82810603	49.1	5.64	30.9	3.13	3.87	1.29	1.87	0.21	0.13
9: 82810814	---nsr	---nsr	---nsr	---nsr	---nsr	---nsr	---nsr	---nsr	---nsr
10: 441589	---nsr	---nsr	---nsr	---nsr	---nsr	---nsr	---nsr	---nsr	---nsr
11: 82810386	46.3	1.01	50.9	1.54	0.97	0.13	0.68	0.04	0.04
12: 82813359	53.0	2.90	39.0	1.50	1.36	0.46	1.29	0.09	0.15
13: 443457	48.7	4.28	34.6	4.20	3.48	1.79	0.97	0.24	0.22
14: 443473	46.7	2.20	45.9	1.90	1.47	0.26	1.44	0.08	0.08
15: 443479	47.1	3.04	42.8	2.62	2.16	0.48	1.54	0.14	0.15

Sample ID	MnO %	Cr2O3 %	V2O5 %	LOI %	Sum %	S %	Feed g	Mag g	Non Mag g
1: 82810611	0.07	0.02	< 0.01	2.12	99.2	---	20.06	1.14	16.77
2: 82810400	0.02	0.02	< 0.01	-1.02	99.5	---	20.06	10.01	8.50
3: 82810411	< 0.01	0.01	< 0.01	-0.79	99.7	---	25.04	10.33	7.92
4: 82810423	0.02	0.01	< 0.01	-0.83	100.6	---	20.07	9.72	8.68
5: 82810438	0.04	0.01	< 0.01	-0.95	100.1	---	20.06	9.78	8.40
6: 82810588	0.05	0.04	< 0.01	1.72	99.9	---	20.03	5.12	12.43
7: 82810593	0.04	0.03	0.01	0.34	100.6	---	20.04	7.24	10.54
8: 82810603	0.07	0.04	0.01	2.82	99.0	---	20.01	3.45	13.99
9: 82810814	---nsr	---nsr	---nsr	---nsr	---nsr	---	NS	NS	NS
10: 441589	---nsr	---nsr	---nsr	---nsr	---nsr	---	NS	NS	NS
11: 82810386	0.02	0.01	< 0.01	-0.76	100.9	---	20.07	9.40	8.26
12: 82813359	0.04	0.02	< 0.01	0.36	100.2	0.49	20.08	6.76	10.66
13: 443457	0.06	0.05	0.01	0.85	99.4	---	20.03	5.84	10.27
14: 443473	0.04	0.02	< 0.01	-0.29	99.9	---	20.05	8.18	9.20
15: 443479	0.04	0.06	< 0.01	0.19	100.3	---	20.04	7.36	10.05

SGS Canada Inc.
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Sample ID	SiO2 %	Al2O3 %	Fe2O3 %	MgO %	CaO %	Na2O %	K2O %	TiO2 %	P2O5 %
16: 445944	47.4	5.29	35.1	3.34	3.21	1.18	1.68	0.26	0.15
17: 445951	46.7	2.68	44.8	1.92	2.06	0.73	1.00	0.11	0.13
18: 445961	53.2	1.74	38.5	2.39	1.84	0.18	0.65	0.08	0.19
19: 445968	47.2	2.18	44.3	1.95	2.60	0.57	0.90	0.11	0.16
20: 445978	44.9	2.71	45.3	2.38	2.35	0.60	1.45	0.14	0.11
21: 82813136	53.2	5.53	33.8	1.88	1.63	1.07	1.99	0.20	0.20
22: 82813160	51.9	3.45	38.6	1.93	1.70	0.78	1.85	0.12	0.14
23: 82813166	53.0	3.73	34.1	2.03	1.83	0.20	2.41	0.11	0.11
24: 82813173	51.9	2.67	38.9	2.05	1.76	0.44	1.55	0.09	0.14
25: 82813237	53.0	2.20	40.3	1.57	1.20	0.15	1.48	0.06	0.16
26: 82813256	50.9	3.45	36.2	3.06	2.42	0.99	1.31	0.14	0.16
27: 82813262	53.1	4.79	32.5	2.68	1.66	0.89	2.05	0.15	0.17
28: 82813267	53.7	5.09	31.6	2.24	1.87	0.76	2.27	0.17	0.19
29: 82813273	50.6	4.64	35.7	1.99	2.33	1.28	1.54	0.19	0.20
30: 82813277	54.6	5.44	31.5	1.92	1.74	1.09	1.92	0.18	0.18
31: 82813344	49.3	1.26	45.5	1.72	1.18	0.11	0.84	0.04	0.15
32: 82813349	49.3	2.55	42.2	2.38	1.73	0.53	1.08	0.12	0.17
33: 82813356	50.4	2.61	40.5	2.13	1.67	0.45	1.21	0.11	0.16
34: 82813371	52.2	5.35	34.8	1.58	1.32	1.54	1.58	0.17	0.17
35: 445855	50.0	3.59	41.7	1.50	0.72	0.03	1.90	0.14	0.17
36: 445857	47.2	2.02	47.5	1.57	1.42	0.12	0.65	0.14	0.10
37: 445861	56.6	2.06	37.1	2.13	1.46	0.32	0.82	0.07	0.09
38: 445864	46.6	2.45	44.5	2.26	2.04	0.55	0.66	0.14	0.18
39: 445868	49.9	2.33	42.0	2.53	1.79	0.43	0.89	0.10	0.18

Sample ID	MnO %	Cr2O3 %	V2O5 %	LOI %	Sum %	S %	Feed g	Mag g	Non Mag g
16: 445944	0.07	0.03	0.02	2.12	99.9	---	20.04	4.94	12.80
17: 445951	0.03	0.02	< 0.01	0.31	100.5	---	20.07	8.06	9.60
18: 445961	0.04	0.02	< 0.01	1.45	100.3	---	20.02	4.50	13.20
19: 445968	0.03	0.02	< 0.01	-0.02	100.0	---	20.09	7.98	9.36
20: 445978	0.03	0.02	< 0.01	-0.11	99.9	---	20.09	8.32	9.89
21: 82813136	0.04	0.02	< 0.01	0.62	100.2	0.58	20.12	4.10	12.63
22: 82813160	0.04	0.01	0.01	-0.14	100.4	---	20.11	6.58	11.98
23: 82813166	0.10	0.02	< 0.01	1.56	99.2	---	20.14	3.99	14.26
24: 82813173	0.04	0.02	< 0.01	0.32	99.9	---	20.03	6.70	9.84
25: 82813237	0.03	0.02	< 0.01	-0.15	100.1	---	20.03	6.75	11.22
26: 82813256	0.04	0.02	< 0.01	0.67	99.4	0.23	20.04	6.13	11.39
27: 82813262	0.04	0.03	0.01	1.28	99.4	---	20.03	4.93	13.05
28: 82813267	0.04	0.03	< 0.01	1.49	99.4	---	20.08	4.50	14.09
29: 82813273	0.04	0.02	< 0.01	1.53	100.1	---	20.06	5.55	13.05
30: 82813277	0.04	0.02	< 0.01	1.78	100.4	---	20.05	4.63	14.24
31: 82813344	0.04	0.02	< 0.01	-0.02	100.2	---	20.08	8.00	10.64
32: 82813349	0.03	0.03	< 0.01	0.44	100.5	---	20.08	7.18	11.87
33: 82813356	0.03	0.02	< 0.01	0.12	99.4	0.39	20.04	7.20	11.80
34: 82813371	0.03	0.02	0.01	1.13	99.9	---	20.03	5.81	12.63
35: 445855	0.05	0.04	< 0.01	0.68	100.5	---	20.09	6.02	12.77
36: 445857	0.04	0.02	< 0.01	-0.31	100.4	---	20.02	8.10	11.01
37: 445861	0.04	0.03	< 0.01	-0.18	100.6	---	20.04	6.63	12.27
38: 445864	0.09	0.02	< 0.01	0.41	99.9	---	20.08	7.94	11.07
39: 445868	0.03	0.03	< 0.01	0.26	100.5	---	20.02	7.91	10.22

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Sample ID	SiO2 %	Al2O3 %	Fe2O3 %	MgO %	CaO %	Na2O %	K2O %	TiO2 %	P2O5 %
40: 445881	51.3	1.82	39.7	1.35	0.48	0.16	3.65	0.07	0.09
41: 445884	55.8	3.40	33.3	1.67	1.35	0.07	3.06	0.12	0.10
42: 445889	44.4	0.79	52.5	1.63	1.01	0.07	0.40	0.01	0.18
43: 445894	44.0	1.12	48.0	1.85	2.86	0.06	0.13	0.06	0.20
44: 445896	48.7	1.74	47.2	1.12	1.26	0.30	0.71	0.05	0.15
45: 44998	35.6	0.05	55.4	1.93	2.36	0.03	0.02	< 0.01	0.01
46: 44999	43.1	0.05	43.2	1.60	4.42	0.03	0.01	< 0.01	0.02
47-DUP: 445968	47.3	2.20	44.5	1.94	2.62	0.57	0.90	0.10	0.16
48-DUP: 445864	46.8	2.50	44.8	2.30	2.04	0.57	0.67	0.15	0.18

Sample ID	MnO %	Cr2O3 %	V2O5 %	LOI %	Sum %	S %	Feed g	Mag g	Non Mag g
40: 445881	0.04	< 0.01	< 0.01	0.53	99.2	---	20.08	2.93	15.34
41: 445884	0.06	0.03	< 0.01	0.51	99.5	---	20.06	3.40	14.57
42: 445889	0.02	0.02	< 0.01	-0.87	100.2	---	20.04	10.76	8.25
43: 445894	0.31	0.02	< 0.01	1.92	100.5	---	20.06	7.97	10.53
44: 445896	0.03	0.02	< 0.01	-0.90	100.3	0.20	20.05	9.41	9.15
45: 44998	0.48	< 0.01	< 0.01	3.70	99.6	0.01	20.04	3.12	15.13
46: 44999	0.73	< 0.01	< 0.01	6.67	99.9	0.01	20.02	5.08	14.12
47-DUP: 445968	0.03	0.02	< 0.01	-0.04	100.4	---	20.05	8.04	9.97
48-DUP: 445864	0.09	0.02	< 0.01	0.49	100.6	---	20.05	7.93	10.94

Control Quality Analysis - Not suitable for commercial exchange

This report supersedes the certificate CA03456-JUN issued 06-Sept-12 revised to show adjustment to sample ID's.

April Rice
Project Coordinator

Email: ken.lam@centuryiron.com; allan.gan@centuryiron.com; ricky.chan@centuryiron.com; claudebritt@hotmail.com; ghislain.arel@centuryiron.com; hubert.vallee@centuryiron.com; zhihuan.wan@centuryiron.com; matthew.chong@centuryiron.com; sandy@chim.ca; mgirard@soutex.ca

Appendix C – Port and Shipping Study by Portha Inc.

Portha Inc.

NEW PORT INFRASTRUCTURE AND SHIPPING ALTERNATIVES

Chisasibi, Québec

Duncan Lake Project

Century Iron Mines Corporation

March 2013

TABLE OF CONTENTS

SUMMARY

Portha Inc. was mandated to provide an independent shipping and port infrastructure study for Century Iron Mines Corporation's Duncan Lake Iron Property. The shipping analysis component in this report evaluated different transportation routes, ship sizes and different export seasons and determined the best location for the port is Stromness Island. It also illustrates that marine transportation costs are much less than the ground transportation costs. The demurrage calculations component of this report quantified the extent of the port infrastructure; this being two berths and two high capacity shiploaders. The overall findings indicate that, if a shortened export season is acceptable, the project has an advantage compared to certain other active iron mines. This cost advantage must be weighed against other factors such as tramp or time charter restrictions and premiums, stockpiling, inventory costs and the cost of additional port infrastructure required for exporting larger quantities of ore in a shortened shipping season. Other options that are more flexible and/or accommodate a longer shipping season with better time charter rates should also be considered for further analysis.

1 INTRODUCTION

Present Situation

A new mine at Duncan Lake near Chisasibi, Québec on the eastern shore of James Bay, presently being planned by Canadian Century Iron Ore Corporation (Century), will require port facilities designed to export approximately 12 million tonnes of iron ore per year. Since the project is still at a preliminary stage, there is no soil information for the area and the only underwater contours available are from current hydrographic charts. This study therefore is based on many assumptions regarding the physical data at the site. However, at this stage it is more important to determine the best shipping route and season, optimum ship size and loading rates.

Two possible locations for construction of port infrastructure have been studied previously by Century. Both are in the immediate vicinity of the town of Chisasibi. One location is near Wastikun Island and the other near Stromness Island. Both are exposed to a relatively long fetch from the west northwest and southwest. The presence of small rock outcrops and small islands would damper the waves to a certain degree in some directions. Wastikun Island was a definite first choice by many because of favourable on-shore features, access and ease of construction. This site is also suitable for some of the route options based on small shuttle vessels routed through Moosonee or Churchill. However, for other route and shipping options, a site near Stromness Island offers deeper water and a better approach channel according to the existing navigation charts. The fact that the two sites offer different advantages reinforced the need for a transportation study to determine the optimum route and ship size. It was obvious that the transportation component of the study should include all the routing possibilities in order to decide between the Wastikun Island site and the Stromness Island site. Also, the different costs for each shipping season were evaluated.

The topography in the surrounding area indicates that the soil types vary from rock outcrops to sand with areas of bog. From visual observations it was assumed that the bottom of the bay in the area designated for the port infrastructure consists of a rock surface with very little overburden. The type of port infrastructure was chosen accordingly.

Scope of Work

The purpose of this analysis was to evaluate the costs of the port infrastructure as a complement to a PEA study for the mine. Indirectly the analysis included an estimate of shipping costs based on calculated values as part of the transportation component to establish the port infrastructure requirements, the size of the ships, berth occupancy and the location for the new facilities that could accommodate the most favourable ship size.

The site near Wastikun Island was suggested first in a 2010 study by Anna Klimek. The study mentioned Stromness Island as a second option. The cost of shipping from Wastikun Island with smaller ships determined that the marine advantages at Stromness Island seem to outweigh the extra cost for onshore facilities at the site. Also, the approach channel to Stromness Island requires less turns, is deeper and offers a better uniform draft for two berths according to the chart data presently available.

Marine transportation in the arctic region is also in evolution. The recommendations and standards for Ice class and Arctic class vessels are expected to be modified in the near future. This study used the existing International Marine recommendations and guidelines in addition to the Canadian guidelines for the Port of Churchill.

This analysis is extensive and provides as much information as possible for future decisions. It is comprehensive enough to cover almost all future scenarios. At such an early stage project changes or modifications to the initial criteria are common, especially in the marine environment. In addition, the new port infrastructure must be able to accommodate the necessary inward supply for the mine including bentonite, fuel and general cargo since there are no other port facilities in the area.

All evaluations and recommendations are presented as best of knowledge evaluations for comparing different options on the same level and must be considered as advisory only. The transportation costs are based on expected future prices for fuel and ship charter and on approximations for ship replacement costs, in particular the Ice and Polar class vessels.

2 SITE PECULIARITIES

Geotechnical Assumptions

Since there is no geotechnical data and the chart depths are incomplete, this evaluation is preliminary but it can be considered a reasonable assessment for the PEA study. In order to estimate construction costs it was assumed that no dredging was necessary and that the entire area where all of the port infrastructure will be located was rock. It was also assumed that most of the rock fill for the access dykes and for the breakwater will be available from the mine. Additional borrow quantities can be obtained from rock outcrops nearby observed during the visit at Wastikun Island and surrounding area.

Marine Environment

Considering how critical the loading operation is and how serious any damage to the conveyor system could be, the port infrastructure at this location will have to be protected from major waves and storms. A gravity full face type structure is recommended as opposed to an open pile or dolphin type structure.

As shown on preliminary drawings in Appendix B, the solution is to locate the berths on the inside face of a wharf and incorporate a breakwater on the outside. The design would integrate both the wharf and the breakwater plus serve as an access structure for the conveyors, pipelines and truck traffic. The impact of the waves will be minimized but there could be some lost time during extreme storms when it would be too risky for ships to navigate to their berth from the approach channel or once berthed, to use the shiploaders.

The area is not in a high earthquake risk zone which favors a gravity full face type of construction such as the cell design. This will need to be reassessed when more data is available. Considering the location, availability of materials, construction challenges and other features, the combination of cells and rock fill is the most practical design for this project.

The northern location and shipping through the Hudson Strait dictates specific requirements for ships depending on the shipping season chosen. The Port of Churchill, Manitoba is a good example of what to expect at Chisasibi, Québec. The Port of Churchill shipping season extends from mid-July to the beginning of November for 1C Ice class vessels entering the Hudson Strait. The shipping season can be extended approximately twenty days if the vessels used are 1A Ice class.

3 TRANSPORTATION OPTIONS

Ocean transportation costs in normal open water are inversely proportional to the size of the ship. As shown in Figures 1 and 2, both ship building costs and transportation costs decrease considerably as ship size increases.

If the port infrastructure and corresponding deep channels are possible, the use of larger ships will reduce the marine freight rate significantly. For example, the difference in cost between a 75,000 dwt (Panamax) Wastikun Island size ship and a 200,000 dwt (Capesize) Stromness Island size ship for the Labrador terminal option is plus or minus $4.00 per metric tonne based on a four month shipping season. The difference is even greater for a longer shipping season.

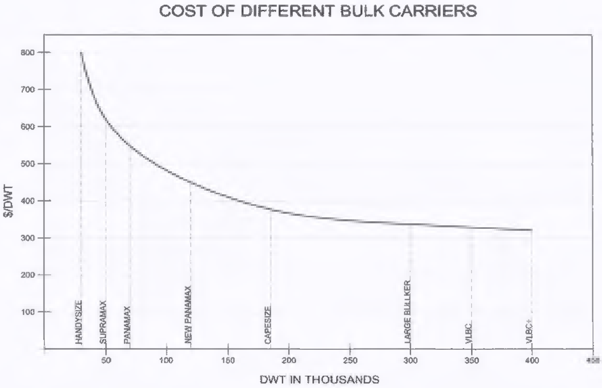

COST OF DIFFERENT BULK CARRIERS
$/DWT
800
700
600
500
400
300
200
100
HANDYSIZE
SUPRAMAX
PANAMAX
NEW PANAMAX
CAPESIZE
LARGE BULKER
VLBC
VLBC+
50
100
150
200
250
300
350
400
DWT IN THOUSANDS

Figure 1

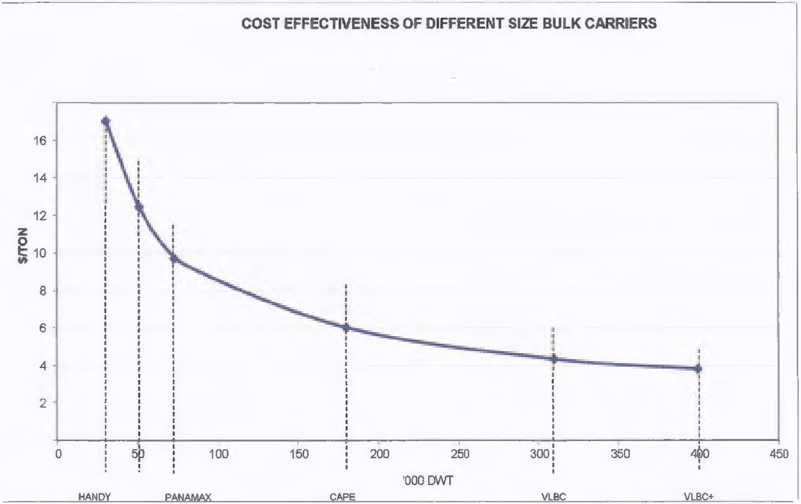
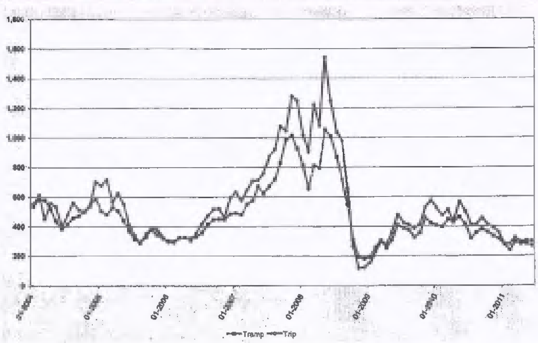


Figure 2 (Typical Unit Cost)

Many widely varying factors influence shipping costs. These include piracy, transfer links, the presence of ice and the fluctuations in the shipping charter index. The UNCTAD secretariat 2011 Review of Maritime Transport reported that freight rates for Capesize vessels on the Far-East-Europe route varied from $57,587 to $17,358 per day between January 2010 and January 2011. It was considered that the best method to establish the projected costs for this project was to prepare a numerical model based on projected values and compare the results with various sources such as the UNCTAD monograph. This monograph was produced by the Institute of Shipping Economics and Statistics and covers the period between 2004 and 2011 (see Figure 3). The model is based on values not on current quotes. The different methodologies used include direct routing, ship to ship transfer and ship to land terminal transfers that included both rail/ship and ship/ship options. Values for daily ship charter costs were calculated for crew, ownership and fuel costs separately for each classification of ship using a projected Dry Cargo Freight Index for the near future and the Ice class requirements for the Port of Churchill.

The results of all the options vary from $34.59 to $118.83 and are shown in Appendix A. The results include, where applicable, published operational costs for the ship to ship or terminal transfers (ship loading) but do not include any capital costs to construct transfer terminals, the ship to ship intermediate barges or the cost to build new Polar /Ice class shuttle ships.

Figure 3.2. Dry cargo freight indices, 2004–2011



Source: UNCTAD secretariat, based on various issues of *Shipping Statistics* and *Market Review*, produced by the Institute of Shipping Economics and Logistics.

Figure 3

Note: Dry cargo tramp time charter refers to a charter for a time period and trip charter for a specific voyage.

The numerical model is based on the two flow diagrams shown on Figures 4 and 5 which outline the sequence of operations. The model was restructured by adding the "seasons to be studied" category to the flow chart after the initial model indicated the cost of shipping all year was not economically viable.

ROUTE COST EFFECTIVENESS

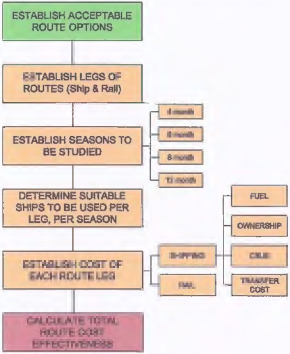


Figure 4

Figure 5

To calculate the values for each option, the variables are:

- period of the year divided into summer, winter, extended summer and extended winter
- distance of each travel section of each leg
- the ship classification for Ice class to Polar class
- premium for Ice or Polar class over non Ice basic charter
- ship acquisition costs
- the ship speed for each travel section for each season,
- calculated ship power requirements
- calculated fuel consumption during travel fully loaded, at ballast and at port,
- ship ownership value including insurance, maintenance etc.
- fixed time for docking and leaving the berth
- time for loading or transfers
- daily charter at port, fully loaded and at ballast
- crew costs
- navigation requirements and regulations
- canal fees and tariffs

Ice Navigation

The major factor affecting decisions determining transportation cost from this remote site is obviously the additional cost of navigating in ice. There is very limited real data on that subject except for ice breakers. Very few large ships have been built to transport large quantities of bulk materials from the north. Due to this lack of real data, it was necessary to estimate values for constructing new ships and estimate the power requirements for different class ships to calculate the approximate fuel costs for different voyage situations.

There are many class designations for navigating in ice each with specific design requirements, equipment and crew. The designations are Ice class or Polar class. Each class is based on a variety of standards issued by different countries and organizations. The main requirements according to the latest IMO guidelines include special requirements for structure, stability, machinery, life-saving devices and fire protection, ship routing, navigation systems and equipment, radio and other communication systems, pollution prevention equipment, liability and safety management systems. Unfortunately, except for the lower rated Ice class ships, the shape of the hull to meet the guidelines is not optimal for long distance travel. In many cases power requirements further increase long distance transportation costs.

All the possible main transportation routes were evaluated with different ship size and methods. Each route was also separated in different sections to account for the difference between arctic water, river ice and the open ocean. An analysis was completed for different shipping seasons (see Appendix A).

It is evident from the results that ice navigation is possible but at a very high cost. Only Polar class ships are recommended in Arctic waters (see Figures 6 & 7). These ships are built with stronger hulls, have larger motors and consume more fuel.

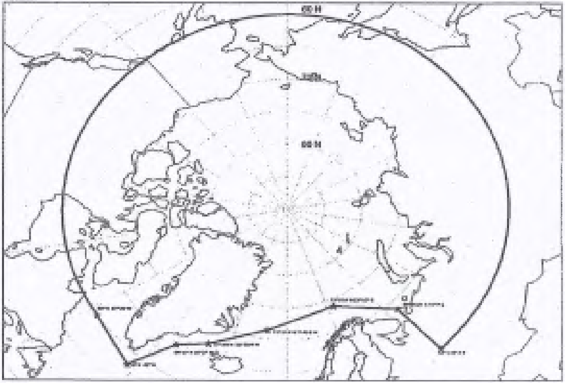
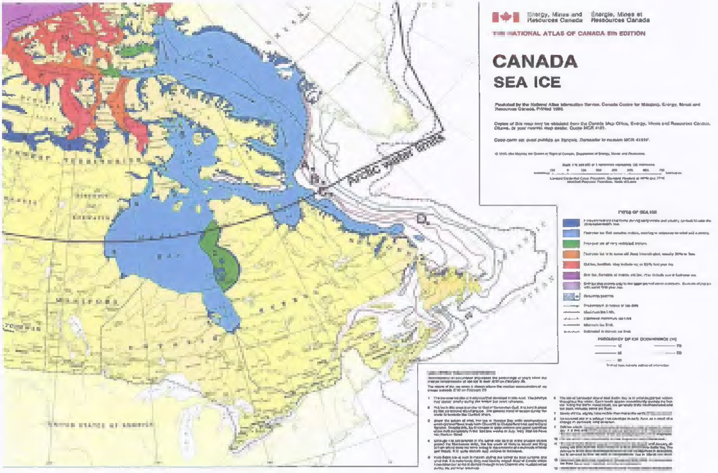


Figure 6

The Arctic zone as defined by the International Marine Organization (IMO)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CANADA
SEA ICE

Figure 7

The results of the transport study component of this analysis are found Appendix A. The results show operating costs and charter rates that are based on approximated values of ship construction cost. If new dedicated ships were to be built for this project or if there were additional terminal construction costs outside the proposed port, it is very probable that some increased capital costs would have to be considered. Potentially some of the terminal costs could be negotiated with the local government authorities. The results in Appendix A confirm the advantage of a shorter shipping season but do not reflect differences between trip and time charters. They are based on what are considered fairly safe assumptions for the four month shipping season for the Port of Churchill. However, some of the restrictions applicable for Arctic waters, as defined in Resolution A.1025(26) 2012, include Hudson Strait area and could apply to both the Port of Churchill and this project in the future. The shipping industry should have time to adjust accordingly. The IMO are guidelines only and enforcement rests with each country. In this case it can be assumed that the same Port of Churchill compliance requirements enforced by Transport Canada would apply. The promoter should seek a definite ruling when the project moves to the next stage.

The numerical model also shows that any option with a rail component is more expensive than all marine options although the Saguenay rail option could offer indirect advantages. Based on the shipping season for the Port of Churchill, Manitoba it appears that the three to four month shipping season could be acceptable for ships with minimal ice classification. If a shorter season is acceptable at the delivery end this option is certainly the most economical for the mine location. A classification upgrade to Polar class 7 ships would be required to extend the shipping season an additional two months and to Polar class 5 ships for an all year shipping season through the Hudson Strait. The cost of upgrading to Polar class 7 ships is only a fraction of the cost of a Polar class 5 upgrade. Upgrading would only be a good solution if it is profitable to extend the shipping season.

The maximum draft at the Wastikun Island site corresponds to a 75,000 dwt ship. The maximum draft at Stromness Island corresponds to a Capesize (200,000 dwt max) vessel. There is a possibility that a Suezmax (240,000 dwt max) might also be able to berth at Stromness Island. The possibility will have to be confirmed once the bottom contours and the geotechnical data are available. When it became apparent that Capesize vessels could berth at the Stromness Island site it opened up the possibility of different options such as direct shipping from Chisasibi to Qingdao. It also became evident that the numerical model had to accommodate different conditions and different shipping seasons and was adjusted on that basis. One of the options that is promising is the Northwest Passage route. The cost estimate for the route could improve with further investigation for larger ships.

4 PORT INFRASTRUCTURE

The type of structure recommended for the wharf is a sheet pile cellular design complete with intermediate arcs (see Appendix B). The access dykes can be built with rock fill that should be readily available in the area. Both the wharf and part of the access dykes can be incorporated on the inside face of the breakwater. A separate breakwater is another option but the cost would increase considerably.

The cell wharf type and rock fill access are recommended for large ice loads and the design can be well integrated with the breakwater. The cell design is a proven construction design for harsh conditions. This type of construction was used by the Port of Prescott, Ontario in 2012 (see Appendix C). Other types of wharf structures would not be as practical at the site. However, certain types of gravity design structures could be an alternative if the assumptions for the supporting soil are not correct. Should this be the case some dredging would probably be required.

The estimated cost (see Figure 8) for the port infrastructure is 200 M$ for the two berths, the breakwater and access dyke. The cost of two 12000 tph shiploaders and rail foundations is estimated at 50 M$. The normal allocation for surveys, studies and engineering, conveyors and other land based installations are in addition.

Estimate of cost for the wharf and access dyke

Duncan Lake Project (PTDL2012-00-03)

Item	Description	Amount	[illegible]	[illegible]	[illegible]	[illegible]
	Mobilization, demobilization and other fixed site costs					
[illegible]	[illegible]	100%	---	100%	1 350 000,00 $	1 350 000,00 $
[illegible]	[illegible]	100%	---	100%	540 000,00 $	540 000,00 $
[illegible]	[illegible]	100%	---	100%	600 000,00 $	600 000,00 $
4	[illegible]	100%	---	100%	600 000,00 $	600 000,00 $
[illegible]	[illegible]	100%	---	100%	120 000,00 $	120 000,00 $
[illegible]	Work area and situation [illegible]	100%	---	100%	720 000,00 $	720 000,00 $
TOTAL						[illegible]
	General Fill					
[illegible]	Cell					
	[illegible]	[illegible]	---	5 400 m³	25,00 $	135 000,00 $
	[illegible]	[illegible]	---	470 000 m³	15,00 $	7 050 000,00 $
	[illegible]	[illegible]	[illegible]	1 243 599 yd³	5,25 $	6 530 000,00 $
8	[illegible]					
	[illegible]	[illegible]	---	14 500 m³	[illegible]	363 000,00 $
	[illegible]	[illegible]	---	1 169 000 m³	[illegible]	17 600 000,00 $
	[illegible]	[illegible]	[illegible]	3 095 919 yd³	[illegible]	16 300 000,00 $
TOTAL						[illegible]
	Circular Cell Construction					
9	[illegible]					
	[illegible]	5480 units	13,25 m²/unit	72 610,00 m²	260,00 $/m²	18 879 000,00 $
	[illegible]	5480 units	13,25 m²/unit	72 610,00 m²	200,00 $/m²	14 522 000,00 $
[illegible]	[illegible]					
	[illegible]	[illegible]	13,25 m²/unit	1 272,00 m²	260,00 $/m²	331 000,00 $
	[illegible]	[illegible]	13,25 m²/unit	1 272,00 m²	200,00 $/m²	255 000,00 $
[illegible]	[illegible]	[illegible]	---	[illegible]	500 000,00 $	500 000,00 $
TOTAL						[illegible]
	Copewall & Accessories					
12	Copewall	940 m	[illegible]	4 700,00 m³	1 000,00 $	4 700 000,00 $
13	Fenders	84 units	---	84 units	40 000,00 $	3 360 000,00 $
14	Bollards	25 units	---	25 units	50 000,00 $	1 250 000,00 $
15	Ladders	14 units	---	14 units	8 500,00 $	119 000,00 $
16	[illegible]	2 300 m	---	2 300 m	400,00 $	920 000,00 $
17	[illegible]	1 700 m	---	1 700 m	600,00 $	1 020 000,00 $
18	[illegible]	6 800 m³	---	6 800 m³	1 000,00 $	6 800 000,00 $
19	[illegible]	11 200 m	---	11 200 m	229,30 $	2 568 160,00 $
20	[illegible]	25 units	26,5 m/unit	663 m	229,30 $	152 000,00 $
[illegible]						[illegible]
	[illegible]					
21	[illegible]	136 000 m²	---	[illegible]	50,00 $	6 800 000,00 $
22	[illegible]	100%	---	[illegible]	$8 584 640,00	8 585 000,00 $
23	[illegible]	100%	---	[illegible]	$69 750 200,00	69 751 000,00 $
24	[illegible]	100%	---	[illegible]	$8 048 100,00	8 049 000,00 $
TOTAL						[illegible]
TOTAL COST						$200 500 000,00

Figure 8

Berth Occupancy

The infrastructure requirements were established as per correlation and experience and verified using the United Nations monograph No. 6 "Measuring and Evaluating Port Performance and Productivity". The monograph was produced by a team of experts with the co-operation of the International Association of Ports and Harbours and was developed to calculate approximate waiting time as a percentage of service time for different berth occupancies and number of berths. Also considered were different types of operations, marine terminals and infrastructures.

Our mathematical model is an approximate method modified to correlate to real terminals and is generally acceptable for planning purposes. It was derived by relating a statistical formula based on the queuing theory formula with Poisson distributed arrivals and exponential service times on the basis of first come, first served, queue discipline within real terminals. After an initial study, based on the queuing theory, a number of further analyses compared the initial pure random numbers with practical experience from ports and resulted in adjustments using Erland distributions to correspond with actual conditions corresponding to different types of terminals. The method is used to approximate waiting time under different conditions and calculate costs in order to balance port requirements versus demurrage. Interpretation of this method, combined with previous experience, in-house statistics from bulk terminals and the results of the calculations for a four month loading season produced the following findings:

NUMBER OF BERTHS	TONNAGE	CAPACITY SHIPLOADER tph	NUMBER SHIPLOADERS	% BERTH OCCUPANCY	WAIT TIME hrs	EXPECTED DEMURRAGE $
1	8 000 000	8000	1	91%	10349	14 425 579 $
1	8 000 000	12000	1	70%	1741	2 426 598 $
1	8 000 000	8000	2	59%	866	1 207 572 $
1	10 000 000	10000	2	66%	1181	1 645 900 $
1	10 000 000	12000	2	61%	825	1 149 661 $
1	12 000 000	12000	2	80%	2863	3 990 423 $
2	12 000 000	8000	2	48%	538	749 441 $
2	14 000 000	8000	2	55%	850	1 184 100 $
2	16 000 000	8000	2	62%	1428	1 989 769 $
2	16 000 000	10000	2	56%	857	1 194 419 $
2	16 000 000	12000	2	52%	627	873 564 $
2	16 000 000	10000	3	47%	453	631 674 $
2	16 000 000	8000	3	52%	627	873 564 $

Based on the numbers which are applicable to of Capesize and Suezmax vessels, two berths are definitely recommended for a tonnage of 16 million tpy. Two high capacity 12,000 tph shiploader units would limit demurrage to less than 1 M$ per year based on charter contracts with no lay day allocations.

For more accurate demurrage calculations, computer simulations are generally considered as more accurate than the above but any simulation is still based on approximations and in many cases the results are not much different. For general planning purposes and considering all the other assumptions at this stage of the project, Portha Inc. found the above method acceptable as a proven planning tool if the interpretations of the projected conditions for the new terminal are accurate.

5 RECOMMENDATIONS

The construction of a double berth port infrastructure with integrated breakwater and two large capacity ship loaders is recommended at Stromness Island for the expected throughput that is presently planned for the mine.

The proposed mine marine operation has an advantage over most other active mines due to the fact that the cost of marine transportation is much less than the cost of ground transportation which most mines utilize. Assumptions made for the transportation component of this study should be verified through a risk analysis for the next phase. The analysis should include possible improvements to the assumptions made for the Northwest Passage option.

The impact of shipping during a shortened season with Ice class 1C vessels, under tramp time charter restrictions and premiums versus the cost of stockpiling and inventory should be investigated further. These additional costs should be compared to the benefit of shipping during a longer period with time charter models.

The huge difference between the highest and lowest cost options was not evident initially. The completed analysis revealed that a shortened shipping season and storing product for an eight month period represented a huge saving in shipping costs. If delivery is acceptable on a shortened shipping season and if shipping on a spot charter basis is possible for full year production, then it is the best option and a good base case scenario. If the present charter prices are maintained then the prices for the shorter shipping option could improve further.

If delivery on a shortened shipping season is unacceptable or if spot charter rates are higher than time charter rates, then storage away from the area where Ice class ships are required might be advantageous. The possibility of a new terminal in Labrador would extend delivery to six months and should be considered in particular if there are other potential users to share the terminal. A new terminal in an ice free area in the Maritime Provinces could extend shipping to a full year.

Bottom contours, geotechnical and other studies are required for the next phase of analysis. The geotechnical study should be coordinated with the bottom sampling requirements normally required for future environmental reviews.

APPENDIX A

MAIN ROUTES AND SHIPPING COST EVALUATION ALL SHIPPING SEASON

Type of Routing	Main Landmark	First Ship	Second Ship	DWT	Total Distance km	4 Month Season		6 Month Season		8 m Month Season		12 Month Season	
						est. cost	Ice Class	est. cost	change	est. cost	change	est. cost	change
Direct	Hudson Strait	Suezmax		220 000	24 239	$34,59	1C	N/A	N/A				
Direct	Hudson Strait	Capesize		185 000	24 239	$36,94	1C	N/A	N/A				
Terminal transfer	Labrador	Stromness P7	VLBC	350 000	29 850	$37,56	Polar 7	$37,83		N/A	N/A	N/A	
Terminal transfer	Nova Scotia	Stromness P7	VLBC	350 000	30 239	$38,46	Polar 7	$38,72		$44,14	Polar 5	$53,04	
Direct	NW Passage	Stromness P5	Stromness P5	195 139	15 049	$38,62	Polar 5	$39,25		$82,26	Polar 2	$99,54	
Ship to ship	Greenland	Stromness P7	Suezmax 1C	218 929	24 689	$39,82	Polar 7	N/A	N/A	N/A		N/A	
Terminal transfer	Sept-Îles	Stromness P7	VLBC	350 000	31 320	$40,47	Polar 7	$40,74		$46,84	Polar 5	$60,28	1A
Terminal transfer	St John NB	Stromness P7	VLBC	350 000	32 416	$42,07	Polar 7	$42,34		$49,16	Polar 5	$58,06	
Ship to ship	Iceland	Stromness P7	VLBC	350 000	31 420	$42,12	Polar 7	$42,68		$48,72	Polar 5	$57,92	
Ship to ship	Sept-Îles	Stromness P7	VLBC	350 000	31 320	$42,12	Polar 7	$42,40		$48,72	Polar 5	$62,14	1A
Ship to ship	St John NB	Stromness P7	VLBC	350 000	32 416	$43,84	Polar 7	$44,12		$51,00	Polar 5	$60,26	
Rail-Marine route	Saguenay	Rail	Suezmax	220 000	23 907	$54,97	N/A	$54,97		$54,97		$57,72	1A
Marine-rail-marine	Moosonee	Laker size	Suezmax	220 000	24 822	$76,97	Custom	$77,08		$77,94		$84,34	
Marine-rail-marine	Churchill	Laker size	Suezmax	185 000	13 250	$102,01	Custom	$102,41		$105,56		$118,83	

MAIN ROUTES AND SHIPPING COST EVALUATION BASED ON A 4 MONTH SHIPPING SEASON

4 Month Shipping Season

				FIRST LEG							SECOND LEG					
Type of Routing	Main Landmark	Total Distance km	Estimated Cost	Ship First Leg	Class	DWT	Route	km to Intermediate Point	km to Transfer Station	Cost First Leg	Ship Second Leg	DWT	Route	km to Qindao	Cost Final Leg	Seasonal Restriction
Direct	Hudson Strait	24 239	$34,59	Suezmax	Ice class 1C	220 000	point C	2 250		$5,64	Suezmax	220 000	Suez	21 989	$25,35	SUMMER ONLY
Direct	Hudson Strait	24 239	$36,94	Capesize	Ice class 1C	185 000	point C	2 250		$5,82	Capesize	185 000	Suez	21 989	$28,52	SUMMER ONLY
Terminal transfer	Labrador	29 850	$41,54	Wastikun P7	Polar 7	78 785	point C	2250	600	$10,46	VLBC	350 000	NL - South Africa	27000	$26,78	summer months
Terminal transfer	Labrador	29 850	$37,56	Stromness P7	Polar 7	196 658	point C	2250	600	$6,48	VLBC	350 000	NL - South Africa	27000	$26,78	summer months
Terminal transfer	Nova Scotia	30 239	$38,46	Stromness P7	Polar 7	196 658	point C	2250	1250	$7,63	VLBC	350 000	NS - South Africa	26739	$26,53	summer months
Direct	NW Passage	15 049	$38,62	Stromness P5	Polar 5	195 139	Hudson Strait	8 238		$21,45	Stromness P5	195 139	Bearing Strait	6 811	$17,17	summer months
Ship to ship	Greenland	24 689	$39,82	Stromness P7	Polar 7	194 658	point A	2 225	475	$6,43	Suezmax 1C	218 929	Nuuk -Suez	21 989	$26,40	summer months
Terminal transfer	Sept-Îles	31 320	$40,47	Stromness P7	Polar 7	196 658	Points C and D	2250	1925	$9,25	VLBC	350 000	SI - South Africa	27145	$26,92	summer months
Terminal transfer	St John NB	32 416	$42,07	Stromness P7	Polar 7	196 658	point C	2250	3 125	$10,95	VLBC	350 000	Reykjavik -Africa	27 041	$26,82	summer months
Ship to ship	Iceland	31 420	$42,12	Stromness P7	Polar 7	194 658	point B	2200	2 330	$9,84	VLBC	350 000	St John - South Africa	26 890	$27,28	summer months
Ship to ship	Sept-Îles	31 320	$42,12	Stromness P7	Polar 7	194 658	Points C and D	2250	1925	$9,60	VLBC	350 000	SI - South Africa	27145	$27,53	summer months
Ship to ship	St John NB	32 416	$43,84	Stromness P7	Polar 7	194 658	point C	2250	3125	$11,42	VLBC	350 000	St John - South Africa	27041	$27,43	summer months
Rail-Marine route	Saguenay	23 907	$54,97	Rail			New rail line		900	$22,24	Suezmax	220 000	Saguenay - Suez	23007	$26,43	summer months
Marine-rail-marine	Moosonee	24 822	$76,97	Laker size	Polar class 7	25 000	existing rail lines	316	1499	$39,94	Suezmax	220 000	Saguenay - Suez	23007	$26,43	summer months
Marine-rail-marine	Churchill	13 250	$102,01	Laker size	Polar class 7	25 000	existing rail lines	1150	3497	$83,24	Capesize	185 000	Prince-Rupert	8 603	$10,17	summer months

NOTE: Wastikun P7 shown for comparison purposes only

MAIN ROUTES AND SHIPPING COST EVALUATION BASED ON A 6 MONTH SHIPPING SEASON

6 month shipping season

				FIRST LEG							SECOND LEG					
Type of Routing	Main Landmark	Total Distance km	Estimated Cost	Ship First Leg	Class	DWT	Route	km to Intermediate Point	km to Transfer Station	Cost First Leg	Ship Second Leg	DWT	Route	km to Qindao	Cost Final Leg	Seasonal Restriction
Terminal transfer	Labrador	29 850	$37,83	Stromness P7	Polar 7	196 658	point C	2250	600	$6,75	VLBC	350 000	NL - South Africa	27000	$26,78	6 month season (maximum practical)
Terminal transfer	Nova Scotia	30 239	$38,72	Stromness P7	Polar 7	196 658	point C	2250	1250	$7,90	VLBC	350 000	NS - South Africa	26739	$26,53	6 month season
Direct	NW Passage	15 049	$39,25	Stromness P5	Polar 5	195 139	Hudson Strait	8 238		$22,08	Stromness P5	195 139	Hudson Strait	6 811	$17,17	6 month season
Terminal transfer	Sept-Îles	31 320	$40,74	Stromness P7	Polar 7	196 658	Points C and D	2250	1925	$9,52	VLBC	350 000	SI - South Africa	27145	$26,92	6 month season
Terminal transfer	St John NB	32 416	$42,34	Stromness P7	Polar 7	196 658	point C	2250	3 125	$11,22	VLBC	350 000	St John - South Africa	27 041	$26,82	6 month season
Ship to ship	Iceland	31 420	$42,68	Stromness P7	Polar 7	194 658	point B	2200	2 330	$10,40	VLBC	350 000	Reykjavik -Africa	26 890	$27,28	6 month season
Ship to ship	Sept-Îles	31 320	$42,40	Stromness P7	Polar 7	194 658	Points C and D	2250	1925	$9,88	VLBC	350 000	SI - South Africa	27145	$27,53	6 month season
Ship to ship	St John NB	32 416	$44,12	Stromness P7	Polar 7	194 658	point C	2250	3125	$11,70	VLBC	350 000	St John - South Africa	27041	$27,43	6 month season
Rail-Marine route	Saguenay	23 907	$54,97	Rail			New rail line		900	$22,24	Suezmax	220 000	Saguenay - Suez	23007	$26,43	6 month season
Marine-rail-marine	Moosonee	24 822	$77,08	Laker size	Polar class 7	25 000	existing rail lines	316	1499	$40,05	Suezmax	220 000	Saguenay - Suez	23007	$26,43	6 month season
Marine-rail-marine	Churchill	13 250	$102,41	Laker size	Polar class 7	25 000	existing rail lines	1150	3497	$83,64	Capesize	185 000	Prince-Rupert	8 603	$10,17	6 month season

MAIN ROUTES AND SHIPPING COST EVALUATION BASED ON A 8 MONTH SHIPPING SEASON

8 month shipping season

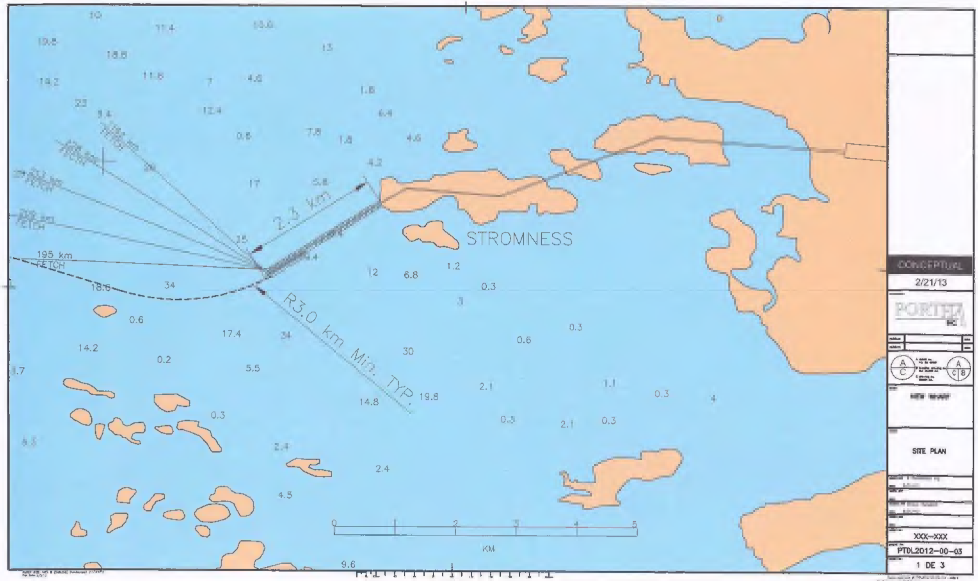
				FIRST LEG							SECOND LEG					
Type of Routing	Main Landmark	Total Distance km	Estimated Cost	Ship First Leg	Class	DWT	Route	km to Inter-mediate Point	km to Trans-fer Station	Cost First Leg	Ship Second Leg	DWT	Route	km to Qindao	Cost Final Leg	Seasonal Restriction
Terminal transfer	Nova Scotia	30 239	$44,14	Stromness P5	Polar 5	196 658	point C	2250	1250	$13,32	VLBC	350 000	NS - South Africa	26739	$26,53	8 month season
Ship to ship	Iceland	31 420	$48,72	Stromness P5	Polar 5	194 658	point B	2200	2 330	$16,44	VLBC	350 000	Reykjavik -Africa	26 890	$27,28	8 month season
Terminal transfer	Sept-Îles	31 320	$46,84	Stromness P5	Polar 5	196 658	Points C and D	2250	1925	$15,62	VLBC	350 000	SI - South Africa	27145	$26,92	8 month season
Terminal transfer	St John NB	32 416	$49,16	Stromness P5	Polar 5	196 658	point C	2250	3 125	$18,05	VLBC	350 000	St John - South Africa	27 041	$26,82	8 month season
Ship to ship	Sept-Îles	31 320	$48,72	Stromness P5	Polar 5	194 658	Points C and D	2250	1925	$16,19	VLBC	350 000	SI - South Africa	27145	$27,53	8 month season
Ship to ship	St John NB	32 416	$51,00	Stromness P5	Polar 5	194 658	point C	2250	3125	$18,57	VLBC	350 000	St John - South Africa	27041	$27,43	8 month season
Rail-Marine route	Saguenay	23 907	$54,97	Rail			New rail line		900	$22,24	Suezmax	220 000	Saguenay - Suez	23007	$26,43	8 month season
Direct	NW Passage	15 049	$82,26	Stromness P2	Polar 2	195 139	Hudson Strait	8 238		$51,39	$0,15	195 139	Hudson Strait	6 811	$30,71	8 month season
Marine-rail-marine	Moosonee	24 822	$77,94	Laker size	Polar class 7	25 000	existing rail lines	316	1499	$40,91	Suezmax	220 000	Saguenay - Suez	23007	$26,43	8 month season
Marine-rail-marine	Churchill	13 250	$105,56	Laker size	Polar class 7	25 000	existing rail lines	1150	3497	$86,79	Capesize	185 000	Prince-Rupert	8 603	$10,17	8 month season

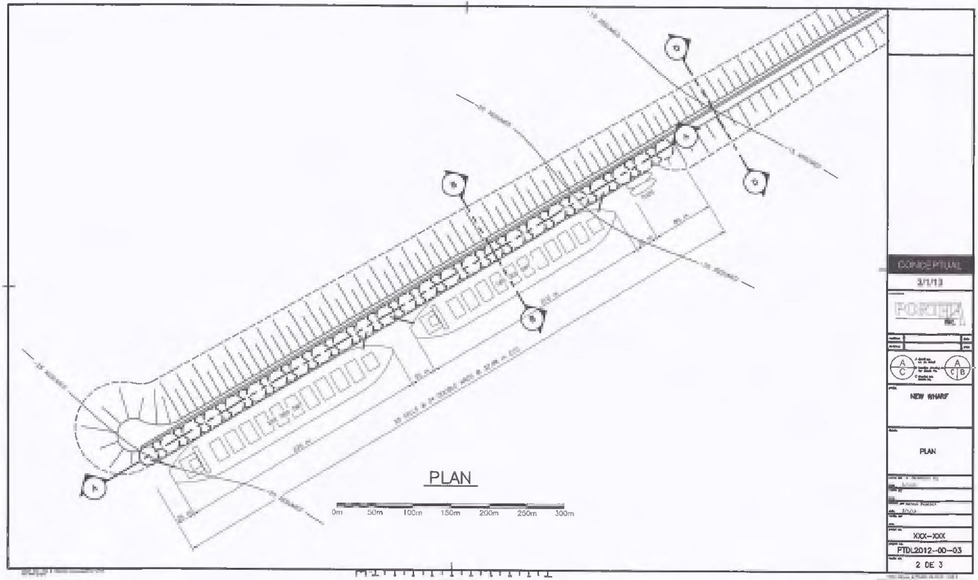
MAIN ROUTES AND SHIPPING COST EVALUATION BASED ON A 12 MONTH SHIPPING SEASON

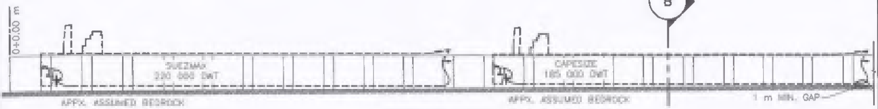
12 Month Shipping Season

							FIRST LEG						SECOND LEG			
Type of Routing	Main Landmark	Total Distance km	Estimated Cost	Ship First Leg	Class	DWT	Route	km to Inter-mediate Point	km to Trans-fer Station	Cost First Leg	Ship Second Leg	DWT	Route	km to Qindao	Cost Final Leg	Seasonal Restriction
Terminal transfer	Nova Scotia	30 239	$53,04	Stromness P5	Polar 5	196 658	point C	2250	1250	$22,22	VLBC	350 000	NS - South Africa	26739	$26,53	summer months
Rail-Marine route	Saguenay	23 907	$57,72	Rail			New rail line		900	$22,24	Suezmax	220 000	Saguenay - Suez	23007	$29,18	summer months
Ship to ship	Iceland	31 420	$57,92	Stromness P5	Polar 5	194 658	point B	2200	2 330	$25,64	VLBC	350 000	Reykjavik -Africa	26 890	$27,28	summer months
Terminal transfer	St John NB	32 416	$58,06	Stromness P5	Polar 5	196 658	point C	2250	3 125	$26,95	VLBC	350 000	St John - South Africa	27 041	$26,82	summer months
Ship to ship	St John NB	32 416	$60,26	Stromness P5	Polar 5	194 658	point C	2250	3125	$28,13	VLBC	350 000	St John - South Africa	27041	$27,43	summer months
Terminal transfer	Sept-Îles	31 320	$60,28	Stromness P5	Polar 5	196 658	Points C and D	2250	1925	$26,33	VLBC	350 000	SI - South Africa	27145	$27,53	summer months
Ship to ship	Sept-Îles	31 320	$62,14	Stromness P5	Polar 5	194 658	Points C and D	2250	1925	$26,93	VLBC	220 000	SI - Suez	22600	$33,59	summer months
Direct	NW Passage	15 049	$99,54	Stromness P2	Polar 2	195 139	Hudson Strait	8 238		$67,82	$0,00	195 139	Hudson Strait	6 811	$30,71	summer months
Marine-rail-marine	Moosonee	24 822	$84,34	Laker size	Polar class 7	25 000	existing rail lines	316	1499	$44,56	Suezmax	220 000	Saguenay - Suez	23007	$29,18	summer months
Marine-rail-marine	Churchill	13 250	$118,83	Laker size	Polar class 7	25 000	existing rail lines	1150	3497	$100,07	Capesize	185 000	Prince-Rupert	8 603	$10,17	summer months

APPENDIX B


2.3 km
STROMNESS
R3.0 km Min. TYP.
195 km
FETCH
0
1
2
3
4
5
KM
CONCEPTUAL
2/21/13
SITE PLAN
XXX-XXX
PT0L2012-00-03
1 DE 3

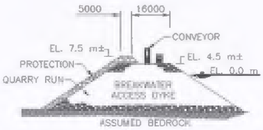

PLAN
0m
50m
100m
150m
200m
250m
300m
CONCEPTUAL
3/1/13
NEW WHARF
PLAN
3/1/13
XXX-XXX
PTDL2012-00-03
2 OE 3



SECTION A

SCALE/ÉCHELLE: 1:2000

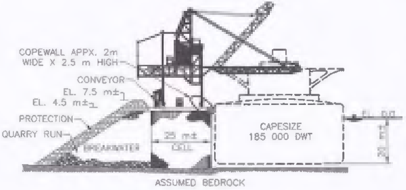
0m 50m 100m 150m 200m 250m



SECTION D

SCALE/ÉCHELLE: 1:1000

0m 10m 20m 30m 40m 50m 60m 70m 80m 90m 100m



SECTION B

SCALE/ÉCHELLE: 1:1000

0m 10m 20m 30m 40m 50m 60m 70m 80m 90m 100m

CONCEPTUAL

3/1/13

NEW WHARF

SECTIONS & DETAILS

3/1/13

XXX–XXX

PTDL2012–00–03

3 DE 3

APPENDIX C

Avril 2012

Avril 2012

Avril 2012

Avril 2012

03/05/2013

Raymond Gaudreault, P. Eng./ing.
Portha Inc.