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The historical mineral resources and other historical data and information mentioned in this Report are strictly historical in nature and are non-compliant with NI 43-101 and should therefore not be relied upon. A "qualified person" as defined by NI 43-101 has not done sufficient work to upgrade or classify the historical estimates as current mineral resources or mineral reserves, and Quebec Iron Ore Inc., Champion Iron Limited and their affiliates are not treating the historical estimates as current mineral resources or mineral reserves.

Technical Report
Bloom Lake Mine
Quebec Province, Canada

Report Date: January 31, 2013

Report Prepared for

Cliffs Natural Resources

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All data used as source material plus the text, tables, figures, and attachments of this document have been reviewed and prepared in accordance with generally accepted industry practices.

Signed on this 15th Day of February, 2013.

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Summary

This document has been prepared for Cliffs Natural Resources (Cliffs) to demonstrate the increase in resources and reserves and to show reconciliation between capital cost and operating cost in reference to the expansions for the Phase II plant and mine. The resources and reserve statement were conducted using the CIM guidelines. This document has been prepared to follow a similar format of the National Instrument 43-101 (NI 43-101) but some sections do not follow the guidelines. If filling is required, SRK must be contacted to ensure that the information in this report is current and requires no updates and that QP certifications are added to the final version.

Property Description

The Bloom Lake property is located in the Labrador Trough area which straddles the border between Quebec and Labrador. There are several iron ore mines in the area including Mont-Wright owned by ArcelorMittal, Carol Lake owned by Iron Ore Company of Canada (IOC), and Wabush Mines owned by Cliffs Natural Resources (Cliffs).

The Bloom Lake West property is located in the northeastern part of the province of Quebec, adjacent to the Labrador/Newfoundland border, in Normanville Township, Kaniapiskau County. The property is centered at approximately latitude 52° 50' 30" North and longitude 67° 17' West. The National Topographic System (NTS) map coverage is 23 B14. The Bloom Lake property is located 13 km northwest of the town of Fermont and 30 km west of the municipalities of Wabush and Labrador City.

Tonnes are reported in metric tonnes (t) of 2,205 lb.

Ownership

The property is owned by Bloom Lake General Partner Limited, a partnership between Cliffs Natural Resources (75%) and Wuhan Iron and Steel (Group) Corporation (WISCO) (25%).

All of the surface rights are property of the Crown (that is, the federal government of Canada).

Environmental

An additional site visit to Bloom Lake was conducted by SRK Environmental Principal, Mr. Mark Willow, in order to gather updated, relevant and appropriate information since the previous Technical Report was issued in December 2011.

An environmental impact study was prepared for the 16 Mt expansion project. Potential non-point sources of dust include the tailings pond, the waste rock piles and the ore and concentrate stockpile areas. The mine has dust mitigation measures for fine particle emissions, such as dust collectors, at some of the crushing and transportation facilities, and is in the process of constructing an A-Frame enclosure to protect the crusher and ore stockpile from precipitation and winds.

The Bloom Lake Mine has a concession to withdraw water from Bloom Lake for domestic (potable) use and for boiler make-up water. There is no restriction regarding the volume extracted. Lake of Confusion water may be used during outage of the reuse water circuit, which recycles tailings water back to the plant for reuse as process water.

The mine has been authorized for operation under the federal environmental authorities and provincial governments. The mine has received authorization for the Phase II expansion (in principle), and been granted permission to construct the associate structures and facilities for that expansion. Authorization to commence operations of Phase II is expected from the Ministry of Sustainable Development, Environment and Parks (Québec) in mid-2013.

The mine conducts routine monitoring of water, wastewater and air as part of their decrees and authorizations. With the exception of some minor releases to air and water, and improper storage of hazardous materials, the mine appears to be in compliance with its permits. The mine has an approved closure plan and should begin in 2013 with amortized annual payments to fund the financial guarantee established for reclamation. The amount of the financial guarantee is set at 70% of the total LoM reclamation costs (excluding the contingency). Both the closure plan and reclamation costs are currently being updated to include the Phase II facilities. The plan and financial guarantee have not get been approved by the Ministry of Natural Resources and Wildlife, but could be up to 150% of the previous estimates.

Geology and Mineralization

The rocks of the Bloom Lake area are part of the highly folded and metamorphosed southwestern portion of the Labrador Trough or Labrador-Quebec fold belt. The Labrador Trough consists of a succession of folded Proterozoic sedimentary and volcanic rocks and mafic intrusions deposited in interconnected sub-basins.

The Bloom Lake deposit is a Lake Superior–type iron formation which consists of banded sedimentary rocks of Proterozoic age that are composed of bands of iron oxides, magnetite and hematite, within quartz (chert)-rich rock. The iron formation at Bloom Lake comprises sequences that are rich in specular hematite and magnetite. They are commonly 25 to 40% iron oxide and can have thicknesses of up to 200 m. Specularite and magnetite may occur in anastomosing to discontinuous stringers and bands less than 10 cm thick in a quartz-rich rock or actinolite-quartz rock matrix. The actinolite-iron formation contains an average of about 15% magnetite, which is higher than the other rock types except magnetite iron formation.

Structurally, the Bloom Lake deposit comprises two gently plunging synforms on a main east-west axis that are separated by a gently north- to northwest-plunging antiform. These synforms result from at least two episodes of folding and are of regional scale. In addition to these regional scale folds, which have created the deposit-scale synforms shaping the Bloom Lake deposit, other folds of varying orientations occur on the property. As a result of various folding events or transposition of stratigraphy by folding, local reversals and repeats in the stratigraphic sequence in the iron formation are common.

Exploration

The Bloom Lake area has been explored since the 1950's, but major drilling programs took place in 1955 to 1957 (Jones and Laughlin Steel Corporation), 1966 (subsidiaries of Cleveland Cliffs Iron Company and J&L), 1998 (Quebec Cartier Mining) and 2005 to 2010 (Consolidated Thompson-Lundmark Mines Limited, later renamed Consolidated Thompson Iron Mines Limited) and 2011 to 2012 Cliffs.

A total of 108,284 m of core have been drilled on the property to date in 467 core holes, with 201 of those drilled by Consolidated Thompson-Lundmark Gold Mines Limited between 2005 and 2010. The samples were analyzed at Lakefield Research Laboratories, now referred to as SGS Lakefield.

The drilling covers an area about 4.7 km in length and 1 to 2 km in width.

Development and Operations

Bloom Lake has a current processing capacity of 7.2 Mt/y of fine sinter feed. The 7.2 Mt/y processing plant is commonly referred as Phase I. Bloom Lake is currently constructing an additional 7.3 Mt/y processing plant which is referred as Phase II. Construction of Phase II already begun and Phase II sinter production is planned for 2014. It is important to note that in November 2012, current operation decided to suspend Phase II construction which will have an effect on the start of Phase II production (estimated to be in early- to mid-2014). Since this news was released, SRK's updated mine plan and cash flow are based on starting in late 2013. SRK did not have time to update the new mine plan based on the delayed startup schedule.

In order to quantify and understand both the current 2012 baseline operational context (based on a 2200 t/h mill throughput rate which is derived from historical operating capacities and 2012 enhancement assumptions), as well as the forecast operational context (based on a 2400 t/h throughput rate which is derived from a forward looking enhancement of the mills with respect to a step change in productivity from 2012 levels), two separate LoM mine schedules were generated based on the same reserve design. With this information, the economics of the various positions could be derived and the sensitivity to this particular assumption can be understood.

The two mine plans generated for this evaluation are the following:

2,200 t/h Mill Throughput Mine Plan (2012 Baseline Case): This mine plan assumed the stated reserves found in this document combined with the following assumptions:

- Phase I mill throughput: 2,200 t/h;
- Phase II mill throughput: 2,200 t/h;
- Phase II Mill startup: End of first quarter 2013 (since the creation of this mine plan, an announcement by Cliffs occurred where construction for Phase II were delayed. SRK did not have the chance to update the mine plan to show the new Phase II start date);
- Constant grade throughput;
- Maximum total tonnes movement of 100 Mt per year;
- Fe recovery for Phase I: 72.2%; and
- Fe recovery for Phase II: 79%.

2,400 t/h Mill Throughput Mine Plan (Forecast Case): This mine plan assumed the stated reserves found in this document combined with the following assumptions:

- Phase I mill throughput: 2,400 t/h;
- Phase II mill throughput: 2,400 t/h;
- Phase II Mill startup: End of first quarter 2013 (since the creation of this mine plan, an announcement by Cliffs occurred where construction for Phase II were delayed. SRK did not have the chance to update the mine plan to show the new Phase II start date);
- Constant grade throughput;

- Maximum total tonnes movement of 113 Mt per year;
- Fe recovery for Phase I: 72.2%; and
- Fe recovery for Phase II: 79%.

Mineral Resource Estimate

The Bloom Lake resource estimation contained in this report was prepared by Mr. Peter Jongewaard, Cliffs Senior Staff Geologist. The geologic model was prepared by Gemcom International, Inc. (Gemcom) consultants in conjunction with Bloom Lake geologists. Mr. Jongewaard used Maptek Vulcan software to complete the resource estimation and the same estimation methodology used by SRK in 2011. The updated resource was reviewed by Ms. Leah Mach, SRK Principal Resource Geologist.

A pit optimization was run with the following parameters to constrain the resources as potentially mineable:

- Mass recovery (MR), where:
  - o Concentrate Mass = [Mass Crude Ore*[Crude Fe(t) * Fe Recovery]]/Concentrate Fe(t), and
  - o Based on 79% Fe Recovery and a 66.2% Fe Concentrate.
- Average Product Value = US$120.00/t of 66.2% Fe concentrate fob Sept-Îles;
- Bulk Mining Cost/t US$2.70/t;
- Surface Mining Cost/t AMT (All undisturbed <20 m upper surface) = US$4.10/amt;
- In situ Mining Recovery = 100%;
- Tailings and water management = US$1.00/t RoM;
- Processing Cost = US$6.62/t RoM;
- G&A Costs = US$4.50/t of concentrate;
- Logistics: Port and Transportation = US$25.00/t Concentrate; and
- Pit Slope Angle = 40.0^{o} for all material.

The Mineral Resources of the Bloom Lake deposit on a dry tonnage basis are presented in Table 1. The resources are stated at a cut-off grade of 15% for material inside the resource pit shell.

The classification meets the standards for resource classification set by CIM.

Table 1: Bloom Lake Mineral Resources, January 1, 2013*

Class	Material	Mt	FeT %	MagFe %	CaO %	MgO %
Measured	IF	393.2	30.1	3.2	0.47	0.58
	SIF	52.9	24.5	9.0	4.05	4.70
	Total	**446.1**	**29.5**	**3.9**	**0.90**	**1.07**
Indicated	IF	538.1	28.5	4.1	0.89	0.92
	SIF	381.7	25.1	8.8	3.96	3.79
	Total	**919.8**	**27.0**	**6.1**	**2.17**	**2.11**
Measured and Indicated		1365.8	27.8	5.38	1.76	1.77
Inferred	IF	143.2	29.8	2.5	0.50	0.59
	SIF	275.7	22.1	14.1	3.92	3.50
	Total	**419.0**	**24.8**	**10.1**	**2.75**	**2.50**

*Effective Date: Resource Estimation October 31, 2012, Topography at July 1, 2012;
Tonnages and grade are rounded to reflect approximation; and
Resources are stated at a cut-off of 15% FeT and are contained within a pit optimization shell.
Mineral Resources are reported inclusive of Mineral Reserves

Mineral resources that are not mineral reserves do not have demonstrated economic viability. Mineral resource estimates do not account for mineability, selectivity, mining loss and dilution. These mineral resource estimates include inferred mineral resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. There is also no certainty that these inferred mineral resources will be converted to Measured and Indicated categories through further drilling, or into mineral reserves, once economic considerations are applied.

Mineral Reserve Estimate

Using a cost structure based on 2012 actual production and the 2012 to 2016 budget, a new economic analysis was completed resulting in a new ore reserve statement for Bloom Lake. The proven and probable ore reserves for Bloom Lake based on this economic analysis are presented in Table 2.

Table 2: Bloom Lake Reserves as of January 1, 2013*

Classification	Tonnes	FeT	CaO	MgO	Mag Fe	Magnetite	Concentrate
	Mt (dry)	%	%	%	%	%	Mt (dry)
Proven	417.4	29.87	0.86	0.98	3.84	5.33	141.9
Probable	633.9	27.97	2.24	2.17	6.57	9.13	201.8
Proven + Probable	1,051.3	28.7	1.7	1.7	5.5	7.6	343.7

- Reserves effective date: July 1, 2012;
- All RoM and product tonnages reported are in dry metric tonne units.
- Reserves are reported not diluted;
- Reserves are reported within Mineral Lease Boundary;
- Average global strip ratio for the reserve was calculated to be 1.48 (waste to ore)
- Reserves based on July 1, 2012 topography;
- Cutoff used for Fet: 20%
- Concentrate Product is 66.2% Fe;
- Crude to Concentrate Mass Recovery Equation For all years based on the following equation: Concentrate Mass = [Mass Crude Ore*[Crude Fe(t) * Fe Recovery]]/Concentrate Fe(t). (Based on 79% Fe Recovery and a 66.2% Fe Concentrate). Phase I Fe recovery is calculated to be 72.2%. Phase II Fe Recovery is calculated to be 79%. Phase I and II Combined Fe recovery is calculated to be 75.3%.
- Reserves use US$112/t of concentrate selling price.
- Reserve material mined from July 1, 2012 through December 31st, 2012 is 6.7Mt of Crushed ore, 2.18Mt of concentrate, Crushed Fe grade of 32.4%. Waste material mined is 7.35Mt.

The 3-year benchmark trailing-average price adjusted for Bloom Lake site as per Cliff's corporate policy was used as the basis to determine proven and probable reserves. From this evaluation, a series of Whittle™ shells were generated based on revenue factors of the current 3-yr trailing price of US$112/t, f.o.b. (Free On Board) the port of Sept-Îles. The selected shell was then used to guide a full engineered pit design, which was then phased appropriately and scheduled for mining activities for the duration of the project. Ore within the new ultimate pit limit is sequenced for all 29 years of the proven and probable reserves at a concentrate sinter production of 12 to 14 Mt/y. The costs used in this study represent all mining, processing, transportation and administrative costs, including loading of the sinter concentrate into lake freighters at Sept-Îles Port, Quebec

The updated reserves of 1,051.3 Mt of proven and probable ore are similar to those reported in 2011. The new reserve is based on the updated 2012 Resource which was audited by SRK. SRK audited an upside case where Phase I and II mill throughput were increased from 2,200 t/h to 2,400 t/h. This upside mill throughput mine plan yielded a higher NPV but no change in the reserves.

Detailed parameters used for the pit optimization are:

- Mass recovery (MR) = Concentrate Mass = [Mass Crude Ore*[Crude Fe(t) * Fe Recovery]]/Concentrate Fe(t). (Based on 75.3% Fe Recovery and a 66.2% Fe Concentrate);
- Average Product Value = US$112.00/t of concentrate;
- Bulk Mining Cost/t Ore = US$2.70/t; Waste = US$2.70/t;
- Surface Mining Cost/t AMT (All undisturbed <20 m upper surface) = US$4.10/amt;
- In situ Mining Recovery = 95% (by Mass) of Crude Ore;
- Tailings and water management = US$1.00/t RoM;
- Processing (Plant) Cost = US$6.62/t RoM;
- Admin/SG&A Costs = US$4.50/t of concentrate; and
- Logistics: Port and Transportation = US$25.00/t Concentrate.

Pit Slope Angle = 40.0^{o} for all material (including ramps).

Metallurgy and Mineral Processing

The initial metallurgical testwork for Bloom Lake was undertaken during the period 1973 to 1976. At that time Lakefield Research (now SGS Lakefield) conducted gravity concentration tests with a Wilfley shaking table on drill core composites crushed to 35 mesh (425 microns). In 2005, eleven mini-bulk samples were taken from iron formation outcropping on the property and sent to SGS Lakefield for testing. Additional confirmatory tests were run in 2006 to 2007 on 32 drill core composites as part of the Bloom Lake Feasibility study prepared by BBA and issued in November 2008. This confirmatory testwork confirmed the results of the previous studies and projected average weight recoveries of 38% without a magnetite circuit and 41% with the inclusion of a magnetite recovery circuit. The average iron feed to achieve these recoveries are 31% crude iron.

During 2010 and 2011, pilot tests were conducted in the existing Phase I concentrator to validate alternate flowsheets and equipment for the Phase II concentrator and included evaluations of:

- Classification hydrosizers;
- Rougher Spirals with current feed;
- Rougher Spirals with hydrosizer overflow as feed; and
- Cleaning hydrosizer and scavenger spirals.

The Phase I concentrator was commissioned during March, 2010. Ore from the mine is delivered by truck to the primary gyratory crusher and crushed to -8 inch. Crushed ore is then fed to a 11 m diameter x 6.1 m long autogenous grinding (AG) mill driven by dual 5,590 kW motors. The AG mill discharges onto two pebble screens with the screen oversize being recirculated back the AG mill. Screen undersize is pumped to four banana-style vibrating screens to produce an 850 micron (20 mesh) separation. Screen undersize from the grinding circuit is advanced to the rougher gravity concentration circuit, which consists of 32 banks of rougher spirals arranged in four lines. Concentrate from each bank of rougher spirals is combined and distributed by a 24-way rubber line cleaner spiral distributor to feed a bank of 12 double-start cleaner spirals. Concentrate from each cleaner spiral feeds directly to a recleaner spiral for final upgrading. Concentrate from the recleaner spirals is dewatered on four horizontal pan filters fitted with steam hoods to dry the concentrate to about 2.5% moisture during the winter months and 4.5% during the summer months.

The Phase I concentrator has not yet achieved design production rates or iron recovery projections. During 2012 the plant has averaged 2,033 t/h versus a design production rate of 2,372 t/h. Iron recovery has averaged 72% and concentrate weight percent recovery has averaged 34.7% into final iron concentrates that averaged 66.2% Fe and 4.6% SiO_2.

The Phase II concentrator plant is currently under construction and scheduled for completion during 2014. The Phase II concentrator flowsheet will incorporate improvements in equipment selection and sizing based on pilot plant testing and experience gained from the operation of the Phase I concentrator. The main difference between the two concentrators is the incorporation of hydrosizers in the Phase II concentrator flowsheet, which will be used, in conjunction with spiral concentrators, to produce an iron gravity concentrate.

Due to improvements in equipment selection, iron recovery in the Phase II concentrator will be better than has been achieved in the Phase I concentrator. It is SRK's opinion that iron recovery in the Phase II concentrator will be in the range of 78% to 80%. Throughput capacity is estimated at 2,200 t/h. Management expects that improvements will increase throughputs to 2,400 t/h. Recent performance has shown increases in mill throughputs. December 2012 average was 2,370 t/h, and the January 2013 average was 2,290 t/h.

Economic Analysis

The financial results of this report are based upon work performed by SRK and have been prepared on an annual basis. All costs are constant U.S. dollars. The exchange rate used for all economic analysis is CAN$1 to US$1.

Two cash flows were put together to show the viability of the project. The first cash flow shows the current 2012 operational metrics assuming no improvements are made. The second cash flow show the Cliffs operation forecast based on the capital investment to improved conditions at the site.

A financial model was prepared on a pre-tax basis the results of which are presented in this section. Key criteria used in the analysis are discussed in detail throughout this report. Financial assumptions used are shown summarized in Table 3. Due to the WISCO and Cliffs partnership agreement, no port and rail expansion capital costs were included.

Table 3: Model Parameters

Description	2,200 t/h Case Value	2,400 t/h Case Value
Mine Life	30 years	27 years
Ore Processed	1,044.2 Mt	1,044.2 Mt
Tonnes Mined	2,593.5 Mt	2,593.5 Mt
Payable Tonnes Fe Concentrate	341.4 Mt	341.4 Mt
Life of Mine Operating Costs	$73.14/t Conc.	$67.77/t Conc.
Life of Mine Capital	$2,844.7 million	$2,894.6 million
Fe Recovery	75.35%	75.35%

*Not including Port/Rail Capital Items

The economic analysis results, shown in Table 4, indicate a pre-tax NPV 9.7% of US$2.54 billion and US$3.48 billion and pre-tax IRR of 34% and 44% for the 2200 t/h and 2400 t/h cases, respectively. The NPV and rate of return were calculated for the project as a whole, including revenue from installed and expansion capacities. Capital identified in the economics is for a Phase II expansion along with sustaining operations and plant rebuilds as necessary.

Pre-tax project economic results and estimated cash costs are summarized in Table 4.

Table 4: Mine and Plant Economic Results

Description	units	2200 t/h Case		2400 t/h Case	
Market Price	/t-product		$112.00		$112.00
Estimate of Cash Flow (all values in $000's)					
Gross Income					
Fe-Concentrate	$112.00		38,239,236		38,239,236
Gross Income			**$38,239,236**		**$38,239,236**
Net Revenue	0.0%		**$38,239,236**		**$38,239,236**
Operating Costs					
Mining (US$/t-mined)		$2.80	7,254,027	$2.83	7,341,227
Concentrating (US$/t-RoM)		$6.62	6,912,351	$4.95	5,168,601
Tailings & Water Manag. (US$/t-RoM)		$1.00	1,044,162	$1.05	1,093,391
Logistics (US$/t-conc)		$21.65	7,390,910	$21.65	7,390,607
G&A (US$/t-conc)		$6.94	2,370,886	$6.27	2,142,144
Total Operating			**$24,972,335**		**$23,135,970**
Operating Margin (EBITDA)			$13,265,905		$15,101,863
Capital			2,844,736		2,894,575
Pre-Tax Cash Flow Available for Debt Service			**$10,422,165**		**$12,207,288**
Pre-Tax NPV 9.7%			**$2,545,153**		**$3,482,001**
Pre-Tax IRR			**34%**		**44%**

Economic results assume that port and rail capital are allocated.
Economics assume that current trans-shipment cost will end in 2014.
2,200 t/h SRK view of the project as it is currently operated.
2,400 t/h Cliffs view of the project applying extra capital to improve cash costs.

Sensitivity analysis for key economic parameters is shown in Tables 5 and 6. The Project is nominally most sensitive to market prices (revenues) followed by operating costs. The Project is the least sensitive to capital.

Table 5: 2200 t/h Case Sensitivity Analysis of NPV 9.7% (US$M)

NPV (US$MM)	-20%	-10%	Base	+10%	+20%
Revenue	(140)	1,203	2,545	3,888	5,230
Operating Costs	4,365	3,455	2,545	1,635	726
Capital Costs	2,901	2,723	2,545	2,367	2,189

Table 6: 2400 t/h Case Sensitivity Analysis of NPV 9.7% (US$M)

NPV (US$MM)	-20%	-10%	Base	+10%	+20%
Revenue	623	2,053	3,482	4,911	6,341
Operating Costs	5,278	4,380	3,482	2,584	1,686
Capital Costs	3,848	3,665	3,482	3,299	3,116

An expansion project has been underway that will double the size of the project. The current capital estimate is US$1.4 billion as detailed in Table 7.

Table 7: Expansion Project Current Budget (US$ millions)

Capital Category	Total Capital
Development	68.7
Mobile Equipment	125.3
Plant	818.6
Housing	7.5
Expansion	**$1,020.0**
Garage	48.0
Tailings Disposal System	195.0
Mine Infrastructure Upgrades	49.2
Port and Rail	-
Common Infrastructure	**$292.2**
Total	**$1,312.2**
Contingency	106.7
Total with Contingency	**$1,418.9**

The Project expansion spending is estimated to be US$582.1 million through the end of 2012 with an estimated US$836.8 spent through 2015 to complete the expansion.

Figure 1 shows the project growth over the life of construction as the Project was modified from the original basis.

A detailed analysis of the Project capital growth is summarized in Section 18.1.1.

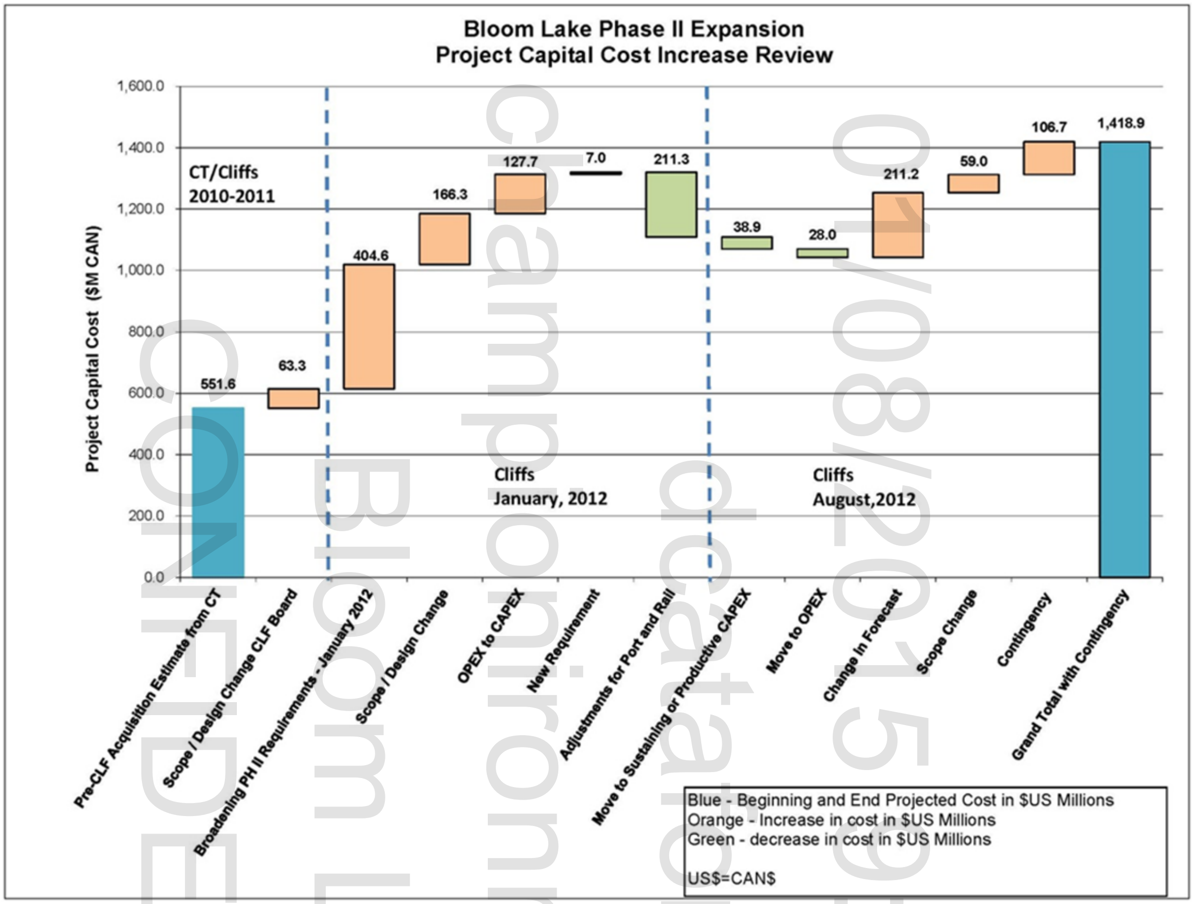

Figure 1: Expansion Project Cost Increase (CAN$ millions)

Conclusions and Recommendations

Environmental and Permitting

The mine has been authorized for operation under the federal environmental authorities and provincial governments. The mine has received initial authorization for the Phase II expansion, but does not yet have the operating authorization which is expected to be received during the first half of 2013. The mine conducts routine monitoring of water, wastewater and air for environmental purposes. It is currently in compliance with its permits; however, there may be a future enforcement action related to permit violations that occurred in May 2011. The mine has an approved closure plan and will be subject to annual payments for financial guarantees starting in 2013.

Mineral Resource Estimate

This report states the combined resources for the Main Pit area and Bloom Lake West. Previous NI 43-101 Technical Reports stated resources separately for each of the areas. SRK was responsible for the auditing resource estimation. The estimation was conducted with ordinary kriging and inverse distance squared for each of the domains. Composites coded as SIF could be used in any of the SIF structural domains and composites coded as IF could be used in any of the IF structural domains. The estimation was accomplished in three passes; the first two having a search distance of 200 m x 200 m x 100 m and the third having a search distance of 600 m x 600 m x 200 m. The first pass required a minimum of two drillholes for estimation.

The resources were categorized as Measured, Indicated or Inferred using the following criteria:

- Measured: Minimum of 3 drillholes with the closest composite within 80 m of the block centroid and estimated in Pass 1;
- Indicated: Minimum of 2 drillholes with the closest composite within 200 m of the block centroid and estimated in Pass 1; and
- Inferred: Blocks estimated in Pass 2 or 3.

The classification meets CIM guidelines for classification of resources.

Mineral Reserve Estimate

- This report states the combined reserves for the Main Pit area and Bloom Lake West. SRK was responsible for the auditing reserve estimation;
- Due to the increase in strip ratio and mill feed, SRK recalculated the mine fleet to use larger mining equipment. An SRK fleet estimation study suggested that 300-tonne trucks would reduce mining costs and improve production;
- Bloom Lake has the opportunity to adjust to different market conditions based on ore availability. This flexibility allows the site to handle swings in commodity prices; and
- The classification meets CIM guidelines for classification of reserves.

Mineral Processing

SRK makes the following conclusions and recommendations with respect to Bloom Lake process facilities:

- The Phase I concentrator was commissioned during March, 2010;
- The Phase II concentrator plant is currently under construction and scheduled for completion during 2014. The Phase II concentrator will incorporate improvements in equipment selection and sizing based on pilot plant testing and experience gained from the operation of the Phase I concentrator;
- The main difference between the two concentrators is the incorporation of hydrosizers in the Phase II concentrator flowsheet, which will be used, in conjunction with spiral concentrators, to produce an iron gravity concentrate;
- It does not appear that the original comminution studies were conducted on samples that represented the extent of the variability that may occur in the deposit. The samples used to for the comminution studies reported by BBA in 2008 were from mini-bulk composites 10 and 11, taken from surface outcrops from the west zone of the ore deposit. Some additional SMC (SAG mill comminution) testing appears to have been done on mini-bulk composites 4 and 8, also from surface outcrop samples. It should be noted that there is the potential of the ore being harder in the eastern zones of the ore deposit due to the higher levels of magnetite. In order to obtain a better understanding of the variability of the ore hardness, SRK would recommend that additional comminution testing be done on selected drill core from throughout the deposit. This will enable a better understanding of throughput capacity achievable with the AG mills installed in both concentrators;

- Iron recovery in the Phase I concentrator has averaged about 72% which is about 12.5% below the original iron recovery target. It is unlikely that iron recovery can be significantly improved without major equipment modifications in the Phase I concentrator. In order to predict iron recovery from the Phase I concentrator, SRK has applied a 12.5% iron recovery discount to the iron recovery equation that was originally used for Phase I;
- The Phase II concentrator has been designed with the same size AG mill that was installed in the Phase I concentrator. However, the Phase II concentrator is projected to process 2,600 t/h. Given that the Phase II concentrator will be processing the same ore as the Phase I concentrator, it is very unlikely that a throughput capacity of 2,600 t/h can be achieved. Phase II concentrator throughput, however, may be incrementally better than the Phase I concentrator due to:
    - Provision for better blending of the ore prior being fed to the AG mill, and
    - Installation of hydrosizers in combination with spiral concentrators, which may serve to off-load of portion of the ore at a coarser size split.
- SRK believes that the Phase II concentrator throughput capacity may likely be in the range of 2,200 to 2,400 t/h;
- Phase I concentrator operating costs for 2012 (January to September) averaged US$18.94/t of concentrate, which is equivalent to US$6.20/t of plant feed. It is likely that Phase II concentrator operating costs will be similar; and
- Secondary comminution circuits should be evaluated for both the Phase I and Phase II concentrators after commissioning of the Phase II concentrator has been completed and actual concentrator performance has been confirmed. The options that should be investigated include:
    - Pebble crushing,
    - Secondary ball milling, and
    - Conversion of the AG mills to SAG mills.

Mineral Economics

The economic analysis results indicate a pre-tax NPV 9.7% of US$2,545 million for the 2,200 t/h case and indicate an NPV 9.7% of US$3,482 million for the 2,400 t/h case. Capital identified in the economics is for sustaining operations, plant rebuilds as necessary and the Phase II Expansion.

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Appendices

1 Introduction

1.1 Terms of Reference and Purpose of the Report

SRK Consulting (U.S.), Inc. (SRK) was engaged by Cliffs Natural Resources (Cliffs) to prepare a Technical Report (the Report) for the Bloom Lake Iron Ore Mine (Bloom Lake or the Project) in Normanville Township, Quebec Province, Canada. This report provides mineral resource and mineral reserve estimates, a classification of resources and reserves in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) classification system and an economic evaluation of the property.

1.2 Qualifications of Consultants (SRK)

The SRK Group (SRK) comprises of over 1,500 staff, offering expertise in a wide range of resource engineering disciplines. SRK's independence is ensured by the fact that it holds no equity in any project and that its ownership rests solely with its staff. This permits SRK to provide its clients with conflict-free and objective recommendations on crucial judgment issues. SRK has a demonstrated record of accomplishment in undertaking independent assessments of mineral resources and mineral reserves, project evaluations and audits, technical reports and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide. SRK has also worked with a large number of major international mining companies and their projects, providing mining industry consultancy service inputs.

This report has been prepared based on a technical and economic review by a team of consultants sourced from the SRK's Denver and Tucson offices. These consultants are specialists in the fields of geology, exploration, mineral resource and mineral reserve estimation and classification, open pit mining, mineral processing and mineral economics.

Neither SRK nor any of its employees and associates employed in the preparation of this report has any beneficial interest in the Company or in the assets of the Company. SRK will be paid a fee for this work in accordance with normal professional consulting practice.

The individuals who have provided input to this technical report, who are listed below, have extensive experience in the mining industry and are members in good standing of appropriate professional institutions. The key Project personnel contributing to this report are listed in Table 1.2.1.

Table 1.2.1: Key Project Personnel

Team Member	Discipline
Fernando Rodrigues, MBA, MMSAQP	Project Manager, Mining and Reserves
Leah Mach, CPG, MS	Resources
Peter Jongewaard (Cliffs)	Geology
Dorinda Bair	Laboratory QAQC
Eric Olin, M.Sc.,MBA, SME	Process
Mark Willow	Environmental, Permitting, Safety, Land Status
Jeff Osborn	Infrastructure
Valerie Obie	Project Economics
Neal Rigby, CEng, MIMMM, PhD	Project Director, QA/QC, Overview and Review

1.2.1 Details of Inspection

The Project was visited by Fernando Rodrigues, Leah Mach and Eric Olin on September 7, 2011. Jeff Osborn and Mark Willow visited the Project on December 19, 2012. The site visit consisted of the following activities:

- Review of regional and property geology;
- Review of drill core and comparison to drill logs;
- Visit to the open pit mine; and
- Visit to the process plant and tailings storage facility and discussion with key personnel on operating and capital costs.

1.3 Reliance on Other Experts

SRK's opinion contained herein is based on information provided to SRK by Cliffs throughout the course of SRK's investigations. The sources of information include data and reports supplied by Cliffs' personnel, as well as documents included/referenced in Section 22.

SRK used its experience to determine if the information from previous reports and other consultants was suitable for inclusion in this Report and adjusted information that required amending. The level of detail utilized was appropriate for this level of study.

Historical selling price has been provided by Cliffs and used in the economical evaluation.

1.3.1 Sources of Information and Extent of Reliance

SRK has relied on information provided by Cliff's personnel as to land and mineral ownership and any associated royalties. The information was supplied to SRK between September and November 2011. SRK has not independently verified ownership of land or minerals and has not reviewed the agreements between the individual owners and Cliffs. This information is cited in Sections 2.2 and 2.3.

1.4 Effective Date

The effective date of this report is January 31, 2013.

1.5 Units of Measure

The metric system has been used throughout this report. All currency is in United States dollars (US$) unless otherwise stated. Tonnes are metric of 1,000 kg, or 2,205 lb. Abbreviations used in this report are presented in Section 26.4.

2 Property Description and Location

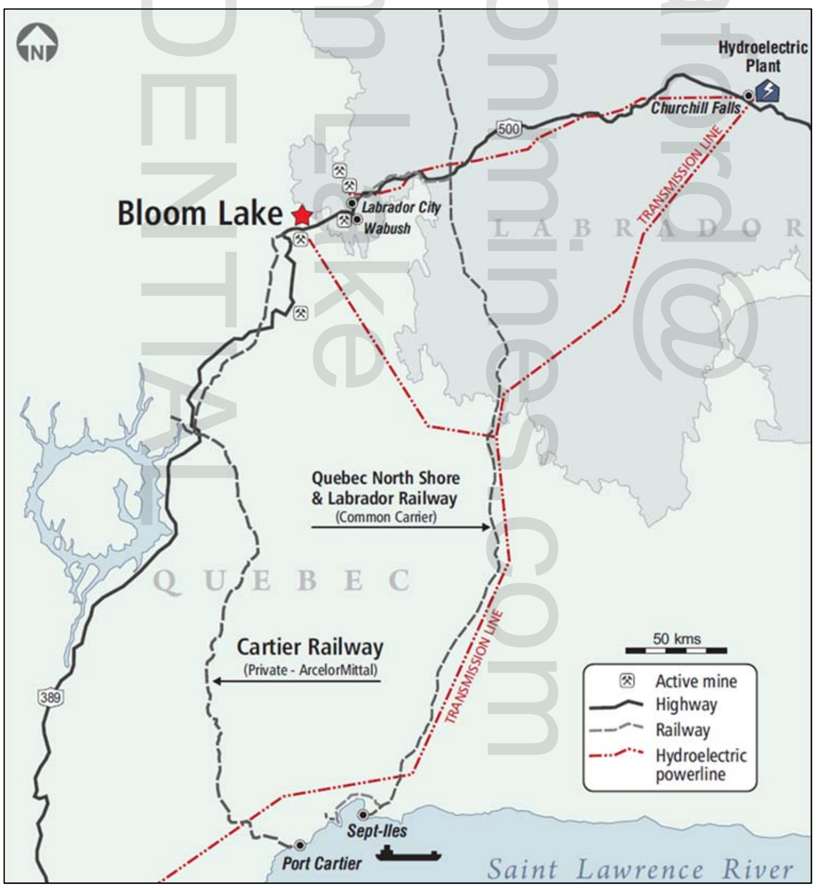
The Bloom Lake property is located in the Labrador Trough area which straddles the border between Quebec and Labrador. There are several iron ore mines in the area including Mont-Wright owned by ArcelorMittal, Carol Lake owned by Iron Ore Company of Canada (IOC), and Wabush Mines owned by Cliffs Natural Resources (Cliffs).

The property is owned by Bloom Lake General Partner Limited (Bloom Lake GPL), a partnership between Cliffs Natural Resources (75%) and the Wuhan Iron and Steel (Group) Corporation (WISCO) (25%).

2.1 Property Description and Location

The Bloom Lake property is located in the northeastern part of the province of Quebec, adjacent to the Labrador/Newfoundland border, in Normanville Township, Kaniapiskau County. The property is centered at approximately latitude 52° 50' 30" North and longitude 67° 17' West. The National Topographic System (NTS) map coverage is 23 B14. The Bloom Lake property is located 13 km west of the town of Fermont and 30 km southwest of the municipalities of Wabush and Labrador City (Figure 2.1.1).

All of the surface rights are property of the Crown (that is, the federal government of Canada).



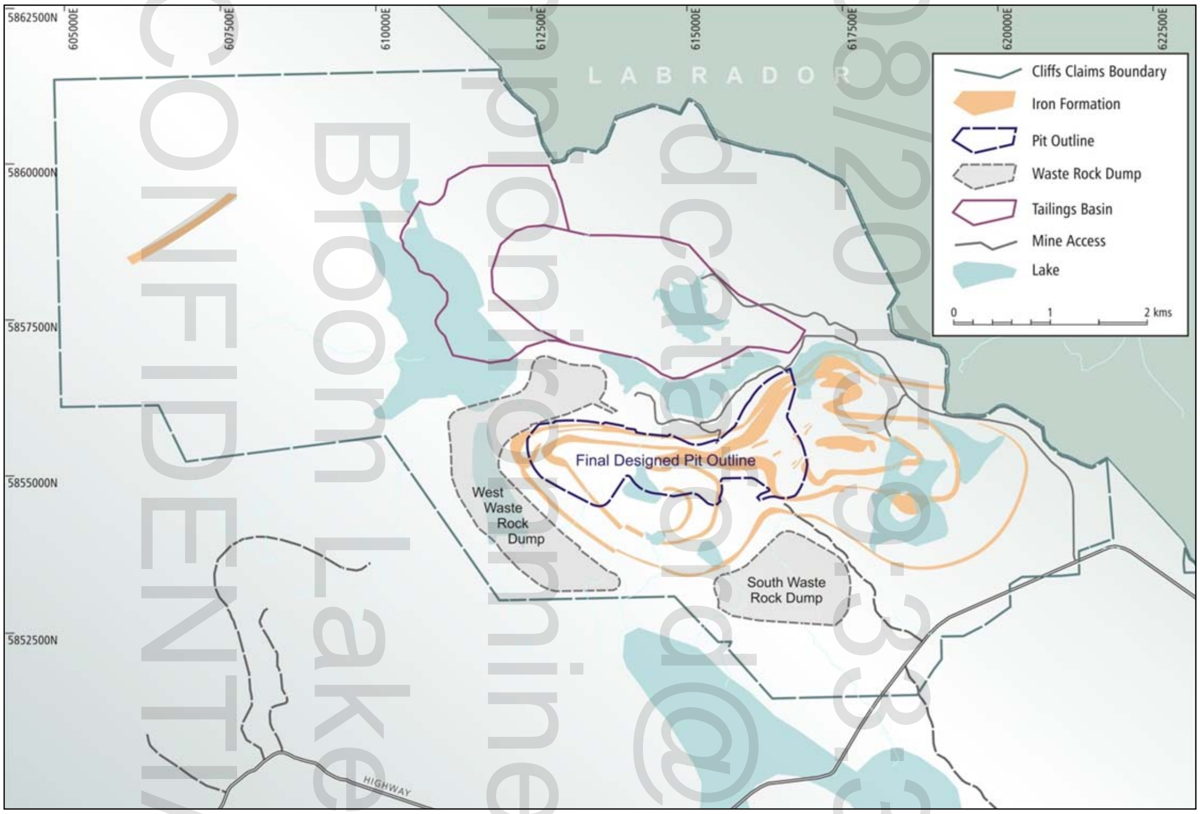
Source: Cliffs, 2013

Figure 2.1.1: Location of the Bloom Lake Property

2.2 Mineral Titles

2.2.1 Nature and Extent of Issuer's Interest

Bloom Lake GPL holds 100% of 114 active claims (4101.27 ha) outside of the Mining Lease (BM 877) which has a total of 6857.7 ha. The mining lease boundaries are in compliance with the restriction zones and the claims within the mining lease have been suspended. Those claims outside the mining lease remain active. The claim boundary is shown in Figure 2.2.1.1. Appendix A is a list of the mining claims and the respective expiration dates.



Source: Cliffs

Figure 2.2.1.1: Bloom Lake Claims and Mining Lease

2.3 Royalties, Agreements and Encumbrances

There are no royalties, agreements or encumbrances on the Bloom Lake property.

2.4 Environmental Liabilities and Permitting

2.4.1 Environmental Liabilities

Several notices of violation for have been issued by the Ministry of Sustainable Development, Environment and Parks, Regional Management to the operation since acquisition by Cliffs in May 2011. Most of these were relatively minor and required minimal effort to address. Two important, though not necessarily material releases, were charged following an inspection between 23 and 27

May, 2011 when the company was found to have discharged waters containing Total Suspended Solids (TSS) in violation of the Law on Environmental Quality, Article 20, as well as a release of a hazardous substance (ferric sulfate) in violation of Regulations on Hazardous Materials, Article 8. A follow-up investigation was conducted by the Ministry in late 2012, but has yet to indicate whether any further enforcement actions will be taken. NOVs for the release of TSS-containing waters to the environment appear to have been issued again following an inspection on August 6, 2012 and September 20, 2012, suggesting a possible chronic issue requiring further mitigation.

Minor infractions for the storage of hazardous materials (in violation of Regulations on Hazardous Materials), and dust emissions (in violation of Regulations on Quality of the Atmosphere), were also noted. There are currently no other known material environmental liabilities associated with the Bloom Lake Mine. Additional information regarding environmental permitting and potential future environmental issues are discussed in Section 18.

2.4.2 Required Permits and Status

The mine has its required environmental permits for the current operation, as shown in Table 2.4.2.1. It has also received the environmental permits and authorizations for the proposed expansion (Phase II). A summary of the more material permits and authorizations relevant to the mine are provided below. This is not a comprehensive list, but was summarized from information provided by Cliffs.

Table 2.4.2.1: Environmental Permits

Permit Name and Description	Agency	Date Authorized	Expiration(1)
Certificate of authorization for Bloom Lake iron mine project	Government of Quebec	2/20/2008	none
Certification of Authorization (CA-8) for operation of the Bloom Lake iron mine	Ministry of Sustainable Development, Environment and Parks (Québec)	3/2/2010	none
Authorization Certificate for construction and operation of 2 wastewater treatment systems for the plant	Ministry of Sustainable Development, Environment and Parks (Québec)	1/24/2011	none
Certificate of authorization for railroad	Ministry of Sustainable Development, Environment and Parks (Québec)	4/20/2010	none
Certificate for operation of six dust collectors	Ministry of Sustainable Development, Environment and Parks (Québec)	9/20/2010	none
Certificate of authorization for modification of Bloom Lake mine, Phase II	Ministry of Sustainable Development, Environment and Parks (Québec)	9/2011	none
Certificate of authorization for construction of new infrastructure	Ministry of Sustainable Development, Environment and Parks (Québec)	9/2011	none
Decrees 608-2012 and 764-2012 modifying decree 137-2008 of February 20th, 2008 to expand the pit(s), waste rock stockpiles, and tailings area, respectively.	Ministry of Sustainable Development, Environment and Parks (Québec)	6/2012 & 7/2012	none

Source: Cliffs, 2013
(1) none noted in information provided

The Phase II expansion project was approved by the Ministry of the Environment in a decree modification in September 2011, which approves the project in principle. An authorization for construction of new infrastructure associated with Phase II was also granted at that time. However, authorization to commence operations for Phase II is currently under review by the Ministry, for which approval is anticipated by mid-2013 (according to Cliffs).

In 2013, the agreement(s) with the city of Fremont for the mine man-camp facilities will expire. As a result, Cliffs has begun the process of relocating those facilities closer to the mine, and within the concession boundary. The process of site selection, near water sources, roads, power, etc., has been initiated, though the formal permitting process has not yet begun. According to Cliffs, the permitting process is expected to take six to nine months, especially with respect to new potable water systems and sewage disposal. In the meantime, Cliffs plans to negotiate with Fremont and others on a short-term agreement to continue utilizing their facilities.

Future expansions to the north of the currently permitting project will require the involvement of the Department of Fisheries and Oceans (formerly the Department of Marine and Fisheries), Transport Canada, given the lakes and water issues in the area. Permitting through this agency typically takes longer than through the Ministry of Sustainable Development, Environment and Parks; as such, conceptual plans are currently being developed for this possible scenario.

2.5 Other Significant Factors and Risks

SRK is not aware of any other known significant factors or risks which have not been disclosed in this report.

3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

3.1 Access

From the town of Fermont, Quebec Canada, Bloom Lake can be accessed by traveling approximately 7 km west on Highway 389, then approximately 6 km northwest on a well-maintained gravel road to the Project.

The town of Fermont is about 565 km north of the town of Baie-Comeau on Highway 389, located on the mouth of the St. Lawrence River. Fermont (meaning Iron Mountain) is the regional county seat of the Municipality of Caniapiscau, and was founded in the 1970's as a mining town supporting the iron deposits of the area.

The Wabush airport is located about 30 km northeast of the Project on the Trans-Labrador Highway (Route 500, and the continuation of Highway 389) between the towns of Labrador City and Wabush, Newfoundland and Labrador, Canada. There is frequent service from this airport to Montreal.

3.2 Topography, Elevation and Vegetation

The topography at Bloom Lake consists of gently rolling hills with the some steeper areas. The elevation ranges from about 600 m above sea level (masl) to about 850 masl.

The property is poorly drained and includes swampy areas and lakes. The vegetation consists of a typical boreal forest of stunted spruce, alders, and willows.

3.3 Climate and Length of Operating Season

The climate is sub-arctic with temperatures ranging from -40°C in winter to 25°C in summer. The average annual temperature is -3.6°C and the average total precipitation is 880 mm. Mines in the area operate year-round with occasional shut-downs due to weather conditions.

3.4 Sufficiency of Surface Rights

Bloom Lake GPL controls sufficient surface rights for the process plant, waste dumps, tailings ponds and other infrastructure necessary for mining operations.

3.5 Local Infrastructure and Resources

3.5.1 Housing and Services

The town of Fermont has a population of approximately 4,000 and is a town originally built by QCM (now ArcelorMittal) for employees who work at their Mont-Wright operations.

Today, the town is mostly inhabited by ArcelorMittal employees who occupy 90% of permanent dwellings. Bloom Lake employees also reside in Fermont. The Project has constructed additional houses and a Hotel complex in Fermont to accommodate its permanent employees, whereas temporary employees will be housed in the company's camp.

ArcelorMittal’s Mont-Wright iron ore mining operation is also planning an expansion in 2013 with 900 new employees which will increase the size of the community.

Fermont has schools, a 72-room hotel, municipal and recreational facilities, a business and shopping complex and provides basic services for the small community.

Regarding socio-economic changes in the Fermont municipality, including social infrastructure availability, Bloom Lake is preparing to manage potential socio-economic impacts. The company undertook studies to evaluate the mine impact on the community (this study will be submitted to the Ministry of Sustainable Development, Environment and Parks (MSDEP) and will be updated every five years). This allowed for the identification of the main issues and the recommendation for the development of a socio-economic strategy including health and housing infrastructure, employment planning and road security program and criminality monitoring program (Cliffs, 2011a).

It is important to note that this mine is considered a Fly-In and Fly-Out operation.

3.5.2 Power and Water

There is grid electrical power supplied by Hydro- Québec (HQ) available nearby, a plentiful supply of water and ample surface area available for constructing mining and processing infrastructure.

3.5.3 Product Handling

The rail network consists of three separate segments. The first segment, 31.9 km long, is the property of Bloom Lake GPL and is located in Labrador. It connects to the Quebec North Shore and Labrador (QNS&L) railway at the Wabush Mines facilities in Wabush, Labrador.

The second segment uses the QNS&L railway from Wabush to Arnaud junction in Sept-Îles and from there, the third section is from Arnaud junction to Pointe-Noire (Sept-Îles), property of ‘’Les Chemins de Fer Arnaud’’, Sept-Îles, Quebec, where the concentrate will be unloaded, stockpiled or loaded to vessels.

The port facility is located at Pointe-Noire, Sept-Îles. All material handling and stockpiling equipment is installed on land leased on a long term basis from the Port of Sept-Îles Authorities.

Infrastructure is further discussed in Section 16.

4 History

4.1 Prior Ownership and Exploration

"In 1951, following the discovery of a cobalt showing at Bloom Lake, James and Michael Walsh staked claims for Mr. Bill Crawford of Sursho Mining Corporation (SMC). In February 1952, Quebec Cobalt and Exploration Limited (QUECO) was incorporated to acquire the claims held by SMC."

"In 1952, a crew of six prospectors under the supervision of Mr. K. M. Brown began a program to prospect an area that included the Bloom lake property. In June 1952, Mr. R. Cunningham, a mining geologist with Québec Metallurgical Industries, began to map the various cobalt occurrences at Bloom Lake. Although the results for cobalt were disappointing, several zones of magnetite-hematite iron formation (IF) were identified between Bloom Lake and Lac Pignac and were sampled. Further exploration was conducted in 1953."

In 1954, Cunningham supervised a program to investigate the iron occurrences through line cutting, geological mapping, and magnetometer surveys. "In 1955, Jones and Laughlin Steel Corporation (J&L) optioned the property from QUECO. Cleveland-Cliffs Iron Company (CCIC) joined with J&L and conducted a diamond drill program from 1956 through 1957. Two drills were brought to the property and two series of holes, the "QC" and the "X" series, were drilled to test IF on the Bloom Lake property. Holes X-1 to X-11 (XRT - ¾" diameter core) amounted to 446 m and Holes QC-1 to QC-30 (AXT size 1.28" diameter core) totaled 4,769 m. The holes were largely drilled on sections 800 to 1,000 ft apart (244 to 305 m). Four of these drillholes were drilled on the west part of the property."

More drilling was conducted in 1966 by Boulder Lake Mines Incorporated, a subsidiary of CCIC, and Jalore Mining Company Limited (Jalore), a subsidiary of J&L. Holes X-12 to 20, totaling 175 m, and other holes were drilled as part of this campaign, but these were not on the present property. Some ground magnetometer surveying was also conducted in 1966. J&L's option on the property terminated in 1968

"In 1971, exploration on the property was renewed by a QUECO-sponsored program that was managed by H. E. Neal & Associates Ltd. (HEN). The exploration program consisted of line cutting, geological mapping, gravity and magnetometer surveys, and diamond drilling in 1971 and 1972. These holes were drilled to investigate the potential for IF beneath the amphibolite on the eastern side of the property. Nine drillholes were done in 1971 for a total of 1,834.23 m (341 samples) and 12 were drilled in 1972 (3,497.79 m and 341 samples). Eight of the drillholes were done on Bloom Lake West in 1971 and five were drilled in 1972. The mapping and magnetometer surveys were designed to fill in areas not previously surveyed. The gravity survey was conducted to help evaluate the potential for IF beneath the amphibolite."

In 1973, Republic Steel Corporation optioned the property and HEN prepared a "Preliminary Evaluation" of the property that consisted of currently held property and claims further to the west. This work was conducted until 1976. The evaluation included "mineral reserve" estimates, a metallurgical test program, and preliminary mine design. The mine design included pit outline, dump area, access roads, and railway spur. Dames and Moore prepared the mine design and "reserve" estimates. Lakefield Research (Lakefield) conducted the metallurgical test work.

In 1998, a major exploration program was conducted by Watts, Griffis and McOuat (WGM) for QCM, which then held the Bloom Lake property under option from Consolidated Thompson-Lundmark Gold Mines Limited (CLM). QCM held the option on the property until 2001, but no work was conducted between 1998 and 2005. The 1998 program included line cutting, surveying, road building, camp construction, diamond drilling, geological mapping, mini-bulk sampling, bench-scale preliminary metallurgical test work, preparation of a “mineral resource” estimate, camp demobilization, and site clean-up.

In 2005, CLM retained WGM to conduct a technical review, including the preparation of a mineral resource estimate for the Bloom Lake iron deposit to assist CLM in making business decisions and future planning. The technical review was prepared in compliance with the standards of NI 43-101 in terms of structure and content. The mineral resource estimate was prepared in accordance with NI 43-101 guidelines and CIM standards (Genivar, 2009). In 2006, Consolidated Thompson-Lundmark Gold Mines Limited changed the name of the Company to Consolidated Thompson Iron Mines Limited. This name change reflected the Company's focus on iron ore mining and exploration.

In 2006 to 2007, CLM drilled 17 drillholes (2,884.36 m) on the site of the future pit in order to get a sample for metallurgical test work. The Lakefield laboratory performed these tests. In 2006, bulk sampling took place in the area of the future pit.

Between 2005 and 2010, CLM drilled 198 drillholes at Bloom Lake.

Cliffs acquired CLM in May 2011.

4.2 Historic Mineral Resource and Reserve Estimates

WGM prepared a resource estimation for the area east of 614500 E in 1998 as part of its exploration program for CLM. WGM produced cross sections and level plans and created a block model with geology and grades for selected elements and used this model to determine the tonnage and grade of the deposit at various cut-off grades. The Bloom Lake Mineral Resource classification was based on drillhole and surface sample density and confidence in mineralization continuity. Drillhole spacing was nominally at 150 m on and between sections. A 15% Fe cut-off was used and was based on that employed at operating mines in the Labrador Trough exploiting deposits similar to Bloom Lake (CIMA, 2010). In 2005, WGM reviewed its estimation procedures and determined that they met NI 43-101 guidelines which came into effect in 2002. Table 4.2.1 lists the resources estimated by WGM in 1998 and verified by them in 2005. The resources are not constrained by a pit shell as it is in SRK’s 2011 estimate. The resources are the basis of the reserves stated in 2008.

Table 4.2.1: Historical (WGM 2005) Mineral Resources at 15% Cut-off Total Fe (east of 614500)

Category	Tonnage (kt)	Average Grades (%)			
		Total Fe	Magnetite	CaO	MgO
Measured	488,465	29.9	10.5	2.3	2.2
Indicated	149,232	29.3	10.6	2.4	2.2
Total Measured + Indicated	**637,697**	**29.8**	**10.5**	**2.3**	**2.2**
Inferred	35,697	31.0	8.5	0.8	0.8

In 2009, Genivar prepared a resource estimation for Bloom Lake West, between 612500 E and 614000 E. These resources are separate from the WGM resources and are stated in Table 4.2.2; no cut-off grade was given in Genivar (2009) for the resource tabulation.

Table 4.2.2: Historical (Genivar 2009) Mineral Resources at 15% Cut-off Total Fe (east of 614500)

Category	Tonnage (kt)	Average Grades (%)		
		Total Fe	CaO	MgO
Indicated	189,230	27.94	0.05	0.05
Inferred	47,250	29.32	0.04	0.06

In 2008, CLM and BBA Inc. prepared a reserve estimate based on 8 Mt/y of concentrate (Table 4.2.3) and a pit with a 20 year mine life. The reserves are tabulated at a 15% total Fe% cut-off. These reserves were used as the basis of the CIMA 2010 feasibility study for the increase in production from 8 Mt/y to 16 Mt/y.

Table 4.2.3: Historical (2008) Total Mineral Reserves at 15% Cut-off Total Fe

Category	Mt	Weight Recovery	Average Grades (%)				
			Total Fe	MagFe	CaO	MgO	Magnetite
Proven	463.4	38.3	30.1	7.6	2.2	2.1	10.5
Probable	116.2	37.7	29.7	7.7	2.3	2.4	10.7
Total Ore	**579.6**	**38.2**	**30.0**	**7.6**	**2.3**	**2.1**	**10.5**
Total Waste	**563.8**						

Source: CLM and BBA, 2008
Stripping Ratio 0.97

In 2011, SRK prepared an updated resource for both the main and west pit areas (Table 4.2.4) and a reserve estimate based on 2 mill operation (Table 4.2.5) and a pit with a 26 year mine life. The resources were calculated at a 15% total Fe% Cut-off and were constrained by an economic pit shell at a price of $120/t. The reserves are tabulated at a 20% total Fe% cut-off and represented a fully designed life-of-mine pit and operating schedule. These reserves were used as the reserve basis for filing in Cliffs annual 10-K report.

Table 4.2.4: Historical (SRK 2011) Mineral Resources at 15% Cut-off Total Fe

Category	Tonnage (kt)	Average Grades (%)			
		Total Fe	MagFe	CaO	MgO
Measured	296,700	29.1	5.4	1.7	1.7
Indicated	866,500	27.6	6.1	2.4	2.1
Total Measured + Indicated	**1,163,200**	**28.0**	**6.0**	**2.3**	**2.2**
Inferred	265,500	25.7	10.9	2.7	2.3

Table 4.2.5: Historical (SRK 2011) Mineral Reserves at 20% Cut-off Total Fe

Classification	Tonnes	FeT	Mass Recovery	CAO	MgO	Mag Fe	Magnetite	Concentrate
	Mt	%	%	%	%	%	%	Mt
Proven	286.0	29.4	35.5	1.6	1.6	5.4	7.5	101.5
Probable	765.3	28.3	33.9	2.3	2.0	6.0	8.3	259.6
Proven + Probable	1,051.3	28.6	34.4	2.1	1.9	5.8	8.0	361.1

1.402.7 Mt of waste was extracted with the Mineral Reserves. Strip ratio was calculated to be 1.33 (waste to ore);
Reserves based on December 31, 2011 topography;

4.3 Historic Production

Tables 4.3.1 through 4.3.3 show production from 2010 through 2012, respectively.

Table 4.3.1: 2010 Production

Month	Waste Tons	Crushed Ore Tons	% Total FE Crude	Concentrate Tons	Conc % FE	% Moisture
January	-	-	0.0	-	0.0	0.0
February	373,000	65,000	8.7	-	0.0	0.0
March	606,000	91,000	12.8	-	0.0	0.0
April	341,000	590,000	33.0	123,460	65.9	5.5
May	483,900	727,000	31.8	244,515	65.7	5.5
June	713,600	1,053,000	31.2	331,885	66.3	5.5
July	962,542	1,229,073	31.9	372,961	66.0	5.8
August	887,736	1,579,685	33.3	480,225	65.9	5.5
September	1,142,475	1,038,546	33.2	375,420	65.9	4.3
October	510,000	1,579,000	33.5	488,205	65.7	3.2
November	977,000	1,059,326	31.9	337,776	64.2	2.2
December	1,063,054	1,243,284	34.2	411,850	65.7	2.4
Total	**8,060,307**	**10,254,914**	**32.4**	**3,166,297**	**65.7**	**4.3**

Table 4.3.2: 2011 Production

Month	Waste Tons	Crushed Ore Tons	% Total FE Crude	Concentrate Tons	Conc % FE	% Moisture
January	893,061	1,259,225	33.9	403,665	65.7	2.3
February	1,055,648	1,107,893	32.2	366,525	65.6	2.5
March	704,666	1,567,819	33.6	510,704	65.6	2.4
April	815,471	1,572,985	32.7	553,697	65.4	2.9
May	1,005,179	1,493,448	33.5	572,970	65.6	3.8
June	746,817	1,691,675	34.5	593,715	65.5	5.2
July	1,403,015	1,242,832	34.3	432,766	65.7	4.7
August	838,582	1,698,612	34.1	592,380	65.7	4.7
September	1,447,246	1,511,568	33.4	456,990	66.1	4.7
October	2,160,229	987,636	34.1	339,885	66.2	4.2
November	1,591,924	1,100,006	30.8	334,395	66.1	3.0
December	1,312,114	1,397,322	29.4	458,775	66.1	2.9
Totals	**13,973,952**	**16,631,021**	**33.1**	**5,616,467**	**65.8**	**3.7**

Table 4.3.3: 2012 Production

Month	Waste Tons	Crushed Ore Tons	% Total FE Crude	Concentrate Tons	Conc % FE	% Moisture
January	1,431,329	1,338,190	30.8	500,325	66.2	2.8
February	1,602,365	1,120,801	31.7	266,325	66.3	2.5
March	794,000	1,366,606	31.5	552,255	66.4	2.4
April	1,097,728	1,252,264	31.6	395,355	66.4	2.7
May	1,459,714	1,159,352	32.1	460,500	66.3	3.8
June	1,670,576	1,258,907	33.1	377,280	66.4	4.1
July	1,323,401	1,392,871	32.3	495,780	66.3	3.9
August	1,496,873	1,384,677	31.8	496,620	66.1	3.7
September	1,611,327	1,281,791	32.0	475,410	66.1	4.0
October	1,481,800	1,455,518	31.3	542,640	66.0	3.8
November	1,520,052	1,323,178	31.4	381,072	66.2	3.4
December	1,470,308	1,550,746	32.8	622,580	66.2	3.1
Totals	**16,959,473**	**15,884,901**	**31.9%**	**5,566,142**	**66.2%**	**3.4%**

5 Geological Setting and Mineralization

5.1 Regional Geology

As illustrated in Figure 5.1.1, the Project area is situated in the southwest portion of the Labrador Trough, a northwest-trending geosyncline also referred to as the Labrador-Quebec fold belt and associated with the New Quebec Orogen. The width of the belt swells to 75 km in places and stretches for over 1,200 km through Northern Quebec and Labrador. To the north, the belt extends as far as Ungava Bay off the Hudson Straight, to the south it extends as far as Lake Pletipi, Quebec.

Source: Cliffs

Figure 5.1.1: The Labrador-Quebec Fold Belt (Labrador Trough) and Bloom Lake Location

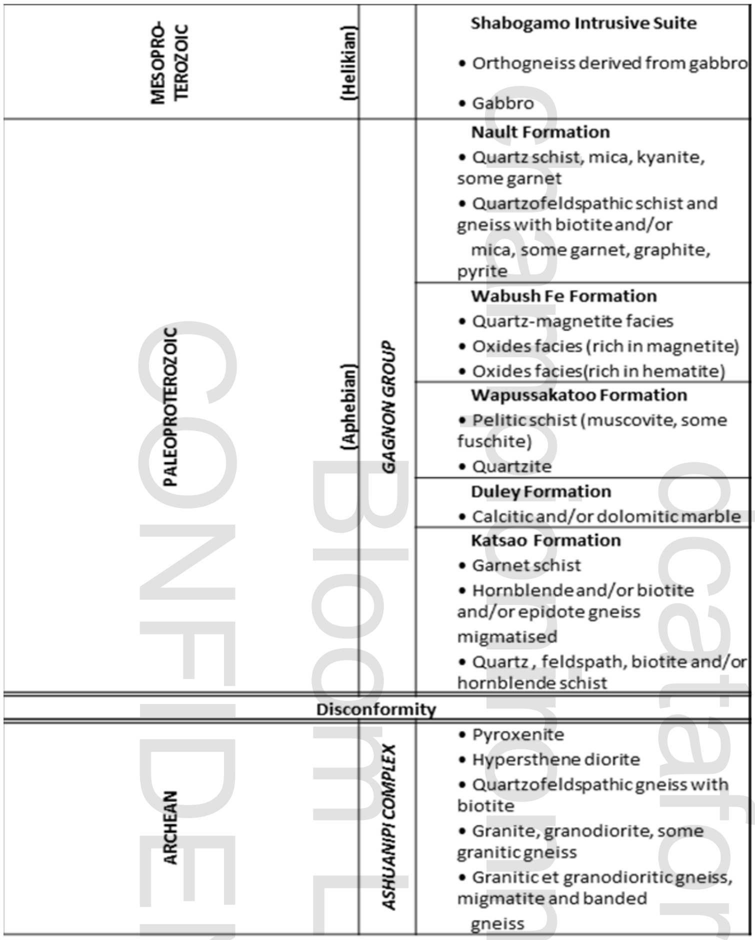
In the vicinity of the Project area, the Labrador Trough is located proximal to the intersection of three major geologic provinces. The belt itself is part of the Churchill Province. The western margin of the belt unconformably overlies the Archean basement rocks of the Superior Craton and the southern portion of the belt is cut-off from the north by the Grenville Front, the northern limit of the younger Grenville Province (www.nr.gov.nl.ca/nr/mines/). This is illustrated in Figure 5.1.2.

The Labrador Trough and accompanying iron formations are composed of folded Lower Proterozoic sedimentary, volcanic and metavolcanic rocks, which were deposited as sediments and lavas 2.17 to 1.87 Ga in interconnected sub-basins. The Lower Proterozoic rocks overlie gneisses of the Superior Craton and include a thick succession of thin-banded, fine-grained clastic sediments, argillite and slate, which is transitional upward to dolomite and chert breccia in local basins (Gross, 2009).

As illustrated in Figure 5.1.1, the southern portion of the Labrador Trough is cut-off from the rest of the belt by the Grenville Front, the northern limit of the Grenville Province. This boundary separates the Churchill Province and associated greenschist metamorphic grade rocks to the north from the highly metamorphosed and folded amphibolite to granulite metamorphic grade rocks of the Grenville Province to the south. The Bloom Lake deposit is located in the Labrador Trough south of the Grenville Front (Genivar, 2009).

The Proterozoic rocks of the Labrador Trough are part of the Kaniapiskau Supergroup. These are the Gagnon Group, also referred to as the Gagnon terrane, within the Grenville Province. Originally interpreted to have been a continuous belt of iron formations, the Gagnon terrane is now a NE-SW trending, 300 km belt of dismembered, folded and oval shaped iron ore bodies. These metasedimentary sequences host the Wabush-Lac Jeannine iron ore belt which includes the Bloom Lake Project (Corriveau and Perreault, 2005).

Within the Labrador Trough, north of the Grenville Front, most of the iron formations are found within the Sokoman Formation. South of the Grenville Front, within the Gagnon terrane, the metamorphosed equivalent of the Sokoman is the Wabush Formation. The Wabush Formation is typically subdivided into Lower, Middle, and Upper members. The Lower member is composed of carbonate-silicate facies with minor oxides, the Middle member is composed of oxide facies with specular hematite, minor magnetite and sugary textured quartz and the Upper member is composed of carbonate-silicate facies with minor oxides. The Middle member is considered the most economically important and includes the bulk of the quartz-magnetite-specular hematite ores that are mined in the area (Corriveau and Perreault, 2005). The stratigraphic column is presented in Figure 5.1.2.



Modified from Genivar, 2009

Figure 5.1.2: Stratigraphic Column of the Bloom Lake Project

5.1.1 Gagnon Group (Gagnon Terrane)

The Gagnon Group encompasses the primarily chemically deposited metasedimentary formations of the Kaniapiskau Supergroup that have been highly metamorphosed and complexly folded by the Grenville orogeny south of the Grenville Front. Gagnon Group rocks extend for a distance of over 270 km from north of Wabush Lake to Matonipi Lake, the town of Gagnon, Quebec, is located near the middle of the belt and was built to exploit the nearby iron deposits. In the Project vicinity the Gagnon Group is made up of, in stratigraphic order from top to bottom, the Wabush Lake Iron Formation, the Wapussakatoo Formation and the Duley Formation. The Duley and Wapussakatoo Formations typically underlie the Wabush Lake Iron Formation, but are known to overlie it in several places (Clarke, 1977).

The Duley Formation

The Duley Formation is most commonly a white, buff-weathering marble composed of 2 to 5 mm grains of white dolomite and thin layers of quartz and calc-silicate minerals. The majority of the marble has a composition of 80% dolomite, 10% calc-silicates and 10% quartz (Clarke, 1977).

The Wapussakatoo Formation

The Wapussakatoo Formation is a medium grained, light grey to milky white quartzite with relic bedding. The bedding is seen in outcrops and is presented in the form of faint color bands in places and a strong joint set in places. The bedding seen in thin-section is presented as alternating bands of coarse and fine grained quartz (Clarke, 1977).

Different varieties of the quartzite exist, including micaceous and schistose varieties, as well as a coarse crystalline variety resembling vein quartz. The pure quartzite variety contains at least 95% quartz with minor amounts of mica, calcite, hematite, cummingtonite, zircon, tourmaline, and apatite. The muscovite-quartzite variety contains approximately 70% quartz, 20% muscovite, 5% microline, and minor amounts of specularite, kyanite and green tourmaline (Clarke, 1977).

The Wabush Lake Iron Formation

The Wabush Lake Iron Formation is present throughout most of the area but composition varies depending on the local oxidation potential. In areas in which the sea bottom received abundant oxygen; chert and ferric iron were deposited, in areas that received less oxygen, considered reducing environments; the iron remained in the ferrous state. The chert and ferric iron of the oxygen-rich environments were later metamorphosed to quartz-specularite iron formation and the ferrous-state iron of the reducing environments now occurs as magnetite and siderite or iron silicates (Clarke, 1977).

According to the differing degrees of Grenville metamorphism, multiple different mineral assemblages occur. Most of the metamorphic changes occur in the silicate-carbonate iron formation, with the quartz and iron oxides being stable through the metamorphic range. The primary constituents of the silicate-carbonate iron formation include quartz, magnetite, iron-rich silicates and carbonate. Both the composition and the appearance of these rocks change throughout the area due to changing metamorphic grade (Clarke, 1977).

The most economically important variety of the iron formation is the oxide facies, which resists erosion and results in outcropping of most of the large iron deposits. Depending on the oxygen state of the original sediment, the iron may occur as either blue-grey specular hematite or as magnetite. The quartz hematite facies is more abundant than the quartz magnetite facies, which is gradational between the quartz hematite and the silicate carbonate facies. Mixed quartz and iron oxide interlayered with grey glassy quartz make up most outcrops. The iron formation generally averages 30% iron oxides and ranges between 10 and 45% (Clarke, 1977).

5.2 Local Geology

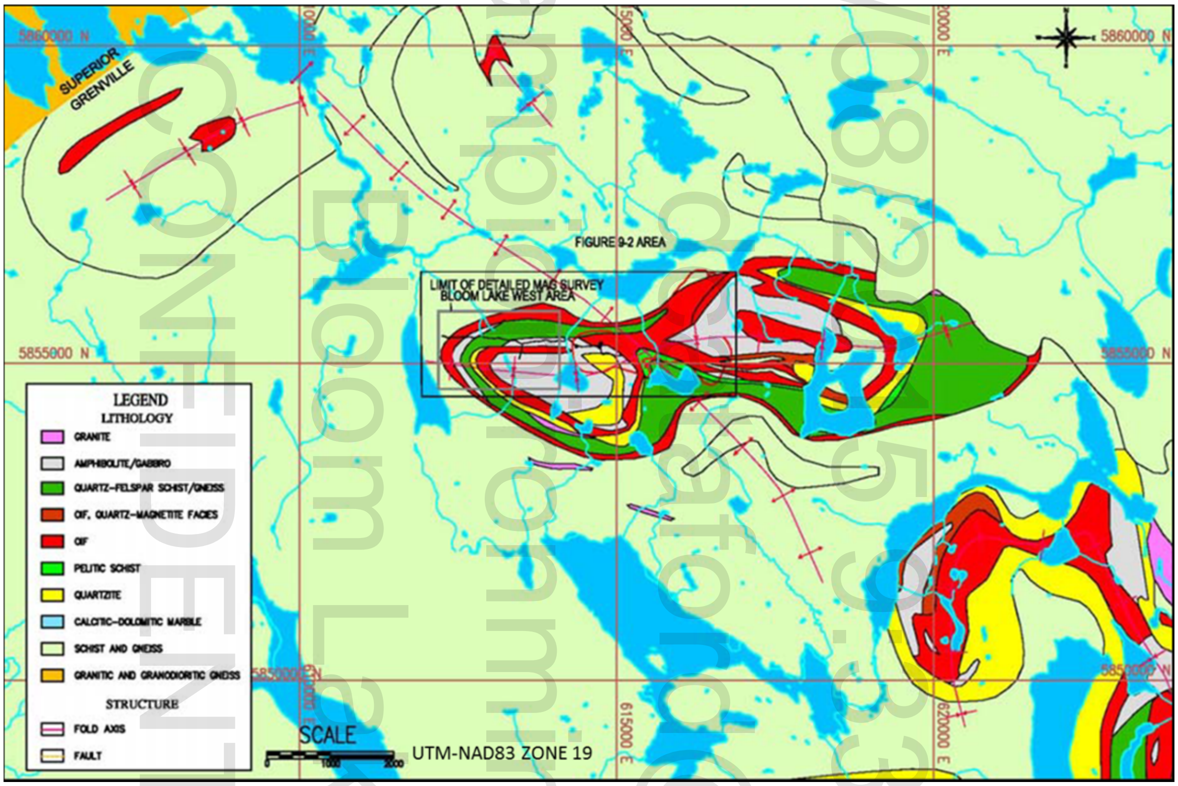
Multiple sources dating as far back as 1957 contribute to the geological understanding and interpretation of the Bloom Lake Property. Geologic data and information is available from numerous drilling and magnetic surveys as well as several mapping programs.

Local geology is described according to seven rock types. For practicality, several rock type codes are codes for hybrids of the main rock types and the accompanying individual rock types are not described separately. The local geology is described according to the following rock classifications:

- Gneiss (GN);
- Quartz Rock (QR) and its Related Variant Quartz Rock Iron Formation (QRIF);

- Quartz Rock Mica Schist (QRMS);
- Silicate Iron Formation (SIF);
- Amphibolite (AMP);
- Gabbro; and
- Oxide Iron Formation (OIF).

A geologic map of the Project area is presented in Figure 5.2.1.



Modified from Quebec Natural Resources and Fauna Ministry, 1998

Figure 5.2.1: Regional Geologic Map of the Bloom Lake Project Showing Local Folds

Gneiss (GN)

Current geological understanding is that the basic meta-sedimentary rock unit is gneiss. The majority of its composition is mafic with a minor quantity of felsic bands dominated by feldspaths with quartz. Banding is typical throughout on a scale of 1 to 2 m. Biotite is present throughout the gneiss and commonly transitions to mica schist. Compared to QRMS, the gneiss contains more feldspar and quartz but less mica (Genivar, 2009).

Quartz Rock (QR) and its Related Variant Quartz Rock Iron Formation (QRIF)

The composition of QR is greater than 95% quartz and has a vitreous luster, common color is either grey or pinkish, and specularite and/or magnetite content ranges from zero to minimal. The origin of QR is unknown but may have derived from either chert, quartzite, or a quartz pebble conglomerate. Multiple varieties of texture are present but have not been differentiated (Genivar, 2009).

The QRIF classification is determined according to dominant rock type, percentage of iron and associated specularite and/or magnetite or silicate. This rock is dominantly quartz with less than 15% total iron and contains at least some specularite and /or magnetite or silicate, also minor actinolite-SIF may be present. The QRIF is considered transitional between OIF and QR or SIF and QR (Genivar, 2009).

The QR and QRIF transition into the basal schist or gneiss unit and commonly contain muscovite, garnet, and feldspar. The two related quartz units consist of various sequences throughout the IF sequence and a unit located at the structural base of the IF directly above the basal mica schist (CIMA, 2010).

Quartz Rock Mica Schist (QRMS)

Composition of QRMS consists primarily of muscovite and biotite schist with characteristic porphyroblasts of garnet and feldspar. It is used to describe the schist sequence located at the base of the IF sequence under the QR unit. Occasionally it is also used for coding of the biotite-rich units within the IF sequence, which are described in AMP (Genivar, 2009). A transition into gneiss (GN) occurs at the base of the QRMS rocks. These GN rocks are composed of more quartz and feldspar and less mica than the QRMS rocks (CIMA, 2010).

Silicate Iron Formation (SIF)

The SIF unit is divided into two subunits within the property; one composed predominantly of actinolite and the other composed predominantly of grunerite. The two subunits commonly transition into one another and there is likely some tremolite-SIF present (Genivar, 2009). As the amount of actinolite increase in the IF, color becomes distinctly greener. This color change has been used as a tool for identifying SIF during core logging (CIMA, 2010).

Actinolite-SIF occasionally occurs on the margins of the thinner amphibolite units near the bottom of the stratigraphic sequence and also on the IF, QRIF or QR contacts. In these areas the IF is commonly enriched with magnetite instead of specularite (Genivar, 2009).

Amphibolite (AMP)

The primary constituents in the AMP are hornblende, biotite, and feldspar. It is dominantly a competent rock that is medium to coarse grained and dark green to black in color. A wide variety of grain sizes exist, but it's relatively homogenous throughout with a very pronounced foliation. Millimeter-scale reddish garnets are present and can be observed over tens of meters (Genivar, 2009).

Sharp contacts define the amphibolite-IF unit and a narrow zone of argillic alteration is common above the IF contact (Genivar, 2009). Both a large, massive body and thinner sequences of amphibolite can be found throughout the IF sequence. The thinner sequences tend to occur towards the bottom of the IF sequence (CIMA, 2010).

Gabbro

Medium grained bodies of gabbro and amphibolite form prominent hills within the quartz-bearing units of the Gagnon Group. During periods of deformation they were injected into the surrounding competent rocks, and then subsequently remobilized during later episodes of metamorphism. In the northern parts of the injected zone gabbro is more common, in the southern parts is amphibolite is more common. Gabbro cores in thick amphibolite sills can occasionally be found. This type of gabbro typically contains 40–50% plagioclase with additional mafic minerals such as olivine and hypersthene and a few percent opaque oxides (Genivar, 2009).

Oxide Iron Formation (OIF)

Sequences rich in specular hematite and magnetite are considered OIF, and can be further categorized as specularite-rich OIF (SOIF) and magnetite-rich OIF (MOIF). These sequences commonly comprise 25% to 40% iron oxide and with thicknesses of up to 200 m are considered the most economically important. Both the specularite and the magnetite commonly occur in anastomosing to discontinuous stringers and bands. The bands are less than 10 cm thick and occur in a quartz rock or actinolite-quartz rock matrix. Typical banding is folded and deformed but regular and tabular banding does exist. In contrast to the bands, intervals of core have been drilled containing near-massive magnetite of up to a couple meters in thickness, this is however uncommon (CIMA, 2010).

Like the magnetite, specularite commonly occurs in bands, but tends to be coarser grained. The specularite is silvery-grey in color and non-magnetic. The SOIF sequence ranges in thickness and can be up to 150 m, but also thins along with the entire sequence near the edges of the deposit (CIMA, 2010).

The MOIF most commonly, but not always, occurs structurally below the main mass of amphibolite or directly under thin intervals of grunerite-SIF. This sequence of MOIF usually ranges between 30 to 50 m in thickness. Magnetite is continually replaced with specularite down sequence until little or no magnetite is present (CIMA, 2010).

5.2.1 Structural Geology

Regional structural history includes multiple episodes of tectonic activity and accompanying deformation. Structural deformation related to the Labrador-Quebec fold belt includes isoclinal folding and the formation of imbricate structures from two separate magmatic events. In addition to these structural deformations, the region also went through another order of folding and deformation related to the Greville Orogeny (Gross, 2009).

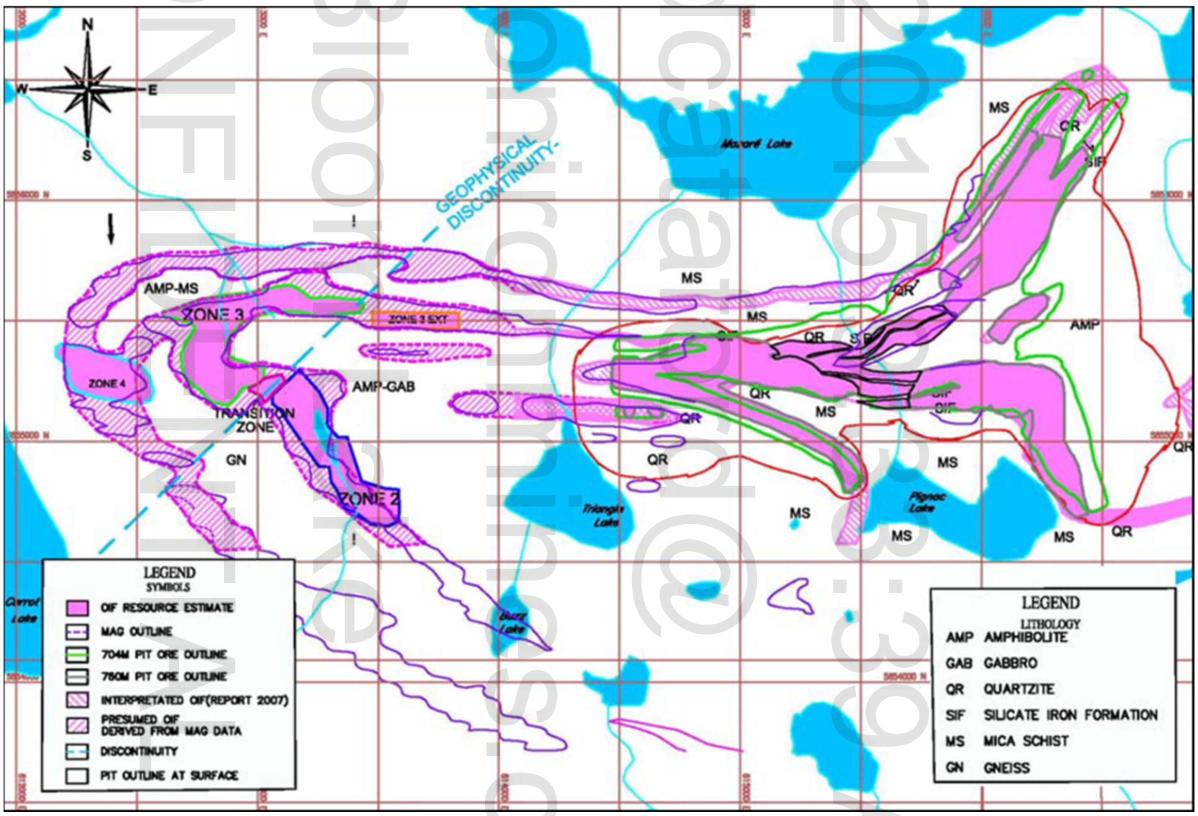
Within the Project area two east-west, mildly plunging synforms separated by a northwest, mildly plunging antiform define the structural geology. The axis of the northwest plunging antiform is approximately centered through Lac Pignac, the axis of the western synforms is approximately centered through Triangle Lake and the axis of the eastern synforms is approximately centered through the north limb of Bloom Lake. The Project area is centered east of the antiform and north of the eastern synform just southwest of Lac Mazaré (Watts, Griffis, and McOuat, 2005).

As shown in Figure 5.2.1, these synforms are of regional scale and are the result of at least two separate episodes of folding. Project area structural geology has been defined by these regional scale folds and associated project-scale synforms as well as multiple other folds of various orientations. It is unknown if the various fold orientations are the result of multiple folding episodes or of changing fold orientations over time (Watts, Griffis, and McOuat, 2005).

On a Project-scale, the iron formation sequence does not appear to be duplicated, repeated, or truncated due to isoclinal folding. Although on a local scale, reversals, repeats, and stratigraphy transposition of the iron formation due to folding events are common. As seen in drill core, the complex folding results in the iron formation having a lensitic, crenulated and stringer nature (Watts, Griffis, and McOuat, 2005).

Locally, the sedimentary rocks and mineralized iron formations are structurally controlled by two doubly plunging, east-west synforms and a gently north to northwest plunging antiform. Multiple other synforms of various orientations on the property have also affected the sequence (Watts, Griffis, and McOuat, 2005).

Although faulting has made less of an impact on the structural geology than folding (Watts, Griffis, and McOuat, 2005), ground magnetic surveying has identified a prominent discontinuity in the western portion of the Project, as shown in Figure 5.2.1.1. Numerous evidence of a significant fault in the area is also apparent in drill core as gravels, gouges and brecciated zones. Owing to the scale of the structure, there has been difficulty in correlating the geology on either side of the discontinuity (Genivar, 2009).

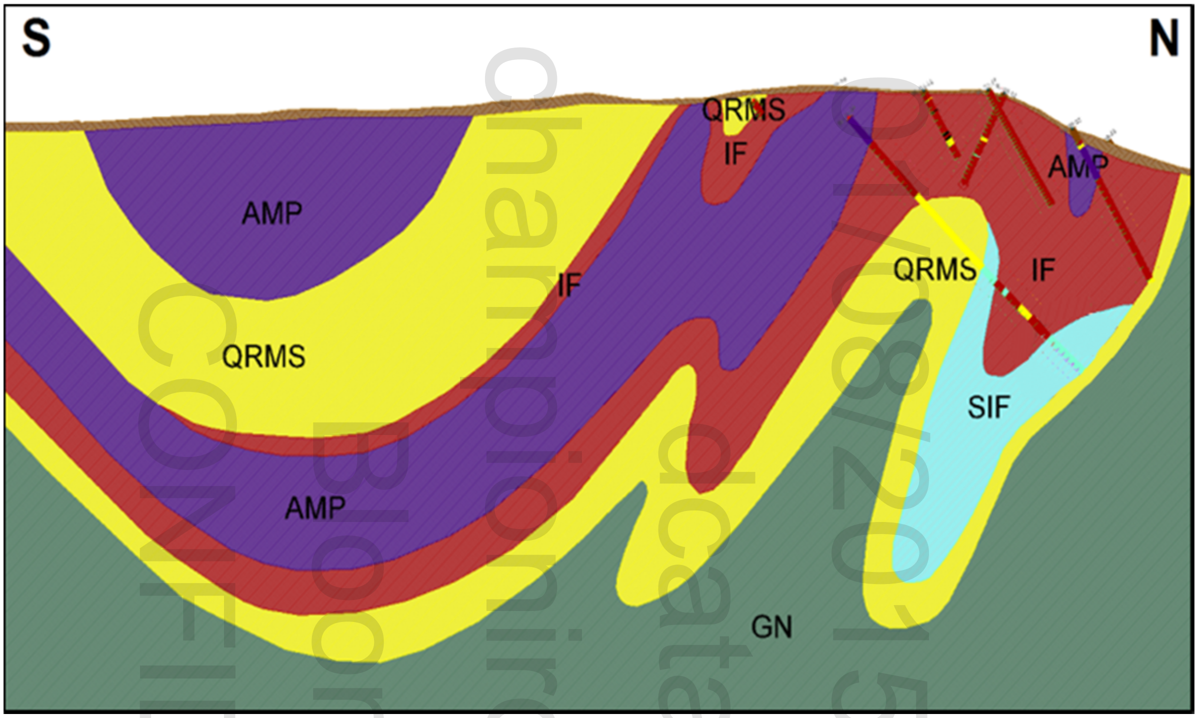


Modified from Genivar, 2009

Figure 5.2.1.1: Map Showing Geology and Magnetic Survey Outlines of Bloom Lake Project

Corresponding with this discontinuity, a north-northeast orientation of geomorphic lineaments is evident in aerial photographs. In addition to the north-northeast discontinuity pattern, there also exists a north-northwest discontinuity pattern. This is suggested by the linear attitude of Triangle Lake and the associated stream configuration. This basic interpretation needs to be verified before it can be used as the basis for detailed geologic interpretations (Genivar, 2009).

Figure 5.2.1.2 is a typical cross–section showing the folded nature of the Bloom Lake Project.



Modified from Gemcom (2011)

Figure 5.2.1.2: Section 614950E, Looking West, Showing Synform Structure of the Bloom Lake Project

5.3 Mineralized Zones

Bloom Lake mineralization is confined to the interlayers of iron formation between Proterozoic aged sedimentary rocks, typical of Lake Superior-type deposits. Mineralization resulted from chemical precipitation of iron minerals and quartz from the sea water along continental margins. This is discussed in detail in Section 6, Deposit Type. The sedimentary rocks and accompanying mineralized iron formation interlayers were subsequently highly metamorphosed and intensely folded with the Grenville Orogeny (Watts, Griffis, and McOuat, 2005).

Locally, structural controls are important to mineralization. The two doubly plunging, east-west synforms and the gently north to northwest plunging antiform as well as multiple other synforms of various orientations have resulted in the mineralized iron formations having tabular to folded and anastomosing bands of economically important minerals (Watts, Griffis, and McOuat, 2005).

Banding in the iron formation is defined by accumulations of specularite, magnetite and intercalated actinolite in a dominantly quartz matrix. Specularite-rich sequences with minor intercalated actinolite dominate the iron formations in the west end of the property and in the east and north parts actinolite-rich sections are more uniformly interwoven at a fine scale with magnetite and/or speculate (CIMA, 2010).

The OIF sequences are considered the most economically important. They are rich in specular hematite (specularite) and magnetite and comprise 25 to 40% iron oxide with the remainder primarily composed of quartz. These sequences include minor intercalated bands of schist and quartz rock (Watts, Griffis, and McOuat, 2005) and are known to exist over several hundreds of meters laterally (Genivar, 2009). Thicknesses of up to 200 m have been observed (Watts, Griffis, and McOuat, 2005).

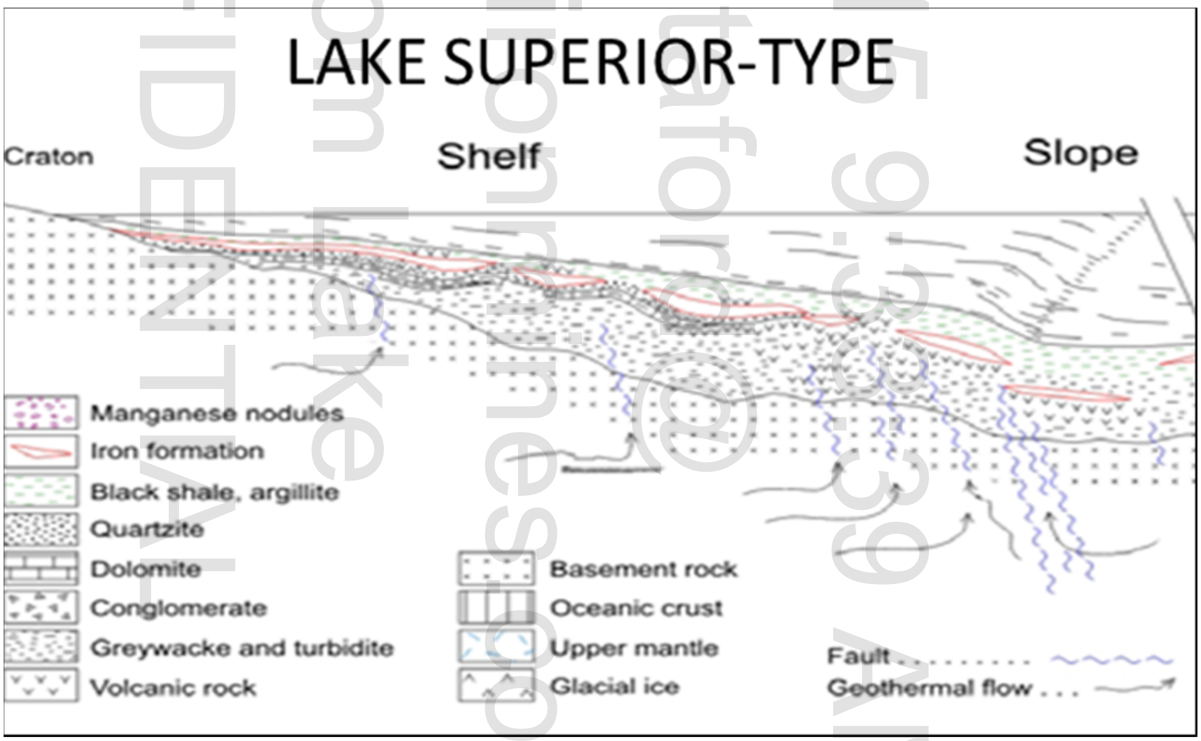
The OIF sequences can be further subdivided according to the main constituent; either specularite-rich OIF (SOIF) or magnetite-rich OIF (MOIF). Both the magnetite and specularite tend to occur in bands and both have an average specific gravity (SG) of 3.47, but the specularite is coarser grained and non-magnetic. The MOIF sequence is commonly 30 to 50 m in thickness and the SOIF is commonly up to 100 to 150 m in thickness. The MOIF is usually, but not always, located structurally below the main mass of amphibolite and the SOIF is located between the amphibolite cap-related magnetite OIF sequence and the characteristic actinolite-SIF sequence (Watts, Griffis, and McOuat, 2005).

6 Deposit Type

The Bloom Lake deposit is a typical Lake-Superior-type iron formation deposit. Such iron formations are found on all continents. Lake Superior-type iron formations developed along the margins or cratons between 2.4 Ga and 1.9 Ga in relatively stable sedimentary-tectonic systems (Gross, 2009). They are characterized by interlayers of sedimentary rocks composed primarily of bands of iron minerals such as iron oxides, magnetite and hematite with bands of quartz (chert) and can include variable amounts of silicate, carbonate and sulfide lithofacies (Watts, Griffis and McOuat, 2005).

At the time of iron formation deposition, sufficient oxygen had accumulated in seawater that dissolved iron from the previously weathered adjacent continent could be oxidized. The combination of oxygen and iron resulted in the formation of hematite, limonite and iron carbonate. The iron minerals then precipitated from the sea water along with quartz in varying proportions, thus producing the bands of Lake Superior-type banded iron formation (http://www.mngs.umn.edu/-cuyuna/lsif.html).

Figure 6.1 presents a cross-sectional view of a typical Lake Superior-type iron formation depositional environment.



Modified from Gross, 2009

Figure 6.1: Tectonic Environment of Lake Superior-Type Iron Formation Deposition

Lake Superior-type iron formations are a major source of iron throughout the world (Gross, 1996). Some well-known and typical Lake Superior-type iron formations include:

- The Kursk Group of Western Russia;
- The Cauê Formation of Minas Gerais State, Brazil;

- Accompanying formations of the Hamersley Basin of Western Australia;
- The Griquatown-Transvaal of Northern Cape Province, South Africa; and
- Accompanying formations of the Lake Superior region in the USA and Canada.

The ability to mine iron formation economically is constrained by the percentage of iron content, which must be greater than 30%, also the iron oxides must be amenable to concentration and the resulting concentrates must be low in manganese and deleterious elements such as silica, aluminum, phosphorus, sulfur and alkalis. For effective bulk mining, the rocks interbedded with the iron formation such as silicate and carbonate, must by adequately segregated from the magnetite. In the Mont-Wright/Wabush area deposits with repeated iron formations due to folding are often required in order to produce mining sections of sufficient thickness (Watts, Griffis and McOuat, 2005).

6.1 Mineral Deposit

Bloom Lake is an example of a meta-taconite deposit of the Lake-Superior-type. Taconite is defined as a lithofacies that has not been highly metamorphosed or altered from weathering and highly metamorphosed taconites are known as meta-taconite or itabirite. Meta-taconite deposits are typical in the Grenville part of the southern Labrador Trough in the vicinity of Fermont and Wabush (Watts, Griffis and McOuat, 2005).

6.2 Geological Model

The geological model used at Bloom Lake is typical of meta-taconite Lake-Superior-type deposits. The model is based on mineralization of banded iron formation interlayers between layers of quartz-rich, Proterozoic aged sedimentary rocks that have been highly metamorphosed and intensely folded. Mineralization resulted from chemical precipitation of iron minerals and quartz from the sea water along continental margins. The sedimentary rocks and accompanying mineralized iron formation interlayers were subsequently highly metamorphosed and intensely folded with the Grenville Orogeny (Watts, Griffis, and McOuat, 2005).

Locally, the sedimentary rocks and mineralized iron formations are structurally controlled by two doubly plunging, east-west synforms and a gently north to northwest plunging antiform. Multiple other synforms of various orientations on the property have also affected the sequence. The geologic structures have resulted in the mineralized iron formations having tabular to folded and anastomosing bands of economically important minerals (Watts, Griffis, and McOuat, 2005).

On a regional scale, the iron formation sequence does not appear to be duplicated, repeated, or truncated due to isoclinal folding. Although on a local scale, reversals, repeats, and stratigraphy transposition of the iron formation due to folding events are common. As seen in drill core, the complex folding results in the iron formation having a lensitic, crenulated and stringer nature (Watts, Griffis, and McOuat, 2005).

Banding in the iron formation is defined by accumulations of specularite, magnetite and intercalated actinolite in a dominantly quartz matrix. Specularite-rich sequences with minor intercalated actinolite dominate the iron formations in the west end of the property and in the east and north parts actinolite-rich sections are more uniformly interwoven at a fine scale with magnetite and/or speculate (CIMA, 2010).

The iron formation sequences rich in specular hematite (specularite) and magnetite are known as Oxide Iron Formation (OIF), typical OIF comprise 25-40% iron oxide with the remainder being primarily quartz (Watts, Griffis, and McOuat, 2005). These sequences are known to exist over several hundreds of meters laterally (Genivar, 2009) with thicknesses of up to 200 m (ignoring minor intercalated bands of schist and quartz rock). These OIF sequences are considered the most economically important (Watts, Griffis, and McOuat, 2005).

SRK is of the opinion that Cliffs has a good understanding of the geology of Bloom Lake and is using an appropriate geological model in its interpretation and its exploration planning.

7 Exploration

Historic exploration (pre-1998) included:

- Drilling by J&L (1956 to 1957);
- Drilling by CCIC and Jalore (1966); and
- Mapping, gravity and magnetometer surveys and drilling by HEN for QUECO (1971 to 1972).

7.1 Exploration by QCM (1998)

In 1998, a major exploration program was carried out by WGM for QCM, which then held the Bloom Lake property under option from CLM. The 1998 program included line cutting, surveying, road building, camp construction, diamond drilling, geological mapping, mini-bulk sampling, bench-scale preliminary metallurgical testwork, camp demobilization and site clean-up.

The drilling was carried out in three phases and QCM had the option of terminating the program at the end of each phase. Seventy-five drillholes totaling 18,705 m were completed during the entire program. A total of 2,529 samples of split core were analyzed at Lakefield. Laboratory work included routine sample analysis, heavy liquid testing, mineralogical studies and liberation testing. In addition, further testing was carried out to authenticate old drilling results, establish weight recoveries and the applicability of Mont-Wright procedures for determining weight recoveries for Bloom Lake property samples.

In the summer of 1998 and independent of the WGM-managed exploration program, QCM carried out a regional high-resolution helicopter-borne aeromagnetic/Horizontal Loop Electromagnetic (HEM)/Very Low Frequency (VLF) survey over an area of approximately 4,100 km^2. This survey was roughly centered on the Mont-Wright operation and the 19 original CLM claims (which are part of the actual 178 claims) were included within the northwest quarter of the survey area. The survey was flown by High-Sense Geophysics Limited of Toronto. Line direction and spacing were NW-SE and 200 m, respectively, and mean terrain clearance was 30 to 45 m for the geophysical birds and 60 m for the helicopter. The magnetic survey outlined the Bloom Lake iron formation, confirming its folded nature and relatively large aerial extent. A bulls-eye magnetic response was recorded over the three-claim (Roach Lake Extension) group. Neither the HEM nor the VLF survey recorded anomalies over either of the CLM claim groups.

7.2 Exploration by CLM (2005)

In 2005, CLM retained WGM to carry out a technical review, including the preparation of a Mineral Resource estimate for the Bloom Lake iron deposit to assist CLM in making business decisions and future planning. The technical review was carried out and prepared in compliance with the standards of NI 43-101 and CIM Standards.

In addition, WGM planned and carried out a mini-bulk sampling program. Eleven mini-bulk samples were drilled, blasted and collected on the North and West Hill areas between July 26 and 28, 2005. The access trail from Highway 389 had been rehabilitated and the sample sites marked out earlier in the month. Sample sites were chosen to match as closely as possible those of the similar program carried out for QCM in 1998.

7.2.1 CLM Bloom Lake West Exploration (2007 to 2009)

CLM conducted an exploration program at Bloom Lake West between Triangle and Carotte Lakes. The target was based on a regional airborne magnetometric survey made by the Geological Survey of Canada. Forty-six drillholes were made between 2007 and 2008 for a total of 10,184 m.

A detailed ground magnetometric survey was done between 612200 E and 614100 E and between 5854600 N and 5855800 N. Geophysique TMC of Val D'Or, Quebec, did the survey in April 2008 using a Geonics GEM-19 magnetometer. Another magnetometric survey was done in 2009 to enlarge the coverage to include the area between 612200 E and 616000 E and between 5853600 N and 5856800 N.

7.2.2 CLM Bloom Lake Exploration (2006 to 2010)

Between 2006 and 2010, CLM drilled an additional 155 holes for an additional 27,292 m.

7.2.3 Cliffs Exploration (2011 to 2012)

Beginning in November 2011, Cliffs drilled an additional 120 holes within the Bloom Lake property. Of these, 107 were drilled by the Bloom Lake mine geology staff, and 13 were drilled by Cliffs Global Exploration Group (GEG), for a total of 41,561 meters. All holes were drilled as NQ-core. Drilling was conducted primarily on the Bloom West area of the property (93 holes), with the remaining 27 holes drilled within the area of the main pit.

7.3 Sampling Methods and Quality

Exploration at the Bloom Lake property consists predominately of drilling and geophysical surveys. There has been limited surface sampling and the results have not been reported and have not been used in the resource estimation.

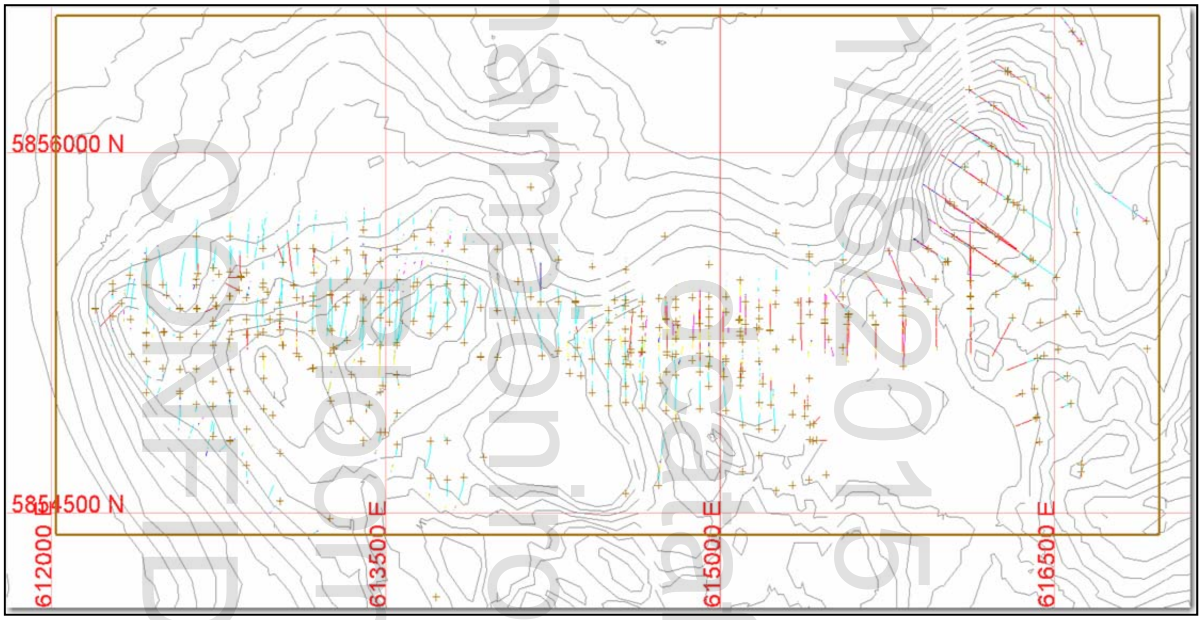
7.4 Significant Results and Interpretation

There are very limited exploration samples other than drill samples.

The geophysical surveys were successful in identifying that the iron formation covered a large areal extent at Bloom Lake. The results were also used in geological and structural interpretation of the area.

8 Drilling

A drillhole location map at Bloom Lake is shown in Figure 8.1 (in relation to the new SRK Reserve Pit).



Source: SRK

Figure 8.1: Drillhole Location Map

The drilling in the resource database consists of 108,284 m in 467 core holes. Table 8.1 is a summary of the drilling in the resource database.

Table 8.1: Summary of Drilling at Bloom Lake

Year	Company	Drillhole Identification	Size	Number	Meters
1956-1957	J&L	X-02 to 06, 09 to 11	XRT (19mm)	8	266
1956-1957	J&L, CCIC	QC-1 to QC-30	AXT (32.5mm)	33	4,769
1966	J&L, CCIC	X-14, 20	XRT (19mm)	9	175
1971	QUECO	71-10 to 71-09		9	1,834
1972	QUECO	72-01 to 72-10, 72-08A, 72-10		12	3,497
1998	QCM	98DN-001 to 73, 08A, 065A	BQ (36.4mm)	75	18,705
2006	CLM	BL-06-01, 02, 04 to 11, 16		11	2,087
2007	CLM	BL-07-003, 012 to 015, 017		6	798
2007 Bloom Lake West	CLM	LBW-07-01 to 04, 07	BQ (36.4 mm), NQ (47.6mm)	5	1,297
2008	CLM	BL-08-01, 03 to 07, 09 to 18, 20, 21, 22, 25, 27 to 29, 31	BQ (36.4 mm), NQ (47.6mm)	24	1,367
2008 Bloom Lake West	CLM	LBW-08-5, 06, 08 to11, 13 to 41, 37A	BQ (36.4 mm), NQ (47.6mm)	41	8,887
2009	CLM	BL-09-01 to 32	BQ (36.4 mm), NQ (47.6mm)	32	3,469
2010	CLM	BL-10-01 to 81, 07A	BQ (36.4 mm), NQ (47.6mm)	82	19,572
2012	Cliffs Bloom Lake	BL-12-01 to 102, 017A, 021A, 051A, 057A, 067A	NQ (47.6 mm)	107	36,445
2012	Cliffs G.E.G. Bloom West	BLW-12-01 to 011, 013, 015	NQ (47.6 mm)	13	5,116
				467	**108,284**

For all drilling since 2006, core has been transported from the drill rig to the core shed by the drilling contractor at each shift change where the core is secured in a controlled area. The core is either of the custody of the drilling contractors, the Company personnel or locked in the core shed at all times. SRK does not know the chain of custody for the core prior to 2006. However, the Project was explored by companies that would have used industry practice for that time.

8.1 Historic Drilling

J&L and CCIC conducted a diamond drill program from 1956 through 1957. The "QC" and "X" series were drilled to test IF on the Bloom Lake property. In 1966, they drilled additional "X" holes. Holes X-1 to X-20 are XRT size core holes (19 mm diameter core) and Holes QC-1 to QC-30 are AXT size core holes (32.5 mm diameter core). The "X" holes are widely spaced across the property east of 614500 E. The "QC" holes are about 300 m apart and are drilled across the property.

In 1971 and 1972, QUECO undertook an exploration program managed by HEN. Twenty-one holes were drilled with BQ drill rods, producing 36.4 mm core. Some of the holes were started with NQ tools where the ground was expected to be difficult. Nine holes were drilled in the Bloom Lake West area, eight in the far eastern portion of the Main Pit area and the remainder in the central part of the deposit.

QCM, which held the property under option from CLM, commissioned WGM to conduct an exploration program in 1998. The drilling consisted of 75 BQ sized core holes (36.4 mm diameter core) located largely in the Main Pit area, but three were drilled in Bloom Lake West.

8.2 QCM Drilling

Between 2006 and 2010, CLM drilled 37,476 m in 201 core holes at Bloom Lake. The initial focus was the Main Pit Area, but 46 of these holes were drilled at Bloom Lake West in 2007 and 2008. The procedures used in the drilling are described in the following sections.

8.2.1 Hole Collar and Down-Hole Attitude Surveys

Drillhole collars were spotted prior to drilling by chaining in the collar locations from the nearest line picket. Drilling azimuths were established by lining up the drill by eye on the cut grid line. Drill inclinations were established using a Brunton compass on the drill head. Down-hole inclinations were tested by the drillers using acid tests in the early holes. Acid tests were taken every 50 to 75 m down each hole as designated by the drill geologist.

Once the hole was finished the drill geologist was responsible for placing a marker in the collar of each hole and labeling it with an aluminum tag. Subsequently the X, Y and Z co-ordinates for these collar markers were surveyed. Where the holes were cased, these casings were for azimuth. Holes without casing were assumed to have azimuths the same as originally designated at the time of drill set-up. Stabilized core barrels were used for all drilling and deflections in azimuth and inclination are believed to be minimal and acceptable considering the scale of the deposit and the drilling density with holes collared on nominal 150 m centers.

The grid line co-ordinates with respect to the appropriate baseline are shown on the drill logs. The logs also show the NAD 83 co-ordinates for each of the holes. All previous drill collars (1956, 1967, 1971, and 1972) that were located were also surveyed. For collars not located, the collar coordinates were calculated using old grid co-ordinates as recorded on the old logs. WGM has confidence in these calculated collar locations because the drillholes located in the field were generally found close to their recorded position.

8.2.2 Core Handling and Logging Protocols

Descriptive Logging

Descriptive logs include a description of the various geological features noted during examination of core including lithological and structural description. Descriptive logs also record hole locations and all down hole survey data, all sample locations, iron and sulfur assays, SG and magnetite determinations for each sample.

The Log II computer program of Log II Systems Inc., Toronto, Ontario was used for descriptive logging.

Geotechnical Logging

The geotechnical logging was done on a drill run by run basis, generally at 3 m intervals. The geotechnical logging is based on a digital code system with the code representative of the drill run as a whole.

Geotechnical data recorded included core recovery as a percentage of recovered core per run; Rock Quality Designation (RQD), a measure of fracture density; type of foliation, i.e., thin bands, wide bands, undulating, anastomosing, discontinuous or folded; foliation angle with respect to core axis; friability; color and granulometry index with values from 1 to 3 with 1 representing aphanitic grains, 2 representing medium grains and 3 representing grains greater than 2 mm in size. A summary of codes is in the footer to each of the geotechnical logs.

Video Recording

A video record of all core was made as part of the logging process. An 8 mm video camera was used and the core was photographed tray by tray. Audio highlights were added as appropriate. Blocks denoting hole number and meters down hole are clearly legible in the record. The video was subsequently converted to VHS format.

Magnetic Susceptibility

As part of the logging routine, magnetic susceptibility measurements were taken on core from all of the drillholes as an approximation of the magnetite content. A Mico Kappa KT-5 instrument distributed by Scintrex was used for the determinations. The unit has a sensitivity of 1 x 10-5 SI units. Readings were taken every meter (approximate position) on the core's un-split surface down each of the holes. Readings were taken by the geotechnical logger. The magnetic susceptibility measurements are listed with the geotechnical log. The readings listed are "apparent" susceptibility, uncorrected for core size.

Core Storage

After core logging and sampling were completed, core trays containing the reference half core and the un-split parts of the holes were stacked in core racks. No core was discarded. The core was stored at the QCM facility.

8.3 CLM Drilling

Consolidated Thompson conducted drilling campaigns from between 2007 and 2010. BQ size drill core was recovered in 2007 and subsequently NQ size tools were used.

The following sections are largely excerpted from Genivar (2009).

8.3.1 Drillhole Location

The holes were collared on-site with a portable Garmin GPS. This position could vary from a few meters to accommodate drilling, depending on the ground conditions but still, was maintaining the relative position and spacing relative to the other holes. Drilling azimuth reference was provided through points of coordinates. The use of a compass was not recommended due to the high level of magnetism developed by some horizons of the underlying iron formations.

8.3.2 Deviation Tests

Deviation and inclination tests were carried out in the holes. Tests with hydrofluoric acid (HF) were done for the drilling of 2007. In 2008, a Flexit instrument was used to measure both orientation and inclination of all the drillholes. This instrument provided useful magnetic susceptibility values. Readings were taken every 15 or 30 m. All the data obtained with the Flexit instrument were analyzed and all the inappropriate data were eliminated if deviation was too large and/or if the magnetic susceptibility was too high. Deviation readings were not taken for many drillholes that were lost or abandoned.

8.3.3 Collars Surveying

All the drillhole collars were surveyed. The firm of land surveyors, Roussy Michaud from Sept-Îles, put in place stations on the pit site. These points were used as references for positioning the West Zone. Surveyors of Roussy Michaud and Consolidated Thompson used a Trimble R8 instrument to survey the drillhole collars. The inclination and direction of the drill collars were not precisely surveyed. An approximate direction was obtained in aiming at a 3 m rod inserted into the drillhole tubing and then, direction was verified against the Flexit readings.

8.3.4 Core Shack

The core shack was established in the industrial area of the Town of Fermont. It is situated in a large warehouse building used for various purposes. In the core shack area, a number of inclined tables were installed for core logging with several core racks for boxes storage. An area was also organized for sampling and shelves were put in to store sample bags before being shipped to the assay laboratory. Drillers also used an area in the core shack to store spare parts. Another closed and locked section, contained core boxes of the drilling programs from previous and actual campaigns. Bloom Lake is in the process of constructing a core storage facility at the mine site.

8.3.5 Geotechnical Data on Drilling

Until July 2008, 10 ft drill rods were used. The drillers identified the marks in the boxes using the imperial system and then, a conversion into the metric system was done at the core shack. After July, 3 m rods replaced the 10 ft rods. Drillers also took care of marking with wooden sticks the core portions not recovered. At the drill rig, all the used core boxes were carefully closed with tape and were transported by either snowmobile or VTT to a pick-up truck which brought them to the core shack in the Town of Fermont at the end of each shift. No core box was left outside the core shack.

All the boxes were labeled, photographed in lots of five and most of them were photographed in detail, three to four pictures being taken for each box. The core boxes were systematically measured to validate the marks of the drillers. Measuring was also done to calculate the RQD and the core recovery. The core recovery is generally greater than 90% but there were problems with some holes and the recovery was much lower in the 40 to 50% range.

8.3.6 Core Logging

The core was logged using standard verified methods. Rock types were identified and intervals were measured according to the marks done by the drillers. Logging took into account the general color of the rock, the relative percentage of constituents, the grain size distribution, texture and the variation of these elements when significant. A particular attention was given to the orientation of foliations relative to the core axis. This was very useful in the structural interpretation. The mineralized units to be sampled were marked with a grease pencil at 1 to 6 m intervals, depending on the mineral content.

All the data were stored in the Geotic-Log software, which uses an MS Access database. The rock type codes used are indicated in Table 8.3.6.1.

Table 8.3.6.1: Rock Type Codes for Descriptive Logs

Code	Rock Type
AMP	Amphibolite
OIF	Oxide Iron Formation
SIF	Silicate Iron Formation
MS	Mica Schist (60-100% Mica)
MSIF	Mica Schist with more than 15% Iron Oxide
QR	Quartzite (0-40% Mica)
QRIF	Quartzite with less or equal to 15% Iron Oxide
QRMS	Quartz Rock Mica Schist (between 40 and 60% Mica)
GN	Gneiss

8.4 Cliffs Drilling

8.4.1 Bloom Lake Geological Staff

Drillhole Location

The holes were collared on-site by the Bloom Lake mine survey crew. Position could vary a few meters to accommodate drilling, depending on the ground conditions, still maintaining the relative position and spacing relative to the other holes. Drilling azimuth reference was provided through points of coordinates.

Deviation Tests

Deviation and inclination tests were carried out in the holes. A Flexit instrument was used to measure both orientation and inclination of all the drillholes. Readings were taken every 15 or 30 m. All the data obtained with the Flexit instrument were analyzed, and all the inappropriate data were eliminated if deviation was too large and/or if the magnetic susceptibility was too high.

Collar Surveying

All the drillhole collars were surveyed by Bloom Lake survey crew staff. Direction was obtained in aiming at a 3 m rod inserted into the drillhole tubing, and was verified against the Flexit readings.

Core Shack

The core shack was established in the industrial area of the Town of Fermont. It is situated in a large warehouse building used for various purposes. In the core shack area, a number of inclined tables were installed for core logging with several core racks for boxes storage. An area was also organized for sampling and shelves were put in to store sample bags before being shipped to the assay laboratory. Drillers also used an area in the core shack to store spare parts. Another closed and locked section, contained core boxes of the drilling programs from previous and actual campaigns. Bloom Lake is in the process of constructing a core storage facility at the mine site.

Geotechnical Data on Drilling

All the boxes were labeled, photographed in lots of five and most of them were photographed in detail, three to four pictures being taken for each box. The core boxes were systematically measured to validate the marks of the drillers. Measuring was also done to calculate the RQD and the core recovery. The core recovery is generally greater than 90%.

Core Logging

The core was logged using standard verified methods. Rock types were identified and intervals were measured according to the marks done by the drillers. Logging took into account the general color of the rock, the relative percentage of constituents, the grain size distribution, texture and the variation of these elements when significant. A particular attention was given to the orientation of foliations relative to the core axis. This was very useful in the structural interpretation. The mineralized units to be sampled were marked with a grease pencil at 1 to 6 m intervals, depending on the mineral content.

All the data were initially stored in Geotic-Log software, which uses an MS Access database. The rock type codes used are indicated in Table 8.3.6.1 above. Mid-way through the drilling program of 2012, core logging was completed using Gemcom logging software designed to integrate directly with their modeling software.

8.4.2 Cliffs Global Exploration Group (GEG)

This drilling program was managed by Watts Griffis and McOuat (WGM).

Drillhole Location

Drillholes were spotted in the field by geological crew members using hand-held GPS. Similarly drillhole azimuth was established using either hand-held GPS or a solar compass and foresights erected for visual alignment of the drill.

Deviation Tests

DGI Geoscience Inc. ("DGI") was contracted by Cliffs to complete location and azimuth surveys of the collars and down-hole gyro surveys. DGI visited the Project during drilling and at the end of drilling to perform its surveys. The collar location and azimuth surveys were done using a Reflex Azimuth Pointing System™. The down-hole surveys were completed using a Reflex Gyro.

Downhole Surveying

DGI also completed down-hole geophysical surveying which included optical televiewer, caliper, magnetic susceptibility, and gamma-gamma density.

Collar Surveying

All the drillhole collars were surveyed by Bloom Lake survey crew staff.

Core Shack

WGM established a core shack and repository/laydown area on Mine property near the mine geology office facilities. A series of quonset tents were erected for logging, sample preparation and storage, and geological interpretation work. Portable generators provided electrical power.

Core Logging

Geotechnical personnel completed the logging and sampling of the drill core. This crew varied through the program and rotated in and out of the field.

Geological logging included both Descriptive Logging and Geotechnical Logging. Geotechnical logging included core recovery, RQD, fracture counts and classification, granulometry and friability. The Cliffs Bloom Lake Mine guide for logging was used for guidance. Logging included a core tray inventory of metreage interval per tray, wet and dry core photography and magnetic susceptibility measurements down the core using a hand-held magnetic susceptibility instrument. Logging generally was done directly into spreadsheet MSExcel logging forms designed for the purpose and then imported into an MSAccess database system. Core trays were labeled with aluminum labels and are stacked in core racks at the camp site protected from weather.

8.5 Interpretation

The drilling at Bloom Lake covers an area that is 4.7 km long in the east-west direction and 1 to 2 km long in the north-south direction. The drillhole spacing is on sections spaced at about 50 m in the center of the deposit and 75 m at Bloom Lake West and in the east. The drillholes are spaced between 50 and 100 m on section. The holes have all been drilled with core, with very small core drilled in the 1950's, increasing to BQ in the 1970's and more recently to NQ.

It is SRK's opinion that the drilling conducted by CLM meets present industry standards and that the earlier drilling met the standards at the time.

9 Sample Preparation, Analysis and Security

9.1 QCM Sampling

The following section was excerpted from Section 12 of CTIM-BBA (2008).

In 1998, the demarcation of sampling boundaries was made generally on a geological basis as selected by the drill geologist during logging. The sampling guidelines required minimum sample lengths of 3 m, with maximum lengths of 4.5 m for NQ core and 7 m for BQ core. This scheme was closely followed for the most part. No samples were taken from holes 98DN-040 and 98DN-053 because all rock intersected was amphibolite.

The average sample length was close to 6 m as most of the drilling was BQ and most samples ranged in weight from 5 to 12 kg. All rock estimated to contain at least 10% iron in the form of oxide was sampled. In addition, one sample on either side of all IF was taken in wall rock to bracket all IF sequences.

Some samples include more than one rock type because of the 3 m minimum sample length stipulation. This situation most often occurs in sequences of IF alternating with narrow, less than 1 m intervals of amphibolite. In such cases, individual samples may include both amphibolite and IF.

Sample intervals were marked out by the drill geologist with a china marker during descriptive logging. Three-part sample tickets, from sample books with unique sequential ticket numbers, were used to number and label core and subsequently sampled intervals and bagged samples.

Sample numbers and intervals were recorded for later computerization and collation in a sample database. A second record of sample number and interval was saved with the descriptive log. One portion of the sample ticket remained in the sample book. One portion was placed under the last few centimeters of core in the sample interval in the core tray. The third portion was inserted by the core splitter/sampler into a 40 cm x 60 cm canvas sample bag with the split core sample. The sample number was also written on the sample bag.

Core samples were split using a hydraulic core splitter. The second half of the spit core sample was returned to the core tray. These reference split core segments were put back into the core trays in the original order and fitted back together as much as possible. Core trays were then stacked on timber core racks located on site.

Sample bags were stored in the splitting tent until removed for transport to the highway and then to QCM for loading onto transport trucks and shipment to Lakefield in Ontario.

Sample shipments from camp to Mont-Wright were nominally twice per week. Sample bags were loaded onto H&S tracked vehicles (GT-1,000s) and transported to the highway; a 1 to 1.5 hour trip. At the highway the sample bags were loaded into 25 gallon steel drums. The sample bags were as much as possible loaded into the drums in numerical sequence. Each drum held from 10 to 15 sample bags depending on sample and drill core size, i.e. BQ or NQ. Each of the sample shipments, consisting usually of 4 to 8 drums, was assigned a sequential sample shipment number and each drum was assigned a consecutive (all-shipments-inclusive) drum number. Shipment numbers and drum numbers were recorded in the sample database. A list of samples comprising the sample shipment was placed into the first drum in each shipment. Sample drums pre-loaded onto a pick-up truck were then sealed and transported to the Mont- Wright warehouse where they would await next-day transport to Lakefield. The entire sample shipment procedure from camp to the Mont-Wright mine was carried out by the sampler/splitter assisted by either a geotechnician or a geologist. Once delivered to the warehouse the samples were in the hands of Mont-Wright mine warehouse staff.

9.2 CLM Sampling

This section was excerpted from Section 14 in Genivar (2009).

The demarcation of sampling boundaries was done using both geological and mineralogical criteria as selected by the geologist during logging, using a china marker.

Three-part sample tickets, taken from sample books with unique sequential ticket numbers, were used to number and label sampling intervals and subsequently bagged samples. Sample numbers and intervals were recorded for later data storage in computer and collation in a sample database. A second record of sample numbers and intervals was saved with the descriptive log.

Core samples were split using a hydraulic core splitter. The second half of each split sample was returned back in the core tray. These reference split core segments were put back into the core trays in the original order, trying to make them fit together as much as possible. Core trays were then stacked on timber core racks located on site.

In general, the average sample length ranged from 4 to 6 m for the BQ size and from 3 to 4 m for the NQ. Most samples ranged in weight from 1 to 10 kg. The recovery of a number of samples (OIF) was poor as they were partly or completely crumbled. As a result, half of the recovered sand was sampled.

One portion of the sample ticket remained in the sample book. One portion was placed at the beginning of the first centimeters of each sampling interval in the core tray. The third portion was inserted by the core splitter/sampler into a 40 cm x 60 cm canvas sample bag with the residual split core sample. The sample number was also written on the sample bag which was properly stapled several times. For transportation, the bags were put into larger ones in bunches of 2 to 4. Blanks (quartz) were also randomly inserted at intervals of about 20 to 30 samples by the geologist to keep the sequential numbers unaffected.

Sample bags were stored in a core shack until removed to go, via pick-up trucks, to TST Overland Express in Wabush, Labrador, which then transported them to the laboratory of SGS Lakefield Research Limited (Lakefield), in Lakefield, Ontario. Once delivered to TST Overland in Wabush, the bags were put on pallets that were sealed with plastic wrap-ups.

9.3 Cliffs Sampling 2012

Sample intervals were marked out by the drill geologist with a china marker during descriptive logging. Three-part sample tickets, from sample books with unique sequential ticket numbers, were used to number and label core and subsequently sampled intervals and bagged samples. Sample numbers and intervals were recorded in the drillhole database.

One portion of the sample ticket remained in the sample book. One portion was placed and stapled under the first few centimeters of core in the sample interval in the core tray. The third portion was inserted by the core splitter/sampler into a polyweave sample bag with the split core sample. The sample number was also written on the sample bag.

Core samples were split using a hydraulic core splitter. The second half of the spit core sample was returned to the core tray. These reference split core segments were put back into the core trays in the original order and fitted back together as much as possible. Core trays were then stacked on core racks located on site.

Sample bags loaded into 205 L barrels which were picked up by the transport company for shipping by truck to the laboratory on a periodic basis.

9.4 Security Measures

The samples were maintained in the operator's core storage facility which was locked when the geological staff was not present. The samples were sent to SGS Lakefield by TST Overland Express based in Wabush, Labrador.

9.5 Sample Preparation used by Both CLM and Cliffs

All split core samples were sent to SGS Lakefield. Each sample was weighed, then jaw crushed to 6 mesh and riffle split. A 2 kg split was then passed through a roll crusher to 100% minus 20 mesh. A 150 g split was then pulverized to minus 200 mesh. A 1.5 gram sub-sample was used for whole rock analysis, a 5 gram sub-sample for determining magnetic iron and a 30 gram subsample was used for SG determinations. These various sub-samples were then subjected to analysis as described below.

9.6 Assaying

CLM and Cliffs uses SGS Lakefield for analysis. SGS Lakefield is an accredited laboratory with accreditation provided by the Standards Council of Canada and conforms with the requirements of CAN-P-1579, CAN-P-4E, which is ISO/IEC 17025:2005 for multiple elements by various preparation and analysis methods. This accreditation includes "*The Preparation and Determination of Iron in Ores, Concentrates and Metallurgical Products by Fusion, Separation and Titration [Fe]*". The accreditation was issued on October 10, 2012 and is valid through June 3, 2016.

9.6.1 1998

All the 1998 split core samples were analyzed for a suite of whole rock elements including: SiO_2, TiO_2, Al_2O_3, Fe_2O_3, MnO, MgO, CaO, Na_2O, K_2O, P_2O5 and loss on ignition (LOI). Analysis was done on lithium tetraborate fused pressed pellets by X-ray Fluorescence (XRF) following sample crushing and pulverization. Additional analysis included determining magnetite by the Satmagan method, and total sulfur by infrared detection analysis using LECO instrumentation following sample combustion. SG for each sample was determined using an air comparison pycnometer. It should be noted that this method does not take into account existing porosity in a rock and some of the IF contains vugs due to calcite removal. Although the degree of porosity has not been quantified, it is estimated on the basis of visual examination of drill core to be generally less than 2%.

Total iron was calculated from Fe_2O_3 by dividing total iron expressed as Fe_2O_3 by a factor of 1.4295.

9.6.2 CLM Assaying

All CLM samples were analyzed at SGS Lakefield. A whole rock analysis was done on each sample to measure the following parameters: 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). Sample preparation entailed the formation of a homogenous glass disk by the fusion of 0.2 to 0.5 gram of rock pulp with 7 gram of lithium tetraborate/lithium metaborate (50/50). The disc specimen was then analyzed by WDXRF spectrometry. The LOI at 1,000°C was determined separately by gravimetry. Additional analysis included determination of magnetic iron with a Satmagan magnetic balance as well as sulfur by combustion-infrared detection on LECO instrumentation.

Specific gravity was determined using an air comparison pycnometer. It should be noted that specific gravity was not measured for a number of drillholes at the beginning of the drilling campaign.

Total iron was calculated from Fe2O3 by dividing total iron expressed as Fe2O3 by a factor of 1.4295.

9.6.3 Cliffs Assaying

SGS Lakefield analyzes samples submitted by the Company using whole rock analysis with a lower detection limit of 0.01% for all constituents. A 250 g pulp is split for analysis and an aliquot of that sample is fused with lithium borate and analyzed by XRF. Analysis includes SiO_2, Al_2O_3, Na_2O, K_2O, CaO, MgO, Fe_2O_3, MnO, TiO_2, P_2O_5, Cr_2O_3, V_2O_5 and LOI. In addition, this fraction was submitted for analysis of STotal, and C by LECO furnace as well as Satmagan and specific gravity determination.

Total iron was calculated from Fe_2O_3 by dividing total iron expressed as Fe_2O_3 by a factor of 1.4295. Selected samples were also assayed for FeOTotal using a H_2SO_4-HF digestion.

9.7 Quality Assurance/Quality Control Procedures

A Quality Assurance/Quality Control (QA/QC) program separate from the laboratories internal QA/QC is an industry best practice used to independently monitor precision and accuracy of the laboratory. This type of program also serves to monitor for sample mix-ups that may be introduced during sample collection by the Company. Independent blanks and standards are samples that should return a known analytical result and are in known position in the sample stream.

An industry best practice QA/QC program includes two to three types of duplicates depending on the stage of the project, preparation blanks, two to three standards and a check sample that is a pulp duplicate sent to a second laboratory. Duplicates are used primarily to test precision of analyses. Duplicate sample types include field or core duplicates, preparation duplicates from the coarse fraction and pulp duplicates.

Preparation blanks are used to monitor cross contamination during sample preparation. Pulp blanks may also be used but check for cross contamination during analysis, which would not be an issue in XRF analysis.

Standards are samples of known value that are used to monitor the accuracy of the analytical laboratory and are selected to bracket the grade range of the deposit. Finally, check samples are used to verify the analytical results from the primary laboratory. Check sample submissions to a secondary laboratory must use the same digestion and analytical procedures as the primary laboratory and must contain a standard in the sample submission. The frequency of the QA/QC samples is based on batch size of analytical methods.

A QA/QC program must be monitored on an ongoing basis, so that QA/QC failures can be identified and corrective action taken during the drilling the program and before the data becomes part of the resource database. Failures range from sample mix-ups to analytical problems. Should a failure result from an analytical problem, either a series of samples bracketing the failure or the entire batch should be reanalyzed depending on the number and types of samples involved and the type of failure. Accepted ranges for analytical failures by QA/QC are listed below:

- Core Duplicates ±30%;
- Preparation Duplicate ±20%;
- Pulp Duplicate ±10%;
- Preparation Blank 5 times the detection limit;
- Pulp Blank 3 times the detection limit;
- Standard ±2 standard deviation (stdev) or approximately ±10%; and
- Check sample to a second laboratory ±10%.

Internal laboratory QA/QC usually includes duplicates, blanks and standards in each batch. The analytical results and descriptions of the types of QA/QC insertions can be requested by clients. However, this does not replace independent QA/QC insertions.

9.7.1 SGS Lakewood Internal QA/QC

The SGS Lakewood internal QA/QC program includes insertion of blanks, standards in the form of Certified Reference Material (CRMs), analytical duplicates and replicates (preparation duplicates). These samples are inserted into the sample batches that SGS Lakefield receives from clients. Typically laboratories include the results for CRMs, blanks and analytical duplicates on analytical certificates. However, companies must request that replicates be included on the certificate.

9.7.2 Cliffs QA/QC

Prior to 2012, Cliffs did not have a QA/QC program independent of the analytical laboratory for analyses at Bloom Lake and relied on the internal QA/QC program at SGS Lakefield. In 2012, Cliffs initiated a QA/QC program that includes field duplicates and blanks. The Company has collected samples for two site specific standards. The standards are in the process of being certified in a round robin analytical program and will be available for subsequent drilling programs. All analyses are currently conducted by SGS Lakefield. Cliffs submits the following QA/QC samples at the following frequencies:

- Duplicates 1 every 50 samples; and
- Blanks 1 every 20 samples.

<u>Duplicates</u>

Cliffs is currently using core duplicates, which represents the remaining ½ core after the original sample is collected. There is no core archived from intervals where a duplicate sample has been submitted and Cliffs relies on core photographs for a record of these intervals. The duplicate sample is assigned a sample identifier that may be in sequence directly following the original sample or inserted at an offset of several sample positions from the original sample. SRK observed that all of the duplicates provided were immediately following the original sample in numbering sequence.

Cliffs submitted 133 duplicate sample pairs during the 2012 drill program. Of these, three were missing the original sample numbers and analytical data for comparison with the duplicate data and three others appeared to be sample number mix-ups. These three samples had the same sample number for both the original and duplicate. Where this was observed, SRK found good correlation by using the preceding samples as an original of the duplicates for comparison. However, SRK recommends Cliffs investigate to verify that this is correct for these samples.

Other missing information included eight samples with duplicate data, but no original data that could be located in the drillhole database, and 24 sample pairs where analytical data had not been received at the time of this report.

SRK reviewed all of the analyses provided for all of the constituents but focused on FeT and Magnetic Fe analysis as well as the following deleterious constituents; SiO_2, Al_2O_3 and P_2O_5. The number of failures and the percentage of failures of the 79 sample pair population are presented in Table 9.7.2.1.

Table 9.7.2.1: Field Duplicate Failures

Constituent	Number of Analyses	Total Number of Failures	Percentage of Failures in the Total Population	Number of Failures in Critical Ranges	Percentage of Failures in the Total Population in the Critical Ranges
FeT	79	13	16%	7 > 15% Cut-off Grade	9%
Magnetic Fe	79	25	32%	0 > 15% Cut-off Grade	0%
SiO_2	79	3	4%	3 = 40-50%	4%
Al_2O_3	79	29	37%	5 > 1.5%	6%
P_2O_5	79	16	20%	0 > 0.75%	0%

Both FeT and magnetic Fe have relatively high failure rates for the sample population. For FeT there were approximately six failures below and seven failures above the resource cut-off grade of 15% FeT. The Al_2O_3 analysis had a 29% failure rate overall with 6% above 1.5% Al_2O_3 concentration. At 1.5%, Al_2O_3 presents a problem during processing, so although elevated it is not considered significant. All P_2O_5 analyses reported concentrations of 0.75% and lower. This is approaching the detection limit for the analytical procedure where analysis becomes difficult to reproduce. This is also below the level where P_2O_5 becomes a processing consideration. Reproducibility for SiO_2 analysis was consistent with only 3 failures or 4% failure rate. Figure 9.7.2.1 presents a scatter plot for FeT.

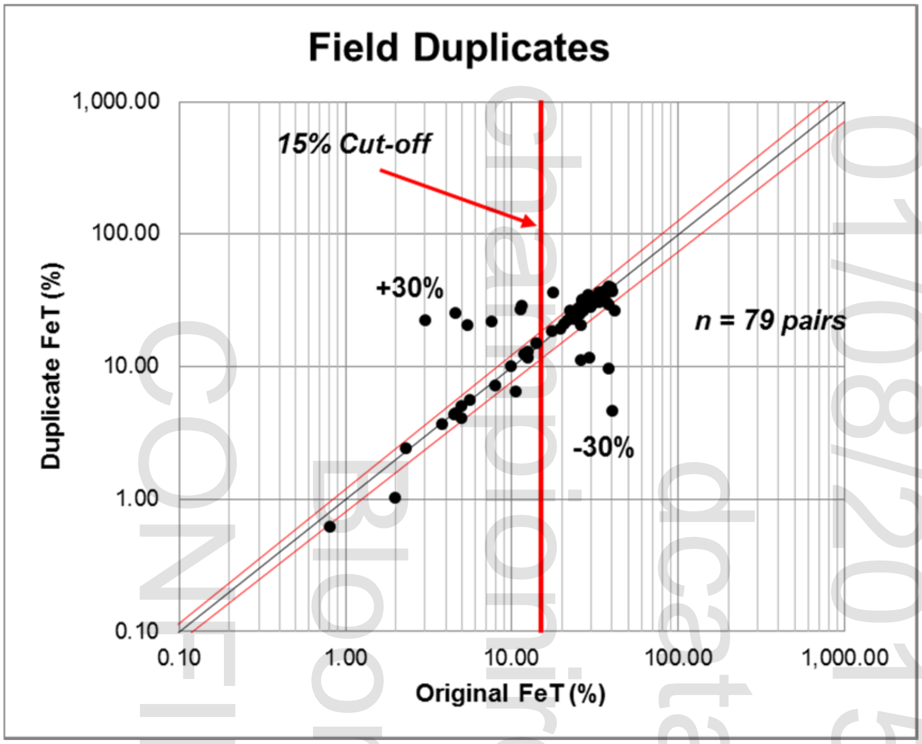


Figure 9.7.2.1: Scatterplot for Fe Results

The FeT failure rate is slightly elevated. Good reproducibility is expected in FeT analyses from banded iron formations, which are relatively homogenous as compared to other deposit types. This is expected even between core duplicates.

Typically, core duplicates are used early in a drilling program to determine if the sample submitted is a large enough volume or prepared appropriately for analysis. If the core duplicates show poor reproducibility, the sample submission can be adjusted to allow more sample to be split for sample submitted is too small, more sample can split during the crushing stages to provide more sample to be pulverized. Problems with core duplicates can also be that the sample is inadequately pulverized and that a finer grind may be necessary. Once it is determined that an adequate sample is being prepared for analysis, core duplicates can be discontinued unless a new area or different deposit is drilled.

SRK recommends that Cliffs first examine the database to determine if some of the failures are the result of sample mix-ups such as mislabeling. If this is not the case, then SRK recommends that Cliffs examine the sample preparation to determine if adequate sample is being provided after crushing or if the grind size during pulverization is adequate.

Preparation duplicates collected prior to pulverization can also provide a check of sample preparation as well analytical reproducibility. However, pulp duplicates provide the most accurate check of analytical precision. Check samples, pulp duplicates sent to a second laboratory, will verify results of the SGS Laboratory. Any check samples sent to a second laboratory must be accompanied by a standard and the secondary laboratory must use the same or as close to the same analytical and digestion procedures as possible.

An additional recommendation is to add preparation and pulp duplicates as well as check samples to Cliffs QA/QC program. SRK recommends alternating between duplicate types during insertions using the following schedule:

- Sample 50 = core duplicate;
- Sample 100 = preparation duplicate;
- Sample 150 = pulp duplicate; and
- Sample 200 = check sample.

If it is determined that the sample size and grind are adequate, then the core duplicates can be discontinued.

Blanks

Currently, Cliffs is using intervals of amphibolite drill core as a field blank. These are labeled FBLK and consist of a course grained massive to cryptically bedded rock comprised of hornblende, plagioclase and biotite. The material was drill core from historic and recent drilling at the Project. The rock is low in magnetite and low in hematite but contains 9 to 12 % FeT. When a Blank was required, half split amphibolite core representing a sample length of 3 to 6 m of core was placed into a regular sample bag and given a regular sample identifier in sequence with the Routine samples.

The amphibolite is inappropriate as a blank since a blank should be zero for the constituents of interest. The FeT is presented as a range of 9 to12%, so a sample would have to be above 12% to trigger investigation for cross contamination. The amphibolite is equal to or exceeds the SiO_2, Al_2O_3 and P_2O_5 content of the resource, so it is not useful for tracking cross contamination of the deleterious constituents. SRK recommends discontinuing the use of this material as a blank.

Finding a suitable coarse blank that is zero for Fe and SiO_2, may be impossible. SRK observes that the XRF analysis has a lower detection limit of 0.01% for FeT. Cross contamination would be any sample ≥0.05%. The database contains duplicate samples with 1% FeT, making it difficult to distinguish between a blank and other QA/QC samples in the event of a sample mix-up. In addition a cross contamination failures of 0.05% in an iron deposit with a cut-off grade of 15% FeT will have little effect if any on resource estimation. SRK is of the opinion that a blank used to monitor Fe cross contamination in an Fe deposit is not useful or necessary. In this type of a deposit, a blank can be used to monitor cross contamination for deleterious elements.

Standards

Cliffs has collected, crushed, pulverized and homogenized two samples to be used as standards. The material was selected to bracket the analytical range in the deposit. This material is currently, being analyzed in a Round Robin for certification and will be available for the next drilling program.

9.8 Opinion on Adequacy

Cliffs is using preparation and analytical techniques that are industry standard for iron deposits and analysis is being conducted by at SGS Lakefield, an accredited laboratory. Cliffs has also implemented a QA/QC program independent of the laboratory that includes core duplicates and preparation blanks. The core duplicate program is early and has identified some potential sample mix-ups that need to be investigated. The sample failure rate above the 15% FeT cut-off grade is somewhat elevated at 9% failure and this should be investigated to determine if this is the result of sample preparation. SRK is of the opinion that the resource database is appropriate and can be used for resource estimation.

SRK recommends that Cliffs monitor its QA/QC program continuously during the drilling programs and investigate failures to determine if they are the result of sample mix-ups or actual issues with analytical or preparation at the laboratory. An additional recommendation is to add preparation and pulp duplicates as well as check samples to Cliffs QA/QC program. Check samples sent to a secondary laboratory should include standards and the secondary laboratory must use the exact digestion and analytical technique or one as close as possible to the primary laboratory.

SRK recommends discontinuing the use of the amphibolite material as a blank. Finding a suitable coarse blank that is zero for Fe and SiO_2, may be impossible. SRK observes that the XRF analysis has a lower detection limit of 0.01% for FeT. At these levels a cross contamination failures of 0.05% in an iron deposit with a cut-off grade of 15% FeT will have little effect if any on resource estimation. SRK is of the opinion that a blank used to monitor Fe cross contamination in an Fe deposit is not useful or necessary. However, a blank should be used to monitor cross contamination for deleterious elements such as P_2O_5 that can cause issues in processing. SRK recommends finding a blank that is zero for the deleterious constituents.

10 Data Verification

The following paragraphs were excerpted from Section 16.1 in Genivar (2009).

A database was provided to GENIVAR by Consolidated Thompson. It contained coordinates of drillhole collars, deviation tests, lithological contacts, measures of contact and foliation, and assay results. Verifications were done with the original log books for the 71 to 72 drillholes, available in the archives of the Ministry of Natural Resources offices and with information available on their web site.

The modeling method as well as the block model produced to estimate both grades and tonnes of ore were also verified. The conversion of the old drillholes coordinates was done by Watts, Griffis and McOuat Limited when the resource was calculated for the pit in 2005. The method of conversion was not specified in this 43-101 Technical Review and Mineral Resource Estimate, dated May 26, 2005.

The following paragraphs in Section 10.1 were excerpted from Section 14 in CTIM-BBA (2008).

A limited amount of analytical work was undertaken in 1998 to investigate certain problems related to authenticating, combining and using the analytical data from the old drillholes for the Mineral Resource estimate.

For the 1998 program, as described previously, drillhole samples were analyzed for total iron by XRF and magnetite was determined by Satmagan. Total iron was not determined for either the QC or 71/72 series drillhole samples. The QC and 71/72 series drillhole samples were analyzed for soluble iron rather than total iron and this soluble iron was likely determined using the hydrochloric acid-stannous chloride procedure. Furthermore, magnetite in the QC holes (expressed as magnetic iron) was determined by Davis Tube, while for the 71/72 and 1998 drillholes, magnetite was determined by Satmagan. In addition, although QC samples were likely analyzed at the Jones & Laughlin laboratory in Negaunee, Michigan under a quality control situation, neither the analytical and quality control procedures, nor the laboratory actually responsible for the analysis are documented in reports on-hand. It is known that the 71/72 drillhole samples were analyzed at Lakefield.

Analytical work was undertaken to investigate the relationship between soluble and total iron analysis as part of the 1998 program. It was believed that this research was required before analytical data for the QC, 71/72 and the 1998 holes were combined for the purpose of preparing the Mineral Resource model. To this end two investigations were undertaken:

- 1. Forty-one samples were selected from the 1998 holes and analyzed for both soluble iron and total iron so that the differences between soluble iron and total iron could be assessed for a group of samples containing varying amounts of iron silicates; and
- 2. A 1972 drillhole was twinned. Hole 98DN-070 was drilled as close as possible to hole 72-02. The result was a string of samples for which new assays of total iron and magnetite were obtained. These assays could then be compared against 1972 assays for soluble and magnetite for a closely comparable rock sequence.

10.1 Soluble Iron versus Total Iron

A statistical analysis of for the analytical results indicates that the total iron results are slightly higher than results for soluble iron as shown by the mean value, and median values. Soluble iron and total iron generally increase in a linear fashion.

The sample with the largest difference between its total iron and soluble iron concentrations is sample 100984. This sample was found to contain 76.3% grunerite during X-ray Diffraction Analysis (XRD). Although there is some correlation between the magnitude of the difference between total and soluble iron and MgO content, the degree of correlation is poor and not entirely explained by the actinolite content, as reflected by its MgO concentration.

10.2 Drillhole Twinning

Hole 72-02 was originally assayed for soluble iron. A twin hole was drilled in 1998 and the samples were analyzed for total iron. Magnetite in both cases was by Satmagan.

The statistical analysis shows that the results for soluble iron are slightly lower than those for total iron, fitting with the ideal that soluble iron is a partial analysis. The average for magnetite in hole 72-02 is slightly higher than the average for magnetite in hole 98DN-070 but the medians show the reverse pattern. The distribution of magnetite results is log-normal and accordingly the median should be a better estimate of central tendency for magnetite results than the arithmetic average. The total iron assays have a normal distribution.

WGM concludes that the assays for total iron for the old holes appear to be unbiased quality data and that the use of unadjusted soluble iron assays for the QC and 71/72 series holes in the Mineral Resource estimate is appropriate and if anything probably slightly undervalues the grade of the deposit in terms of total iron content. The magnetite results, for the 71/72 holes, may be a little less reliable and a bit too high but there are not many of these holes and their volume influence for the deposit is relatively small.

10.3 Procedures

This section was excerpted from Section 16 in Genivar (2009).

Verifications were done at the property itself in order to find the collars of holes done during prior drilling programs. Some of these drillhole collars could not be found.

However, the deforest areas observed, were clear evidences of collars location. Further verification was done on the drill core of the future Bloom Lake open pit as well as on the drill core of the current campaign on Bloom West, stored on-site.

The procedures for all of the activities related to the drilling program were established by GENIVAR. Particular attention was paid to the nomenclature used for the various geological units, making sure that it was similar to the one previously used for the pit area.

Five visits were done on-site in Fermont by GENIVAR between October 2007 and February 2009. The objectives of these visits were to carry out visual inspections of the overall site, of the layout and organization of the installations as well as the examination of the drill cores.

10.4 SRK Verification

SRK requested 30 assay certificates for comparison to the database. Only seven of the 30 were available and those were all from 2007 and 2008. SRK compared them to the database and found no errors.

10.5 Limitations

SRK was able to check only a small part of the database, because the requested assay certificates were not available. SRK has relied on Genivar and WGM to provide a suitable database free of errors. Based on the small data check that was conducted by SRK, it is SRK's opinion that WGM and Genivar have performed suitable checks on the data.

10.6 Data Adequacy

Prior to the 2012 drilling program, there were no QA/QC programs independent of the laboratory to check on the laboratory precision and accuracy. The 2012 QA/QC program is discussed in Section 9.

The laboratories used for analyses at Bloom Lake had and have internal QA/QC programs. However, it is a standard industry practice to have a QA/QC program independent of laboratory QA/QC in place for all exploration programs. The majority of exploration data is not supported by a QA/QC data.

Bloom Lake is an operating mine that has economically mined areas delineated by exploration drilling. Based on this, SRK considers that the work by WGM and Genivar have suitably validated the database prior to 2012. Going forward Cliffs will have QA/QC support for its database.

11 Mineral Processing and Metallurgical Testing

The initial metallurgical testwork for Bloom Lake was undertaken during the period 1973 to 1976. At that time Lakefield Research (now SGS Lakefield) conducted gravity concentration tests with a Wilfley shaking table on drill core composites crushed to 35 mesh (425 microns). In 2005, eleven mini-bulk samples were taken from iron formation outcropping on the property and sent to SGS Lakefield for testing. Additional confirmatory tests were run in 2006 to 2007 on 32 drill core composites as part of the Bloom Lake Feasibility study prepared by BBA and issued in November 2008. This confirmatory testwork confirmed the results of the previous studies and projected average weight recoveries of 38% without a magnetite circuit and 41% with the inclusion of a magnetite recovery circuit. The average iron feed to achieve these recoveries are 31% crude iron.

11.1 Testing and Procedures

11.1.1 Lakefield Research (1973 to 1976)

The initial metallurgical testwork was carried out by Lakefield Research during the period 1973 to 1976 on 17 drill core composites. Nine samples were taken from the North-East Extension and eight samples were from the West Extension. Four other samples of magnetite-silicate waste from an area east of the North-East Extension were also tested. The results of these tests are summarized in Table 11.1.1.1 and demonstrate that about 83% of the soluble iron could be recovered into a gravity concentrate containing about 38% of the weight.

Table 11.1.1.1: Summary of Metallurgical Test Results on Bloom Lake Drill Core Composites – Lakefield Research 1973 to 1976

Composite No.	Drillhole	Head		Table Concentrate			Table + Mag Cone.		
		% Sol Fe	% Mag Fe	Wt. %	% Sol Fe	% Dist. Sol Fe	Wt %	% Sol Fe	% Dist. Sol Fe
North-East Extension 1-5, 12-15	71-1,-2	30.26	13.02	38.1	66.3	83.4	52.8	66.6	94.6
West Extension 6-10, 19-21	71-3, -6, -8,-9,-10	31.4	0.74	38.1	67.9	82.6			
Mag-silicate Waste 11,16-18	72-2, -3,	16.1	12.5	18.3	67.8	75.2	21.5	68.2	89.2

11.1.2 Lakefield Research (2005)

During 2005 eleven mini-bulk samples were taken from outcrops and sent to SGS Lakefield for testing, which consisted of Wilfey table tests followed by cleaning of the Wilfley concentrate on a Mozely table. The results of these tests confirmed that, in most cases, a low silica concentrate could be produced with a high rejection of actinolite (an iron-bearing magnesium silicate that is rejected during gravity concentration).

In order to assess the impact of recycle streams on gravity concentrate quality and total iron recovery, a six-stage locked-cycle test was conducted on two separate composites (Composite A and Composite B) formulated from the 11 mini-bulk samples. For each stage of the locked-cycle test a 15 kg composite sample was ground to 425 microns (35 mesh) and treated on a Wilfley shaking table to produce a rougher gravity concentrate, which was then upgraded on a second Wilfley table to produce a final cleaner concentrate. The rougher middlings and cleaner tailings were then combined with new feed and fed to the next succeeding gravity roughing stage. At the end of the 6 stages of testing tailings from cycles 3 to 6 were combined for magnetic separation tests and concentrates from cycles 3 to 6 were combined as final gravity concentrate. The results of these locked-cycle tests are summarized in Table 11.1.2.1. For composite A, 59.9% of the iron was recovered into a gravity concentrate containing 67% Fe. For composite B, 85.5% of the iron was recovered into a gravity concentrate containing 66% Fe. It is not clear why significantly lower iron recoveries were obtained from Composite A, but it is noted that composite A was lower grade than Composite B and contained a significantly lower percentage of hematite and a higher percentage of magnetite than Composite B. Additional magnetic separation tests on composite A gravity tailings resulted in the production of a high grade magnetic concentrate containing about 68% Fe and the combined gravity and magnetic concentrates resulted in an overall iron recovery of about 75% from composite A. No magnetic separation tests were conducted on gravity tailings from Composite B.

Table 11.1.2.1: Summary of Locked-Cycle Tests conducted on Two Test Composites Formulated from Eleven Mini-Bulk Samples – SGS Lakefield 2005

Product			Assay %					% Distribution				
	Kg	Weight %	Fe	SiO_2	MgO	Ht$^{(1)}$	Mt$^{(2)}$	Fe	SiO_2	MgO	Ht$^{(1)}$	Mt$^{(2)}$
Comp. A												
Concentrate	24.9	28.4	67.0	3.5	0.3	71.6	23.5	59.9	2.0	5.0	65.8	47.4
Tails	61.5	70.1	17.4	68.4	266.0	14.1	10.5	38.5	96.3	94.9	32.0	52.1
Head (Calc)	87.7	100.0	31.8	49.9	2.0	30.9	14.1	100.0	100.0	100.0	100.0	100.0
Comp. B												
Concentrate	37.4	43.0	66.1	5.6	0.5	75.5	18.4	85.5	5.0	11.4	91.2	68.5
Tails	50.3	57.9	9.8	78.9	3.1	7.6	6.2	17 0	95.3	89.1	12.3	31.0
Head (Calc)	87.6	100.9	33.8	47.9	2.0	35.6	11.6	102.5	100.3	101.0	103.5	99.5

Source: SGS Lakefield, 2005
(1) Ht=hematite, Fe_2O_3
(2) Mt=magnetite, Fe3O4

Weight % and Iron Recovery Projection

Based on the results of testwork conducted on the 11 mini-bulk samples, the weight recovery versus head grade for gravity concentrates (no magnetite recovery circuit) in the range of 65-66.5% Fe was established to be:

- Weight % Recovery = 1.3788 * X – 3.1746; and
- Where: X = Head grade, %Fe.

With the magnetic separation circuit included, the overall weight recovery was estimated to be:

- Weight % Recovery = 1.3788 * X.

It was recognized that the Bloom Lake deposit contains actinolite and that actinolite is estimated to contain 4.12% Fe and that for each percent of MgO, there is 0.194% Fe occurring as non-recoverable iron contained in actinolite.

On this basis the weight recovery relationship was adjusted as follows to account for the presence of actinolite:

- Weight % Recovery = 1.3788 * X – 0.194 * %MgO * 100/%FeConc – 3.1746

And iron recovery was described by the relationship:

- Fe Recovery% = 1.3788 * %FeConc. - %MgO * 0.194 – 3.176

On the basis of average ore grades of 30% Fe and 1.9% MgO and the production of a 66% Fe concentrate, these relationships would predict a weight recovery of 37.6% and an iron recovery of 87% without consideration of a magnetite recovery circuit.

The following recovery relationships were developed for a process flowsheet that included a magnetite recovery circuit:

- Weight % Recovery = 1.3788 * X – 0.194 * %MgO * 100/%FeConc; and
- Fe Recovery% = 1.3788 * %FeConc. - %MgO * 0.194

Based on average ore grades these equations would project the 90.3% iron recovery and 40.1% weight recovery.

11.1.3 Confirmatory Testing

Confirmatory metallurgical tests were conducted on 32 drill core samples from four areas of the Bloom Lake pit at different levels. The gravity separation tests were conducted at a 425 micron grind size and the results are summarized in Table 11.1.3.1. For most samples, the results of the rougher stage are reported, as the grades and recoveries are high and no further processing was necessary. In the cases where the silica content of the rougher concentrates was 5% or higher, the results of further upgrading tests are reported. The average grade of the drill cores tested was 29.8% Fe, 1.94% MgO and 10.3% Fe_3O_4, which is close to average grade of the deposit. The average result of these tests was the production of a gravity concentrate containing 67.7% Fe and with an 88.3% iron recovery and 38.8% concentrate weight recovery.

Metallurgical studies conducted on the gravity tailings concluded that an additional iron recovery of 2.2% and 1% weight recovery could be achieved by incorporation of a magnetite recovery circuit. This would bring the total estimated iron recovery to about 90.5% and estimated weight recovery to about 40%. The confirmatory test results agreed quite well with the recovery projections derived from the testwork conducted on the 11 mini-bulk composites. The concentrate grades and Fe recoveries achieved in these tests were significantly better than those achieved in the metallurgical tests carried on mini-bulk surface samples in 2005 and 2006. The average concentrate grade and Fe recovery for the mini-bulk samples were about 65% and 83%, respectively. The average weight recovery was slightly higher than the 38% achieved with the mini-bulk samples tested in 2005 and 2006. In both series of tests, the ore samples were ground to -35 mesh. The drill core results seem to suggest that the Fe was well liberated in most cases and may not have required such a fine grind as in the earlier tests carried out in 2005 and 2006[2]. The main differences between the two series of tests are that the mini-bulk samples were surface samples and were taken mainly from the northeast and west sections of the pit, whereas the drill cores samples were taken in all areas of the pit and at different depths.

Table 11.1.3.1: Summary of Confirmatory Test Results on 32 Drill Core Test Samples

Product Type	Sample ID	Location	From (m)	Head Assay			Concentrate assay				Recovery/Rejection		
				%Fe	% MgO	Mag Fe	%Fe	% SiO_2	% MgO	% Fe_3CU	Fe Rec	MgO Rej	Wt Rec
Rougher cone.	3053	Central	80	29.7	0.7	6.7	68.1	2.4	0.2	12.8	91.7	89.4	407.0
Rougher cone.	3054	Central	89	21.7	1.0	13.8	68.6	2.7	0.3	42.6	86.8	92.2	27.0
Rougher cone.	3055	Central	98	30.8	1.2	7.0	68.2	3.0	0.2	14.9	89 3	92.4	38.2
Cl. Cone. + Mid 2-1	3058	Central	141	42.8	0.3	2.0	66.3	4.7	0.1	2.4	93.6	77.5	59.2
Rougher cone.	3059	Central	150	43.6	1.9	3.1	67.6	1.8	0.3	4.0	96.1	89.6	63.7
Cl. Cone. + Mid 2-1	3061	Central	67	31.3	0.4	10.2	68.3	2.1	0.1	21.1	82.5	90.8	36.7
Rougher cone.	3063	Central	75	34.6	0.1	4.0	66.0	4.3	0.1	7.3	909.0	80.9	426.0
Rougher cone.	3066	Central	102	31.7	0.7	35.2	68.5	4.2	0.1	72.5	84.9	93.1	38.2
Cl. Cone.	3070	Northeast	140	25.5	3.2	21.2	66.7	3.5	0.7	59.3	81.7	93.0	28.9
Rougher cone.	3071	Northeast	203	29.0	3.7	5.0	68.5	1.4	0.3	11.5	87.3	97.3	37.0
Cl. Cone. + Mid 2-1	3076	Southeast	228	23.2	5.1	0.6	64.3	3.9	1.1	1.6	86.4	91.4	34.3
Rougher cone	3083	Southeast	240	23 3	3 3	30 8	68 2	4.0	0.5	95.8	82.5	95.0	30.0
Rougher cone	3084	Southeast	246	1.4	4.4	21 8	69 2	1.7	0 4	48 8	91.0	96.5	40 9
Rougher cone.	3085	Southeast	264	20.4	2.7	9.0	69.5	1.4	0.2	30 2	91.1	97,2	29.8
Cl. Cone.	3089	Southeast	99	32.9	0.7	40.9	67.9	5.0	0.2	86 8	70.1	90.9	34.7
Rougher cone.	3090	Southeast	105	35.0	2.0	25.7	69.2	1.5	0.2	52.3	92.7	94.5	50.9
Rougher cone.	3092	Southeast	123	41.0	1.0	8.6	68 9	l 42	0.1	14 5	92. 8	94.9	50.0
Rougher cone.	3094	Southeast	243	30.7	4.5	1.2	65.7	3.4	1.1	2.1	89.7	89.6	44.7
Cl. Cone. + Mid 2-1	3099	West	99	33.4		0.7	67.8	2.9	0.1	0.8	81.5		40.9
CL Cone. + Mid 2-1	3100	Northeast	30	20.8	1.7	24 8	665.0	5.0	0 5	83 7	81 7		20 7
Rougher cone	3101	Northeast	98	23 7	0 4	6 1	656.0	4 9	0 1	12 6	92.0	86.8	35.0
Rougher cone	3106	West	50	29.9	5.0	0.2	681.0	2 4	0 02	0 7	92.0		39.0
Rougher cone	3108	West	74	32.3	0.1	46.0	673.0	3.2	0 02	0.6	93.3		44.4
Rougher cone.	3109	West	.30	25.5	0.2	0.4	67.6	2.6	0.0	0 7	90.8	94.8	34.3
Rougher cone.	31144	Northeast	134	24.7	3.9	3.2	68 8	0.6	0.1	7.0	90.7	98.7	35,3
Rougher cone.	3115	Northeast	143	25.0	3.1	2.3	68.1	0.8	0.2	5.0	88.3	93.6	28.2
Rougher cone.	3116	Northeast	185	25.7	3.8	0.9	09 3	0.5	0 1	2.1	90.5	93.9	33.4
Rougher cone.	3118	Northeast	200	32.7	1.9	2.6	67.6	2.1	0.3	4.8	91.7	93.1	39.9
	Average			**29.8**	**1.9**	**10.3**	**67.7**	**2.8**	**0.3**	**25.0**	**88.3**	**92.4**	**38.8**

11.1.4 Pilot Plant Testing For the Phase II Concentrator

During 2010 and 2011, pilot tests were conducted in the existing Phase I concentrator to validate alternate flowsheets or equipment for the Phase II concentrator. Figure 11.1.4.1 shows the four different process unit operations that were evaluated, which included:

- Classification hydrosizer;
- Rougher Spirals with current feed;
- Rougher Spirals with hydrosizer overflow as feed; and
- Cleaning hydrosizer and scavenger spirals.

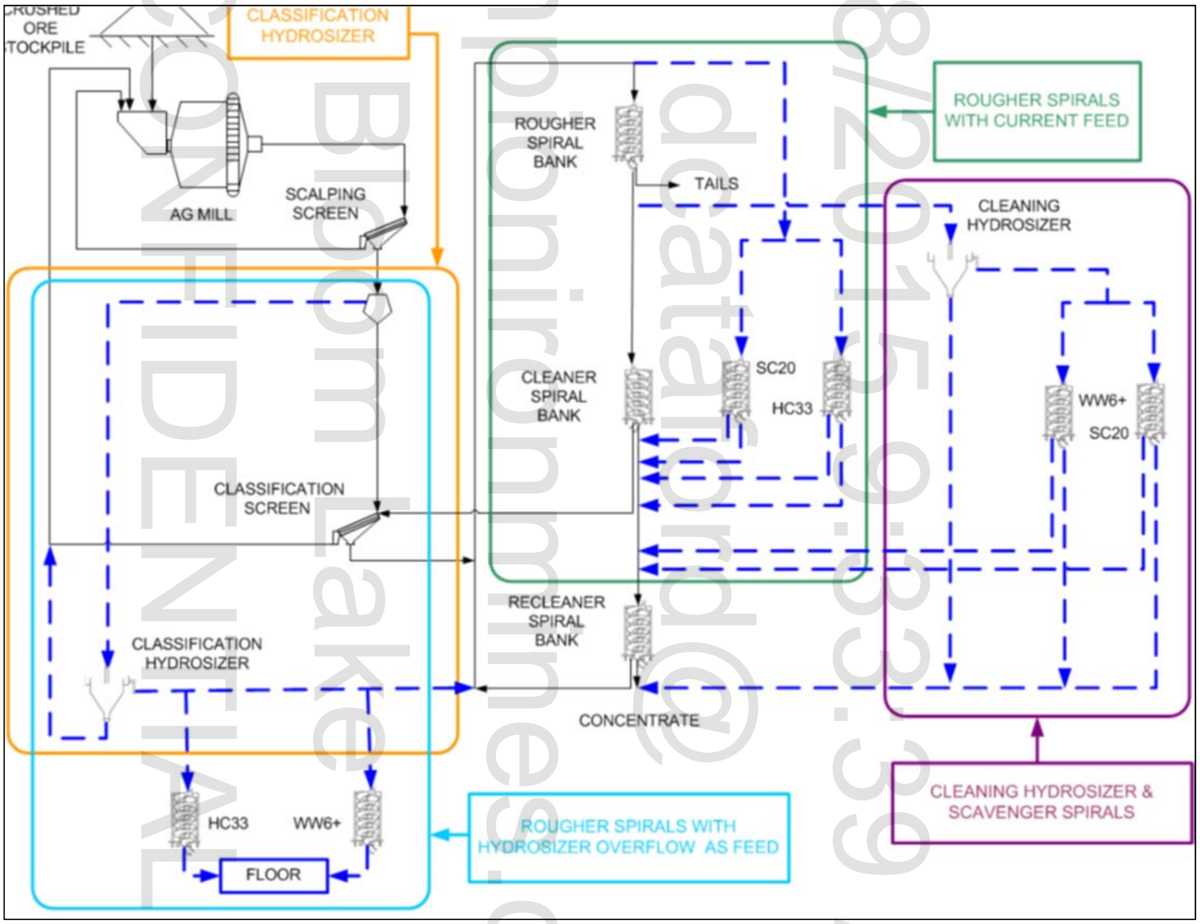


Figure 11.1.4.1: Flowsheet Showing the Four Separate Pilot Plant Flow Configurations That Were Tested

Classification Hydrosizer

The pilot test results showed that it is possible to use hydrosizers to separate small and light particles from large and dense particles contained in the scalping screen undersize. The pilot test results indicated that the hydrosizer can separate particles at an average size (separation size – d50) of 650 μm. A separation size of around 300 μm was observed for the iron particles, while a separation size of around 720 μm was observed for the silica particles. The combined separation effects of the particle density and the particle size of a hydrosizer at the classification stage permits the:

- Elimination of the liberated minus 600 µm silica particles from the hydrosizer underflow;
- Recovery at the underflow of the plus 600 µm silica particles, which can be recirculated back the grinding circuit to achieve improved liberation; and
- Recovery at the underflow of most of the hematite-magnetite particles coarser than 300 µm.

The classification hydrosizer underflow density was consistent around 80% solids and the iron grade varied from 44% to 60% Fe with an average of 52% Fe. A particle size analysis of the hydrosizer underflow showed that particles smaller than 850 µm are of final concentrate quality (≥66% Fe). Additionally, when considering a separation size of 600 µm (by screening the underflow), the resulting concentrate grade was found to be over 67% Fe. Thus, screening of the underflow at 600 µm would allow the production of a final concentrate at the classification stage. It should be noted, however, that screening at 600 µm allows some non-liberated particles to pass through the screen. Considering that the particles smaller than 425 µm are well liberated as demonstrated in the historical testwork, the particles in the range of 425 µm to 600 µm could be as low as 64.5% Fe and 7.8% SiO_2 and still produce a concentrate of final quality when mixed with the lower fractions.

The hydrosizer overflow contains a finer size range of iron particles, which are subsequently recovered in a spiral gravity concentration circuit. The results of this phase of the pilot plant evaluation concluded that a combination of classification hydrosizers and classification screens can replace the classification screens currently used in the Phase I concentrator. This circuit has the advantage of producing a final concentrate when screening the hydrosizer underflow at 600 µm, thus reducing the load at the gravity circuit and ensuring the recovery of the coarse hematite-magnetite particles.

Roughers Spirals – with Actual Feed

Pilot plant tests were conducted to evaluate the performance of the existing Outotec H9000 rougher spirals and to compare with the performance of two alternative spiral models which included the HC33 spiral manufactured by Downer Minerals Technologies, the SC20 model manufactured by Multotec. This series of pilot tests were conducted in parallel using the actual spiral feed. As shown in Table 11.1.4.1, the results showed that the H9000 spirals produced the lowest mass recovery, and lowest unit iron recovery, while the HC33 offered the best overall performance.

Table 11.1.4.1: Summary of Rougher Spiral Tests Using Actual Phase I Concentrator Feed

Spiral Model	Concentrate		
	Iron Grade (%)	Mass Recovery (%)	Iron Recovery (%)
HC33	54.7	53.7	87.0
SC20	56.2	51.0	85.7
H9000	56.5	49.8	83.9

Roughers Spirals – with Hydrosizer Overflow as Feed

A second series of pilot tests were conducted using overflow of the classification hydrosizer as the spiral feed. Two different spiral models (HC33 and WW6+), both manufactured by Downer Minerals Technologies, were tested. The results of these tests are summarized in Table 11.1.4.2 and demonstrated that much better performance was obtained with the WW6+ spiral. The WW6+ spiral was able to recover 83.8% of the iron contained in the hydrosizer overflow at a grade of 55.4% Fe.

Table 11.1.4.2: Summary of Rougher Spiral Tests Using Hydrosizer Overflow as the Feed

Spiral Model	Concentrate		
	Iron Grade (%)	Mass Recovery (%)	Iron Recovery (%)
HC33	52.7	31.8	66.5
WW6+	55.4	41.8	83.8

Cleaning Hydrosizer and Scavenger Spirals

The final series of pilot tests included evaluation of a hydrosizer at the cleaning stage of the separation process. The hydrosizer feed was the rougher spiral concentrate produced from the classification hydrosizer overflow. The cleaner hydrosizer underflow resulted in the production of a final grade iron concentrate. The results of this work showed that the cleaning hydrosizer can produce a 67% Fe concentrate with a unit iron recovery of 65-70%.

The hydrosizer overflow was upgraded by scavenger spirals to produce a final grade iron concentrate. The two spiral models tested included the SC20 and the WW6+. As shown in Table 11.1.4.3, the WW6+ spiral outperformed the SC20 spiral with a unit iron recovery of 86.3% at a concentrate grade of 63.3%.

Table 11.1.4.3: Summary of Scavenger Spiral Tests on Cleaner Hydrosizer Overflow

Spiral Model	Concentrate		
	Iron Grade (%)	Mass Recovery (%)	Iron Recovery (%)
SC20	64.8	45.8	71.9
WW6+	63.3	55.8	86.3

11.1.5 Overall Phase II Flowsheet and Material Balance

Based on the results of pilot plant test program, the process flowsheet shown in Figure 11.1.5.1 has been adopted for the Phase II concentrator. The flowsheet includes scalping of the Autogenous (AG) mill discharge at 5 mm with a vibrating screen. The minus 5 mm screen undersize fraction will be fed to a series of classification hydrosizers. The underflow from the classification hydrosizers will then be screened at 600 microns to produce the first of three finished iron products. The classification hydrosizer overflow will then be subjected to gravity rougher concentration with rougher spirals. The rougher gravity concentrate will be fed to the cleaning hydrosizer where the underflow represents the second of three final iron products. The overflow from the cleaning hydrosizer will be upgraded with scavenger spirals and the resulting scavenger gravity concentrate will represent the third of three finished iron products.

Table 11.1.5.1 provides a summary of the iron recoveries and concentrate grades that have been projected based on the results of the pilot plant investigation. An overall iron recovery of 84.1% into a final concentrate containing 66.5% is projected. A weight% recovery of 37.9% to the combined final concentrates is projected from a feed grade of 30% Fe.

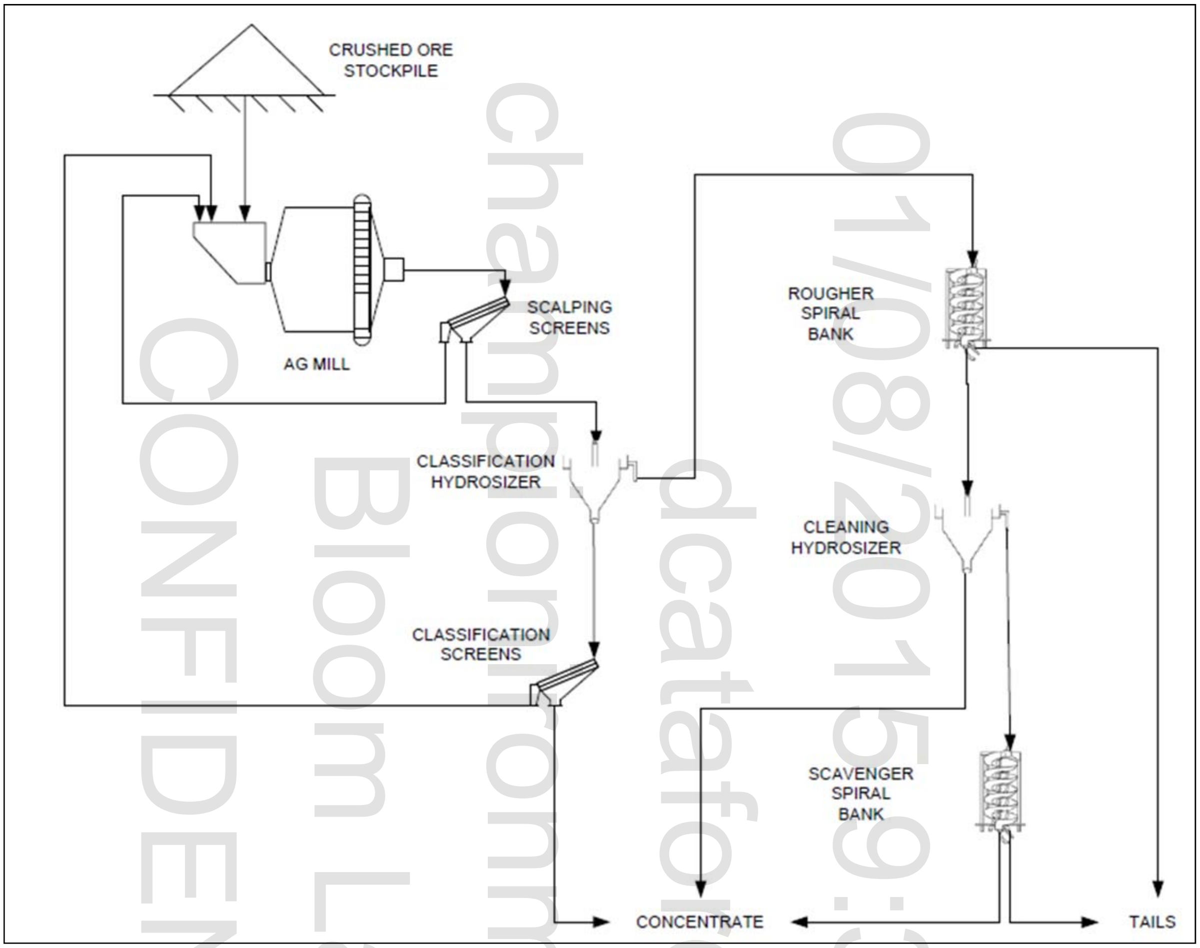


Figure 11.1.5.1: Phase II Concentrator Flowsheet Based on Pilot Plant Test Results

Table 11.1.5.1: Phase II Concentrator Material Balance Based on Pilot Plant Test Results

Stream	Grade		Flow Rate			Recovery		
	Fe (%)	SiO_2 (%)	Solid (t/h)	Fe (t/h)	SiO_2 (t/h)	Mass (%)	Fe (%)	SiO_2 (%)
Fresh Feed	30.0	57.1	2,589.2	776.8	1,478.4	100.0	100.0	100.0
Tailings								
Rouqher Spiral Tailings	7.1	90.2	1417.5	100.1	1278.2	54.7	12.9	86.5
Scavenqer Spiral Tailings	12.3	82.4	189.2	23.3	155.8	7.3	3.0	10.5
Final Tailings	7.7	89.3	1606.6	123.5	1434.0	62.1	15.9	97.0
Concentrate								
Classification Screen Undersize	67.6	2.8	293.3	198.4	8.1	11.3	25.5	0.5
Cleaninq Hydrosizer Underflow	67.4	3.5	461.3	311.0	16.0	17.8	40.0	1,1
Scavenqer Spiral Concentrate	63.1	8.9	227.9	143.9	20.3	8.8	18.5	1.4
Final Concentrate	66.5	4.5	982.5	653.3	44.4	37.9	84.1	3.0

12 Mineral Resource Estimate

This section provides details in terms of key assumptions, parameters and methods used to estimate the mineral resources together with SRK's opinion as to their merits and possible limitations. The resource estimation for the Bloom Lake Mine was prepared Mr. Peter Jongewaard, Cliffs Senior Staff Geologist. The geologic model was prepared by Gemcom International, Inc. (Gemcom) consultants in conjunction with Bloom Lake geologists. Mr. Jongewaard used Maptek Vulcan software to complete the resource estimation and the same estimation methodology used by SRK in 2011. The updated resource was reviewed by Ms. Leah Mach, SRK Principal Resource Geologist.

12.1 Drillhole Database

The drillhole data base consists of 110,681 m of drilling in 463 core holes within the block model limits. The minimum depth is 17 m and the maximum is 720 m; the average depth is 239 m. The drillholes are oriented to be as close to perpendicular to the iron formation as possible.

Figure 12.1.1 is a drillhole location map showing the drillholes in the Bloom Lake deposit.

The database was exported from Gemcom's GEMS software into four csv files, including:

- Collar: Drillhole identification (DHID), X, Y, Z, Total depth and year drilled;
- Survey: DHID, azimuth, inclination, depth of survey;
- Assay: DHID, From, To, Length, Total Fe% (FeT), Fe% in Magnetite (MagFe), SiO_2%, Al_2O_3%, MgO%, CaO%, Na_2O%, K_2O%, TiO_2%, MnO%, P_2O_5%, LOI%, S_ppm, V_2O_5%, Cr_2O_3%,Specific Gravity, Calculated Specific Gravity; and
- Geology: DHID, From, To, Length, Lithology code and Description.

There are 10,082 sample intervals in the database which were analyzed for Fe_2O_3, with an average length of 4.68 m. FeT% was calculated as 0.699 X Fe_2O_3. A total of 10,046 samples were analyzed for MagFe and about 5,700 were analyzed for the remaining elements and oxides. Statistics for the entire database within the block model limits are shown in Table 12.1.1

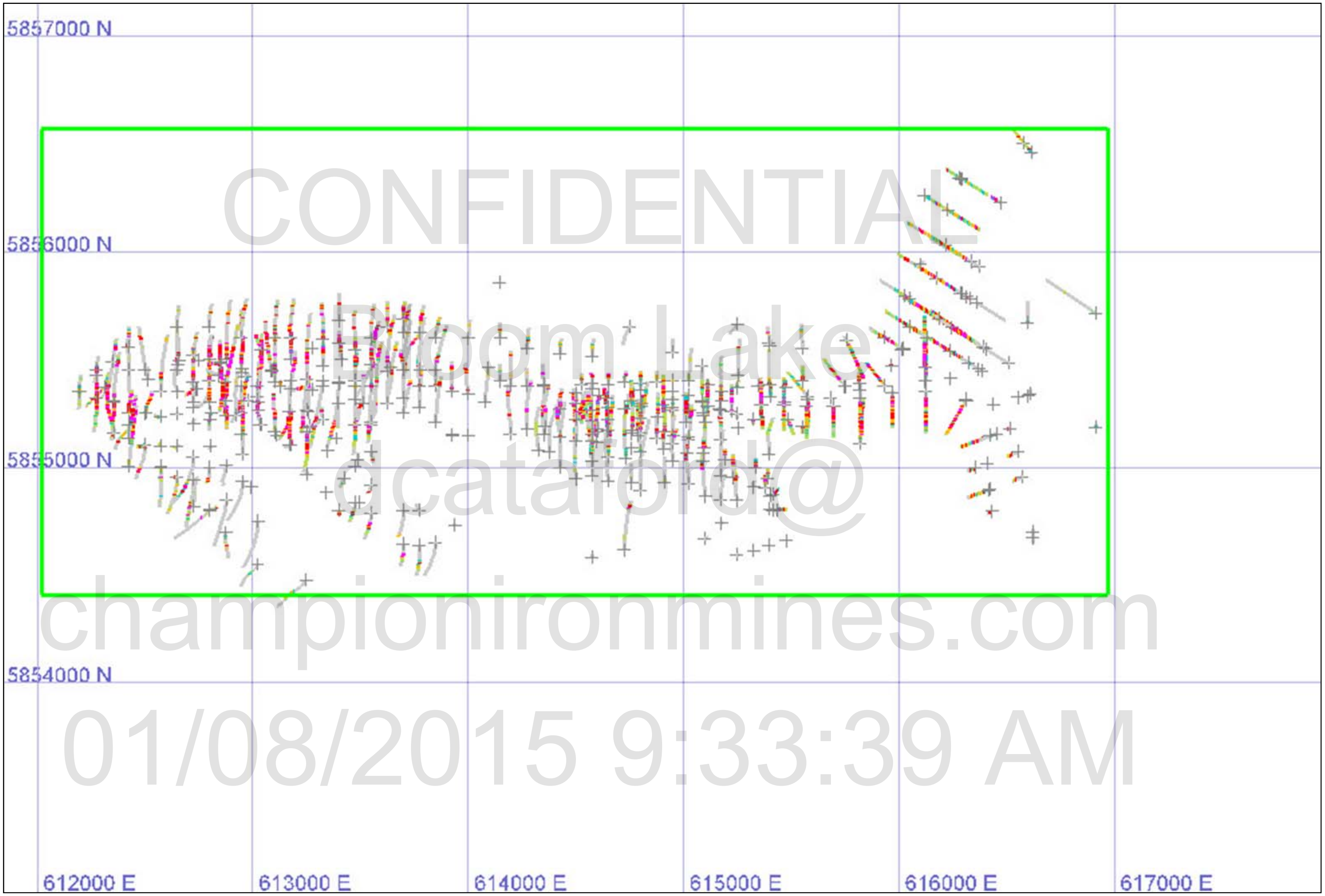


Figure 12.1.1: Drillhole Location Map and Original Topography

Table 12.1.1: Bloom Lake Assay Statistics

	Length	FeT	MagFe	SiO2	Al2O3	MgO	LOI
Number	10746	10082	10046	8249	8238	8564	7373
Minimum	0.26	0.47	0.01	1.57	0.01	0.01	-2.33
Maximum	35.30	68.74	61.20	100.00	22.30	17.10	17.60
Mean	4.69	26.44	4.04	56.82	1.25	1.40	1.00
Median	5.00	29.02	1.10	54.00	0.17	0.07	0.39
Std. Dev.	1.68	10.60	7.17	13.81	3.49	2.39	1.73
CV	0.36	0.40	1.78	0.24	2.80	1.70	1.73

12.2 Geologic Model

The geologic model was produced by Gemcom in conjunction with Bloom Lake). The interpretation was based on diamond drillholes, geological maps, ground magnetic surveys and production data. Cross-sections were generated at 75 m spacing west to east with some sections to the east oriented northwest or southwest depending on the orientation of the geologic structures. Bloom Lake geologists interpreted the geology on the cross-sections and then digitized them into GEMS software. The geological units were gneiss, quartzite, mica schist, specularite-magnetite-actinolite iron formation (IF), silicate iron formation (SIF) and amphibolite. The units IF and SIF are iron formation and used in the resource model.

The geological interpretations were transferred from the cross-sections to plan sections and digitized into GEMS. The plan sections are on 14 m bench heights from the top of the mine to bench 417 and then 28 m bench heights to bench 193. The polygons were interpreted at the middle of the bench and then extruded to the full bench height to make solids. The following lithologies were modeled:

- Overburden (MT), block model code 10;
- Iron Formation (IF), block model code 20;
- Silicate Iron Formation (SIF), block model code 23;
- Quartzite (QR), block model code 30;
- Quartz mica schist (QRMS), block model code 32;
- Amphibolite (AMP), block model code 40; and
- Gneiss (GN), block model code 50.

Figure 12.2.1 is a plan view at elevation 625 showing the lithology model.

Structural domains were modeled according to the strike and dip of the rock units. Figure 12.2.2 shows the structural domains at elevation 625. The IF/SIF iron formations were further coded into lith-domains as follows:

- 21 – IF, Structural Domain 1;
- 22 – IF, Structural Domain 2;
- 23 – IF, Structural Domain 3;
- 24 – IF, Structural Domain 4;
- 25 – IF, Structural Domain 5;
- 26 – IF, Structural Domain 6;
- 61 – SIF, Structural Domain 1;
- 63 – SF, Structural Domain 3;

- 64 – SIF, Structural Domain 4;
- 65 – SIF, Structural Domain 5; and
- 66 – SIF, Structural Domain 6.

Figure 12.2.3 shows the lith-domains at elevation 625. There is no lith-domain 62 – SIF in Structural Domain 2 - as there are no drillholes in that area. Statistics of the assays within each litho-domain are shown in Table 12.2.1.

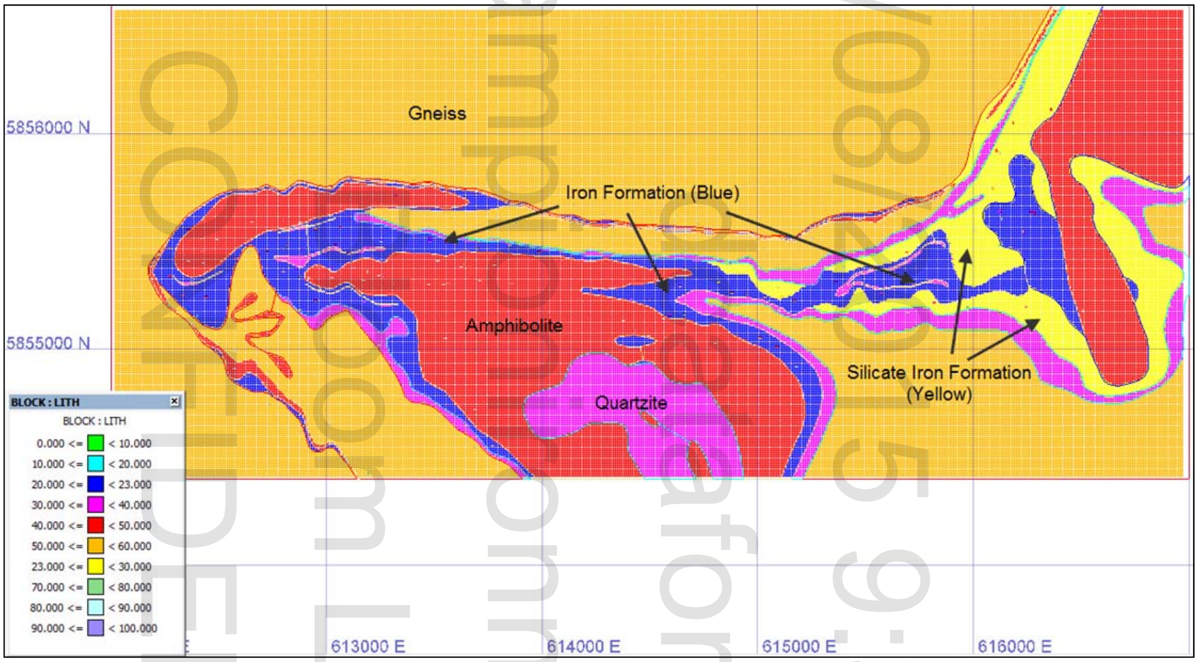


Figure 12.2.1: Elevation 625 Lithology Model

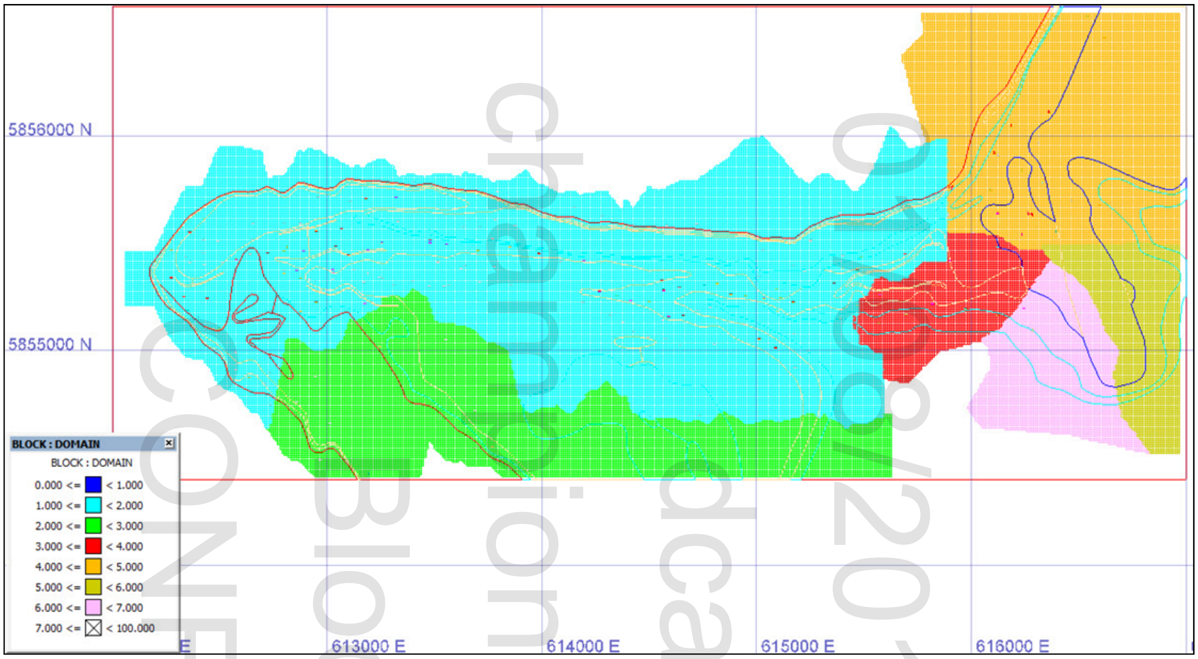


Figure 12.2.2: Elevation 625 Structural Domains

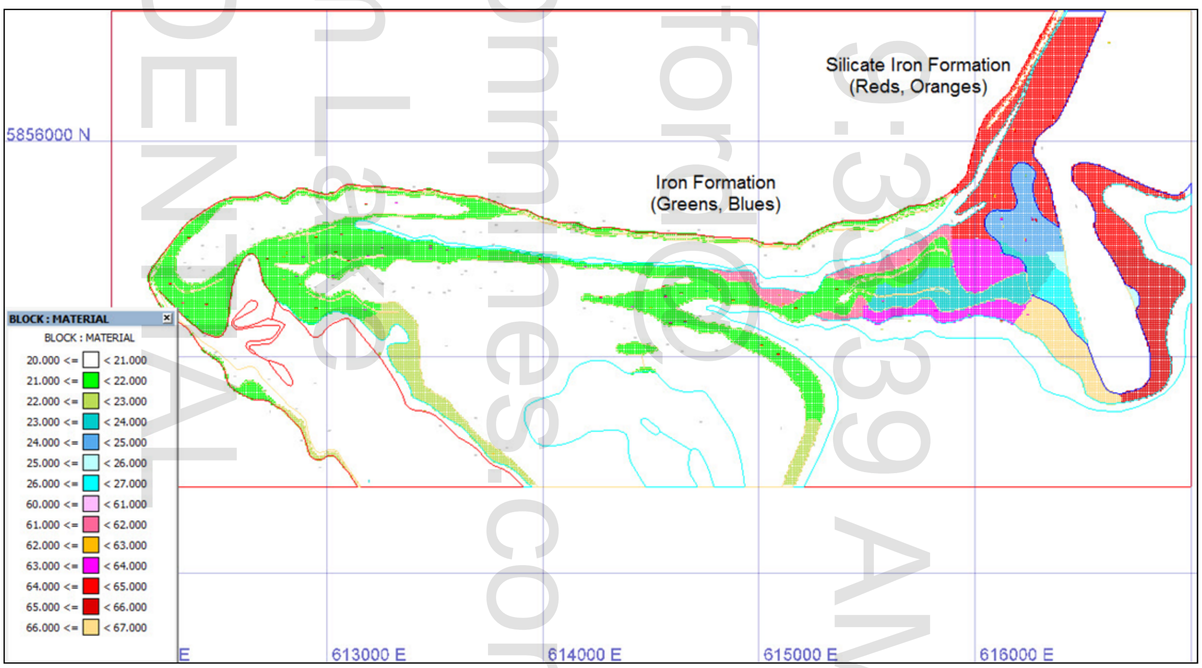


Figure 12.2.3: Elevation 625 Litho-Structural Domains

Table 12.2.1: Statistics of Assays within Litho-Domains

Metal	Domain	All	21	22	23	24	25	26	61	63	64	65	66
FeT	Samples	10082	4801	469	500	455	10	93	314	340	865	45	278
	Minimum	0.47	1.54	1.15	3.41	1.80	5.20	6.86	1.49	2.59	1.28	2.80	1.50
	Maximum	68.74	68.74	60.77	49.20	53.80	34.41	44.00	47.11	46.30	52.40	36.37	44.26
	Mean	26.11	29.34	28.24	30.54	27.26	18.97	30.27	20.41	27.07	23.71	16.22	26.87
	Median	28.66	31.07	29.72	31.91	29.59	21.40	32.73	19.64	28.89	25.10	16.40	28.45
	Std Dev	10.74	9.39	8.63	7.91	9.55	9.40	8.50	9.18	7.71	8.53	8.95	8.24
	CV	0.41	0.32	0.31	0.26	0.35	0.50	0.28	0.45	0.29	0.36	0.55	0.31
MagFe	Samples	10046	4789	468	496	453	9	89	314	340	858	45	276
	Minimum	0.01	0.07	0.07	0.05	0.17	0.45	0.49	0.01	0.02	0.04	1.63	0.49
	Maximum	61.20	61.20	24.89	40.60	50.80	22.09	40.90	44.35	45.30	50.90	30.80	43.93
	Mean	4.21	2.04	0.90	7.21	11.73	9.04	19.33	8.31	6.70	7.97	10.20	11.78
	Median	1.10	0.70	0.44	1.78	7.06	7.30	18.30	5.40	3.27	4.02	9.30	7.39
	Std Dev	7.37	4.58	2.07	10.28	11.35	7.63	11.63	8.51	8.25	8.83	6.48	11.12
	CV	1.75	2.25	2.30	1.43	0.97	0.84	0.60	1.02	1.23	1.11	0.64	0.94
SiO_2	Samples	8249	4286	393	313	294	10	80	246	238	459	16	213
	Minimum	1.57	2.73	8.02	26.20	32.00	44.10	33.50	28.30	28.70	23.80	43.00	31.70
	Maximum	100.00	98.00	98.20	93.10	94.90	89.80	82.20	95.20	91.80	93.90	93.90	95.50
	Mean	57.10	56.16	58.42	52.75	54.47	65.05	52.66	51.92	51.08	51.33	66.03	52.65
	Median	54.20	54.50	56.50	50.50	51.50	67.10	50.50	48.10	48.60	48.80	65.30	51.20
	Std Dev	14.06	12.52	12.41	10.95	12.27	16.43	11.18	13.72	11.26	10.76	15.05	10.79
	CV	0.25	0.22	0.21	0.21	0.23	0.25	0.21	0.26	0.22	0.21	0.23	0.21
Al_2O_3	Samples	8238	4277	393	313	294	10	80	246	238	459	16	213
	Minimum	0.01	0.01	0.01	0.01	0.01	0.12	0.01	0.01	0.01	0.01	0.01	0.01
	Maximum	22.30	18.30	17.20	14.40	16.30	14.20	13.40	8.11	14.20	21.40	0.27	14.60
	Mean	1.37	0.76	0.62	0.39	0.44	3.04	0.73	0.28	0.70	1.10	0.12	1.07
	Median	0.17	0.15	0.15	0.16	0.13	0.31	0.21	0.11	0.17	0.19	0.10	0.18
	Std Dev	3.68	2.61	2.19	1.45	1.95	5.86	2.21	0.69	1.94	2.91	0.07	2.98
	CV	2.69	3.43	3.55	3.69	4.39	1.93	3.03	2.46	2.78	2.65	0.58	2.79
MgO	Samples	8564	4483	469	313	294	10	80	246	238	459	16	213
	Minimum	0.01	0.01	0.01	0.02	0.08	0.40	0.17	0.03	0.05	0.19	0.07	0.14
	Maximum	17.10	16.20	7.40	9.04	9.71	6.67	9.08	13.70	10.50	13.30	5.91	10.70
	Mean	1.39	0.46	0.27	1.63	2.33	1.81	2.10	6.74	3.71	4.15	1.88	2.94
	Median	0.07	0.06	0.06	1.05	2.07	0.81	1.58	7.13	3.41	3.69	1.41	2.64
	Std Dev	2.41	1.43	1.03	1.75	1.75	2.29	1.78	3.70	2.17	2.42	1.74	1.72
	CV	1.73	3.12	3.78	1.07	0.75	1.26	0.85	0.55	0.59	0.58	0.93	0.59
CaO	Samples	8560	4481	469	313	293	10	80	246	238	459	16	213
	Minimum	0.01	0.01	0.01	0.01	0.10	0.45	0.09	0.01	0.01	0.09	0.09	0.07
	Maximum	18.70	14.70	9.07	11.60	15.00	7.87	7.09	18.70	15.20	16.20	9.26	8.68
	Mean	1.44	0.37	0.35	1.47	2.54	2.25	1.84	7.80	4.15	4.99	2.39	2.82
	Median	0.05	0.02	0.02	0.72	2.17	0.98	1.17	6.95	3.69	4.36	2.45	2.33
	Std Dev	2.77	1.33	1.31	1.89	1.90	2.91	1.74	5.45	2.71	3.21	2.24	1.99
	CV	1.92	3.64	3.79	1.28	0.75	1.29	0.95	0.70	0.65	0.64	0.94	0.71

12.3 Compositing

The sample interval length varies from 0.26 to 35.3 m with an average of 4.69 m. Figure 12.3.1 is a histogram of the sample lengths and Figure 12.3.2 is a lognormal probability plot of sample lengths. About 98% of the samples are less than 7 m in length and therefore Cliffs composited the samples into 7 m lengths from the top of the drillhole with breaks at the IF or SIF solid boundary. Samples from domain 25 were combined with domain 26 and samples from domain 65 were combined with domain 66 because there are relatively few samples in each of those domains. Samples less than 2 m in length were added to the previous composite. This would happen only where the drillhole exits the IF or SIF solid. The litho-domain code was assigned to the composite from the block model and the IF/SIF code was assigned from the solids. Table 12.3.1 presents statistics of the composites by litho-domain.

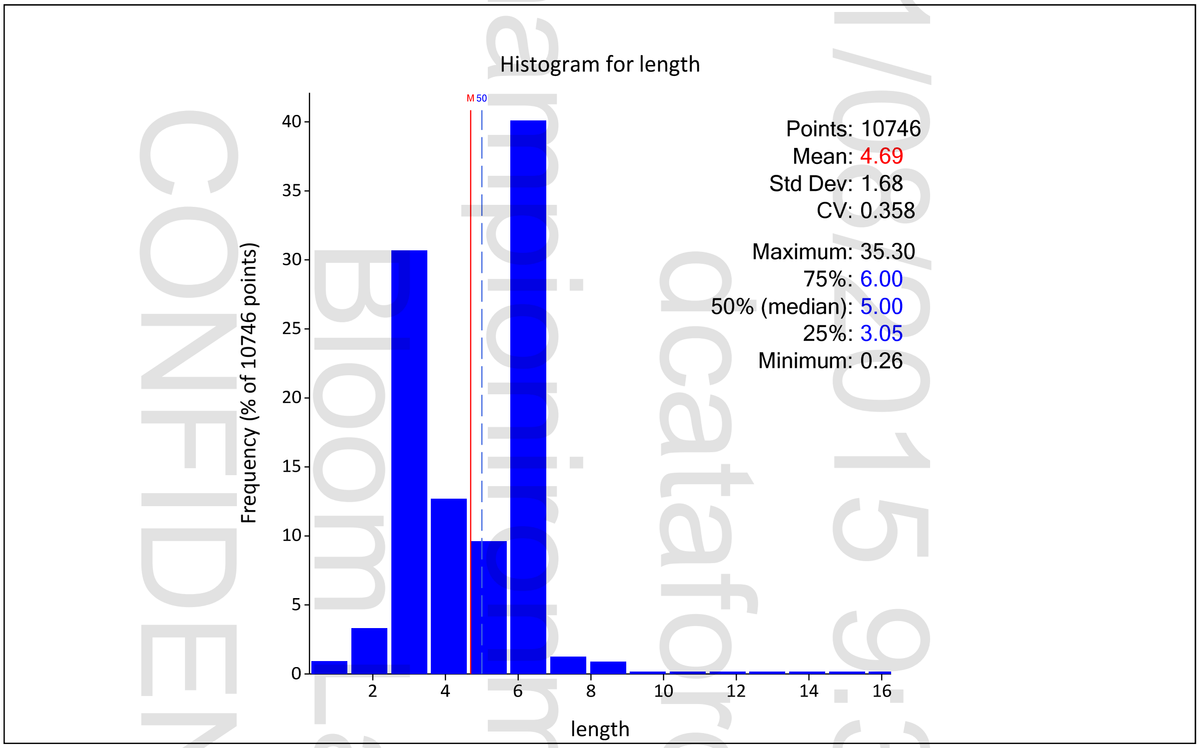


Figure 12.3.1: Histogram of Sample Lengths

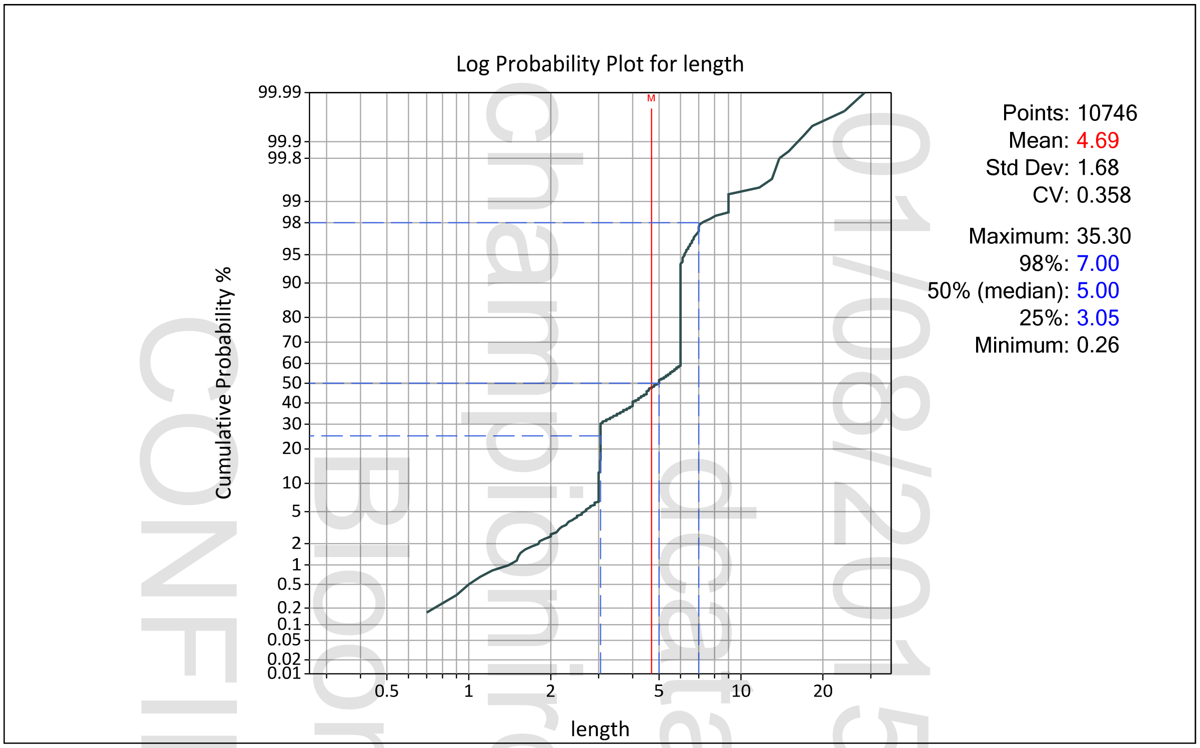


Figure 12.3.2: Lognormal Probability Plot of Sample Lengths

Table 12.3.1: Statistics of 7 m Composites, by Litho-Domain

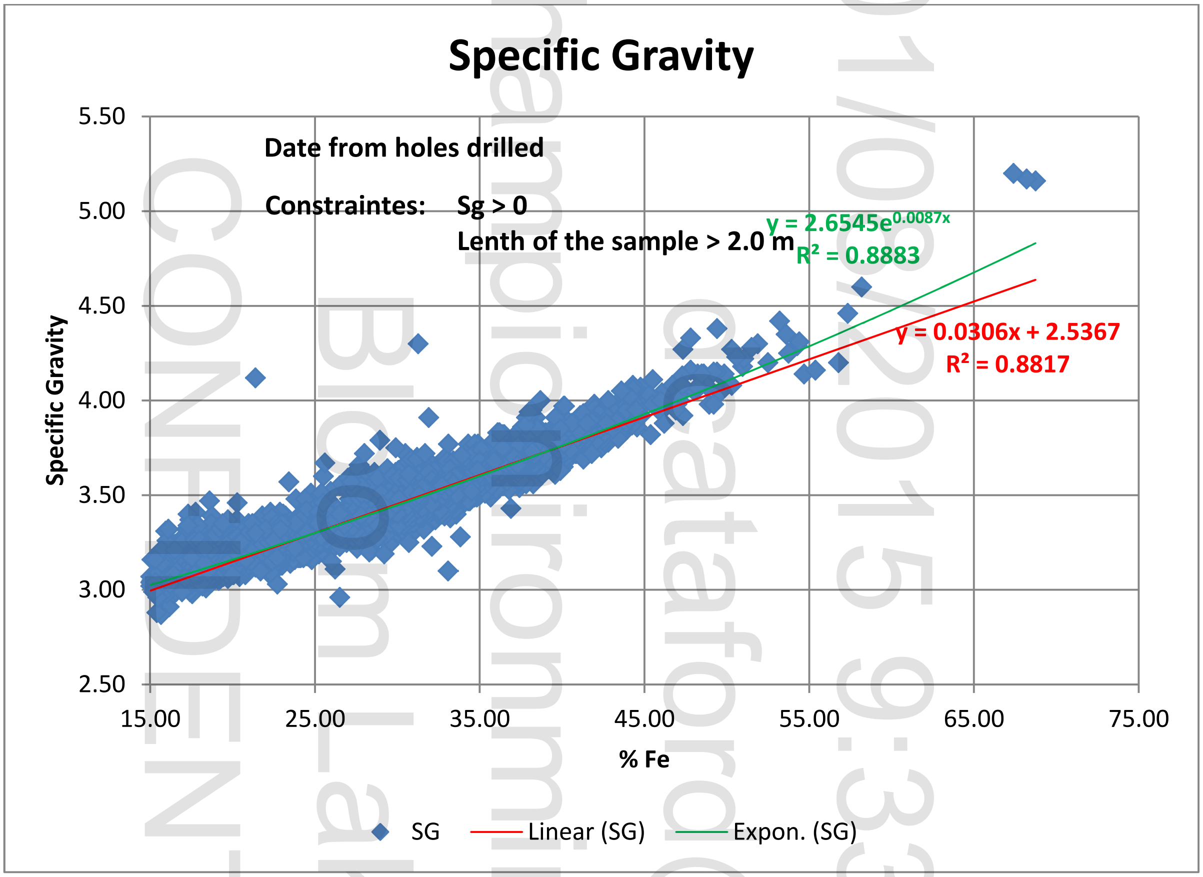
Metal	Domain	All	21	22	23	24	26	61	63	64	66
Fe	Samples	7241	3389	345	356	309	80	166	248	555	226
	Minimum	0.61	2.12	2.69	5.39	4.60	5.20	2.42	4.13	1.28	1.50
	Maximum	67.55	67.55	60.77	45.39	44.78	43.73	38.33	43.15	43.82	40.95
	Mean	26.14	29.59	28.24	30.68	27.66	29.75	20.60	26.70	23.73	26.07
	Median	28.58	31.10	29.37	31.92	29.38	32.56	21.02	28.58	25.04	28.06
	Std Dev	9.93	8.30	7.32	6.90	8.25	8.29	8.65	6.78	7.76	8.11
	CV	0.38	0.28	0.26	0.23	0.30	0.28	0.42	0.25	0.33	0.31
MagFe	Samples	7196	3373	345	350	306	75	166	248	545	224
	Minimum	0.03	0.04	0.07	0.07	0.21	1.22	0.03	0.10	0.10	0.61
	Maximum	43.63	43.63	24.60	38.46	43.45	38.73	35.15	35.47	41.67	41.43
	Mean	3.97	1.89	1.03	6.59	11.46	18.25	8.98	6.52	7.51	11.36
	Median	1.20	0.73	0.49	2.23	7.55	16.67	7.13	3.32	4.39	7.90
	Std Dev	6.64	4.02	2.12	9.05	10.64	11.13	8.22	7.29	7.64	10.21
	CV	1.67	2.13	2.06	1.37	0.93	0.61	0.92	1.12	1.02	0.90
SiO2	Samples	6421	3163	309	274	235	74	137	204	368	184
	Minimum	2.83	2.83	8.15	32.30	33.60	34.43	36.73	35.21	31.57	31.82
	Maximum	98.20	97.30	95.68	86.09	87.53	89.80	94.30	87.50	93.90	95.50
	Mean	56.81	55.65	58.11	52.28	54.03	53.36	52.03	50.87	51.61	53.50
	Median	54.38	54.49	57.39	50.19	51.56	50.63	48.01	48.79	48.94	50.93
	Std Dev	13.03	11.10	10.84	9.51	10.52	10.71	12.80	9.88	9.72	10.90
	CV	0.23	0.20	0.19	0.18	0.20	0.20	0.25	0.19	0.19	0.20
Al2O3	Samples	6416	3159	309	274	235	74	137	204	368	184
	Minimum	0.00	0.00	0.02	0.01	0.02	0.05	0.01	0.01	0.02	0.02
	Maximum	21.40	16.40	12.97	11.21	11.54	11.57	5.82	7.14	21.40	13.29
	Mean	1.37	0.76	0.64	0.36	0.36	0.81	0.31	0.58	0.99	0.85
	Median	0.19	0.17	0.17	0.18	0.14	0.21	0.14	0.18	0.20	0.18
	Std Dev	3.22	2.23	1.72	1.14	1.30	2.08	0.56	1.12	2.26	2.07
	CV	2.36	2.95	2.68	3.17	3.64	2.58	1.85	1.95	2.28	2.43
MgO	Samples	6560	3246	345	274	235	74	137	204	368	184
	Minimum	0.00	0.01	0.00	0.03	0.22	0.20	0.05	0.09	0.36	0.22
	Maximum	14.86	14.86	5.96	8.23	8.44	7.35	12.40	12.61	12.58	10.70
	Mean	1.42	0.43	0.29	1.67	2.33	2.03	6.73	3.97	4.13	2.89
	Median	0.11	0.06	0.06	1.26	2.10	1.67	7.23	3.52	3.73	2.71
	Std Dev	2.23	1.26	0.91	1.59	1.53	1.44	3.43	2.20	2.18	1.54
	CV	1.57	2.95	3.15	0.95	0.66	0.71	0.51	0.55	0.53	0.53
CaO	Samples	6559	3247	345	274	234	74	137	204	367	184
	Minimum	0.00	0.00	0.00	0.01	0.11	0.10	0.01	0.06	0.09	0.18
	Maximum	17.19	15.13	8.34	8.77	13.15	6.44	17.09	17.19	15.17	8.44
	Mean	1.49	0.35	0.36	1.50	2.52	1.81	7.67	4.50	4.93	2.78
	Median	0.08	0.02	0.04	0.79	2.26	1.26	6.99	4.09	4.58	2.54
	Std Dev	2.61	1.21	1.13	1.70	1.71	1.52	5.03	2.95	2.86	1.78
	CV	1.75	3.48	3.17	1.13	0.68	0.84	0.66	0.65	0.58	0.64

12.4 Specific Gravity

Cliffs analyzed results from 4,054 pyncometer tests conducted at Lakefield to derive an equation to assign specific gravity values to the block model. The data were plotted in a Microsoft Excel spreadsheet and a best fit trendline calculated (Figure 12.4.1).

In the 2012 estimation, the following calculation was used:

Sg_calc = (2.6655 * (exp (0.0086 * FeT)))



Cliffs 2013

Figure 12.4.1: Specific Gravity vs Total Fe %

12.5 Variogram Analysis and Modeling

In the 2011 estimation, Gemcom performed variogram analyses for Fe for each of the litho-domains. Limited analyses were done on the variograms of CaO, MgO and MagFe, and Gemcom found that the variograms were similar to the Fe variograms. The down-hole variograms were used to establish the nugget effect.

In the 2011 estimation, SRK reviewed the variograms in Vulcan® and adjusted the ranges for a better fit. The variograms for CaO and MgO did not have good structures. The variogram parameters for Fe are shown in Table 12.5.1 Cliffs used the same variograms for the 2012 estimation.

Table 12.5.1: Variogram Parameters, FeT

Parameter	Domain								
	21	**22**	**23**	**24**	**26**	**61**	**63**	**64**	**65,66**
Nugget	15	15	15	15	15	15	15	15	15
Orientation (°)									
Major Axis	-0.8,269	-12,104	-15,078	00,000	28,210	-0.8,269	-15,078	00,000	28,210
Semi-major Axis	-45,179	-39,004	-52,328	00,270	-17.5,130	-45,179	-52,328	00,270	-17.5,130
Minor Axis	45,178	48.5,028	33.6,358	90,000	28,210	45,178	33.6,358	90,000	28,210
Structure 1									
Sill differential	20	40	24	32	44	20	24	32	44
Range (m)									
Major	35	200	52	13	350	35	52	13	200
Semi-major	30	200	52	13	275	30	52	13	200
Minor	17	100	28	13	150	17	28	13	100
Structure 2									
Sill differential	21	None	10	25	None	21	10	25	None
Range (m)									
Major	200		200	200		200	200	200	
Semi-major	175		200	200		175	200	200	
Minor	100		100	200		100	100	200	

12.6 Block Model

The block model origin and block size are shown in Table 12.6.1.

The litho-domains were assigned to the block model by Cliffs based on the location of the centroid of the block within the wireframe.

Table 12.6.1: Block Model Parameters

Parameter	X	Y	Z
Start	612020	5854410	186
Length	4950	2160	644
Block Size	10	10	14
Number	495	216	46

The following variables were estimated by Cliffs:

- Fe;
- MagFe;
- Cao;
- MgO;
- SG; and
- Number of composites and drillholes used in the estimation of each block, distance to the closest composite and average distance to composites used in the estimation of each block were recorded in the block model.

12.7 Estimation Methodology

The variables for FeT and MagFe were estimated with ordinary kriging using the variograms shown in Table 12.5.1 and the parameters shown in Table 12.7.1. The search ellipsoid has the same orientation as the variograms. The blocks were estimated separately for each domain in order to use the correct search orientation, but composites in domains 21 to 26 could be used for any of the blocks coded 21 to 26; likewise composites coded 61 to 66 could be used in the estimation of blocks coded 61 to 66. Although the statistics indicate there are difference in Fe, MagFe, CaO and MgO by domain, these differences are gradational across the domain boundaries which are based on structural features.

Table 12.7.1: Estimation Parameters

Estimation	Distance (m)	Min/Max Samples	Max per DH
1	200,200,100	4,12	3
2	200,200,100	3,12	3
3	600,600,200	1,12	3

The requirement of a minimum of 4 composites in the first pass means that a minimum of 2 drillholes were used in the estimation.

The CaO and MgO variables were estimated with the Inverse Distance Squared (ID2) algorithm because there were fewer samples and because the variograms did not exhibit good structures. A nearest neighbor estimation was performed to determine the distance to the closest composite.

Specific gravity was also estimated with ID2 using the merged SG data as described in Section 12.4.

12.8 Model Validation

The block model was validated by three methods:

- Visual comparison of the block grades and composite grades by cross-section and plan views;
- Comparison of assay, composite and block model statistics; and
- Swath plots.

The visual comparison of the block and composite grades was, in general, good. The drilling at the east side is sparse and the comparison is not as good in that area. There is no drilling in domain 25 so the blocks were estimated with drillholes outside that domain. Figure 12.8.1 shows the FeT block grades at elevation 625.

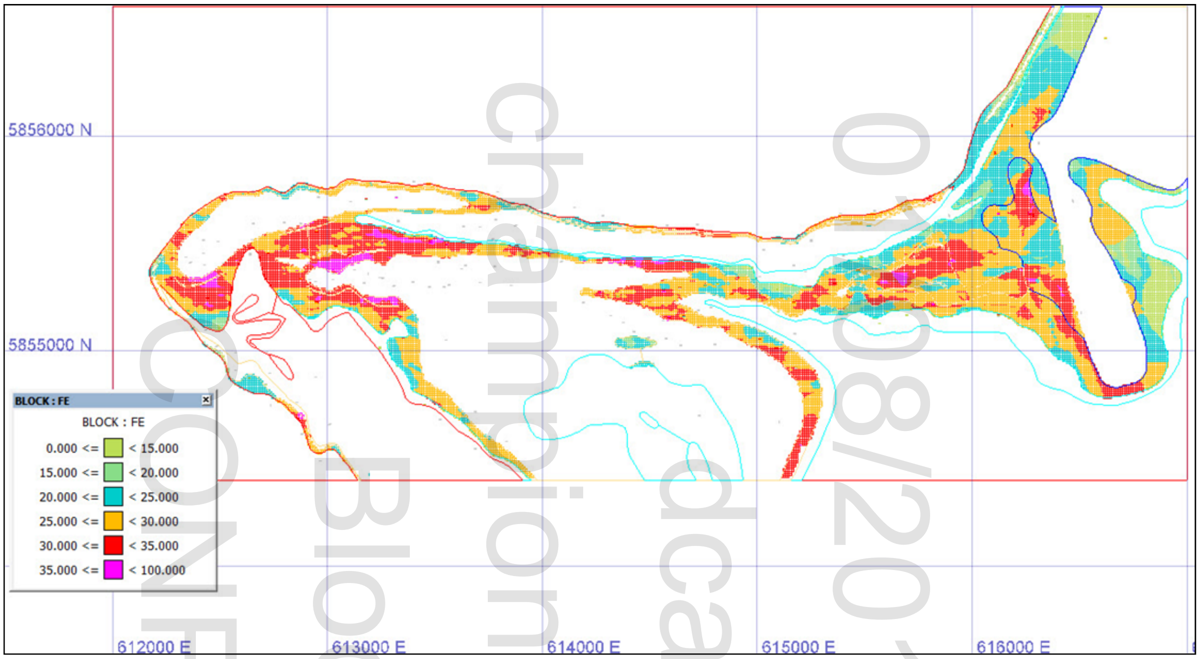


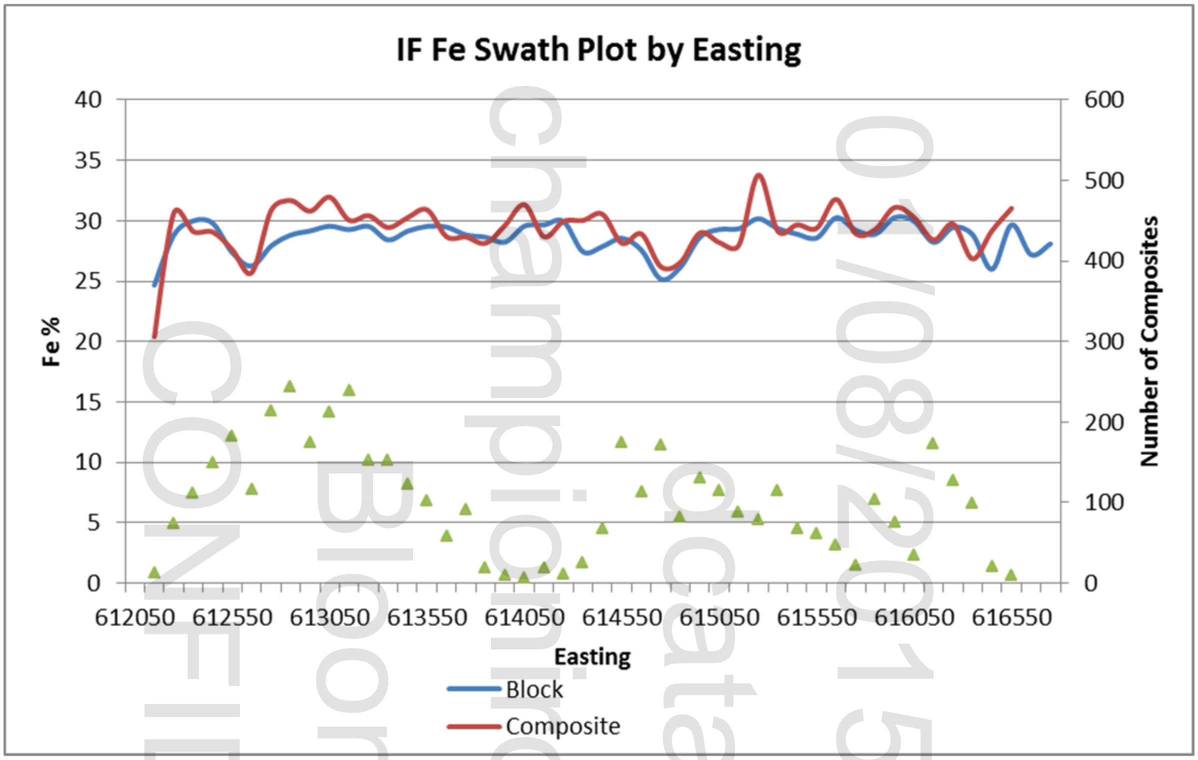
Figure 12.8.1: Elevation 625 FeT Block Grades

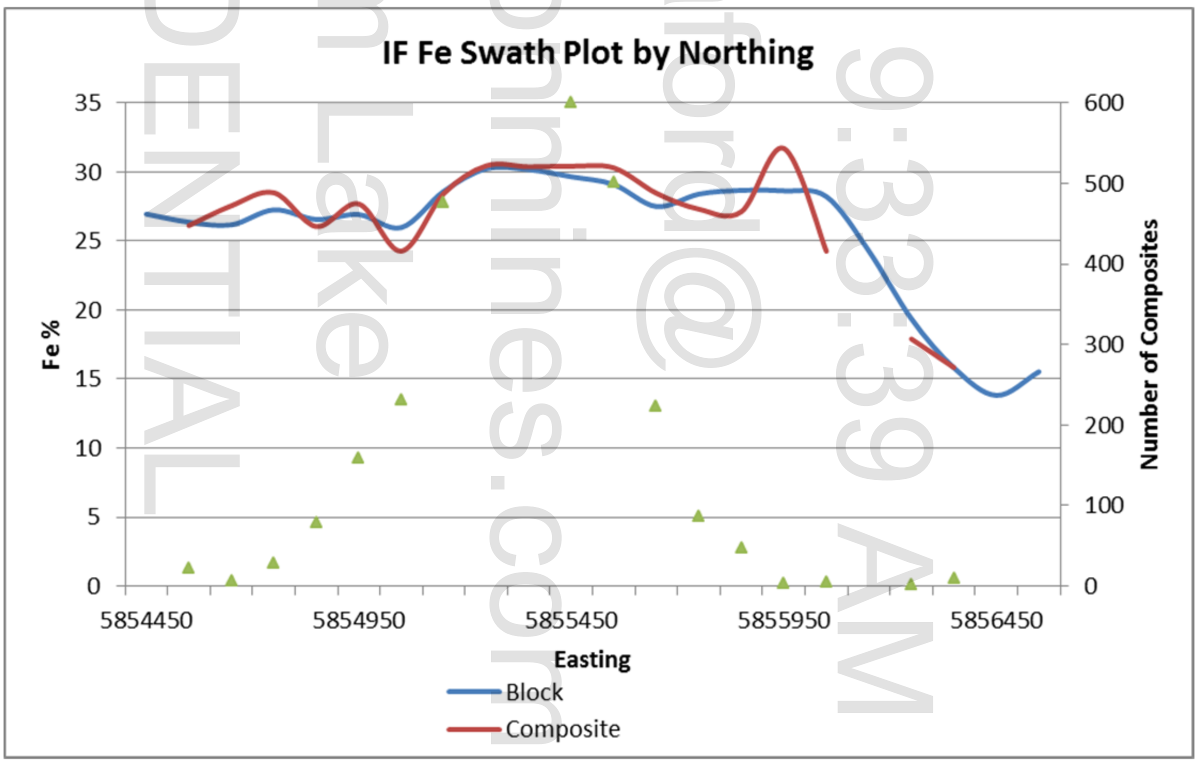
The block statistics are given in Table 12.8.1. The block statistics compare well to the composite statistics, except for domain 25 where there are only 7 composites and domain 65 where there are only 28 composites and the blocks were estimated primarily with composites from other domains. It is SRK's opinion that this acceptable as the domains are based on structural setting and not on FeT or MagFe grades.

Table 12.8.1: Block Model Statistics

Metal	Domain	All	21	22	23	24	25	26	61	63	64	65	66
Fe	Samples	443587	186523	35500	15669	17405	2111	4209	9952	18823	104731	27782	20882
	Minimum	2.87	6.31	2.87	10.82	7.70	16.57	18.18	5.43	10.47	3.41	7.49	13.38
	Maximum	53.59	53.59	39.08	39.59	38.48	36.75	39.72	32.25	38.78	35.93	36.51	36.42
	Mean	26.12	28.99	26.89	29.67	28.32	25.53	30.68	21.35	27.04	20.76	22.01	27.65
	Median	26.94	29.36	27.60	29.85	28.71	26.60	31.80	21.77	27.56	21.00	22.49	28.02
	Std Dev	5.76	4.43	4.87	3.57	4.56	4.35	3.71	3.73	2.91	4.70	5.93	3.46
	CV	0.22	0.15	0.18	0.12	0.16	0.17	0.12	0.18	0.11	0.23	0.27	0.13
MagFe	Samples	443587	186523	35500	15669	17405	2111	4209	9952	18823	104731	27782	20882
	Minimum	0.12	0.12	0.15	0.64	0.87	3.81	3.28	1.04	0.58	0.25	2.34	0.62
	Maximum	35.88	31.42	12.62	33.51	35.07	34.56	32.56	27.20	26.81	34.48	35.88	33.90
	Mean	6.40	2.18	0.95	7.89	13.00	15.52	18.94	8.66	5.89	10.06	15.59	12.22
	Median	4.05	1.17	0.52	5.57	12.49	16.39	18.78	7.68	5.47	9.30	16.05	11.17
	Std Dev	6.37	2.77	1.10	6.53	5.83	5.23	6.21	4.38	2.93	4.99	4.76	6.62
	CV	0.99	1.28	1.16	0.83	0.45	0.34	0.33	0.51	0.50	0.50	0.31	0.54
Cao	Samples	443587	186523	35500	15669	17405	2111	4209	9952	18823	104731	27782	20882
	Minimum	0.00	0.01	0.00	0.01	0.00	0.50	0.29	0.01	0.48	0.00	0.00	0.36
	Maximum	14.99	13.84	7.41	6.71	10.17	5.03	6.16	13.58	14.99	14.49	6.21	6.87
	Mean	2.14	0.50	0.58	1.71	2.23	2.41	1.60	2.29	4.00	5.04	2.73	2.82
	Median	1.55	0.12	0.09	1.76	2.17	2.38	1.50	2.00	3.69	4.64	2.77	2.81
	Std Dev	2.24	0.82	1.27	1.02	1.05	0.79	0.66	1.64	1.42	1.85	0.74	0.86
	CV	1.05	1.66	2.20	0.59	0.47	0.33	0.41	0.72	0.36	0.37	0.27	0.31
MgO	Samples	443587	186523	35500	15669	17405	2111	4209	9952	18823	104731	27782	20882
	Minimum	0.00	0.01	0.00	0.05	0.00	0.65	0.54	0.07	0.79	0.00	0.00	0.49
	Maximum	14.35	14.35	5.12	6.51	6.43	4.97	7.04	11.48	10.71	12.07	6.83	9.25
	Mean	2.02	0.56	0.44	1.90	2.18	2.46	1.94	6.65	3.55	4.03	2.77	3.03
	Median	1.81	0.17	0.07	2.02	2.14	2.28	1.85	6.84	3.36	3.86	2.85	2.95
	Std Dev	1.89	0.82	0.89	0.92	0.78	0.97	0.63	1.92	1.04	0.98	0.77	0.82
	CV	0.93	1.48	2.01	0.48	0.36	0.40	0.33	0.29	0.29	0.24	0.28	0.27

Swath plots comparing block grades to composite grades were constructed on 100 m bands by Easting and Northing and 21 m bands by elevation. Figure 12.8.2 shows the swath plots for Fe in the Iron Formation, litho-domains 21 through 26 and Figure 12.8.3 shows the swath plots for Silicate Iron Formation, litho-domains 61 through 66. There are only 29 Iron Formation composites below elevation 283 and only 10 composites east of 616550. In the Silicate Iron Formation there are no composites deeper than elevation 410 and only 21 east of 616550. The composite and block grades compare well on the swath plots.


IF Fe Swath Plot by Easting
Fe %
Number of Composites
40
35
30
25
20
15
10
5
0
600
500
400
300
200
100
0
612050
612550
613050
613550
614050
614550
615050
615550
616050
616550
Easting
Block
Composite


IF Fe Swath Plot by Northing
Fe %
Number of Composites
35
30
25
20
15
10
5
0
600
500
400
300
200
100
0
5854450
5854950
5855450
5855950
5856450
Easting
Block
Composite

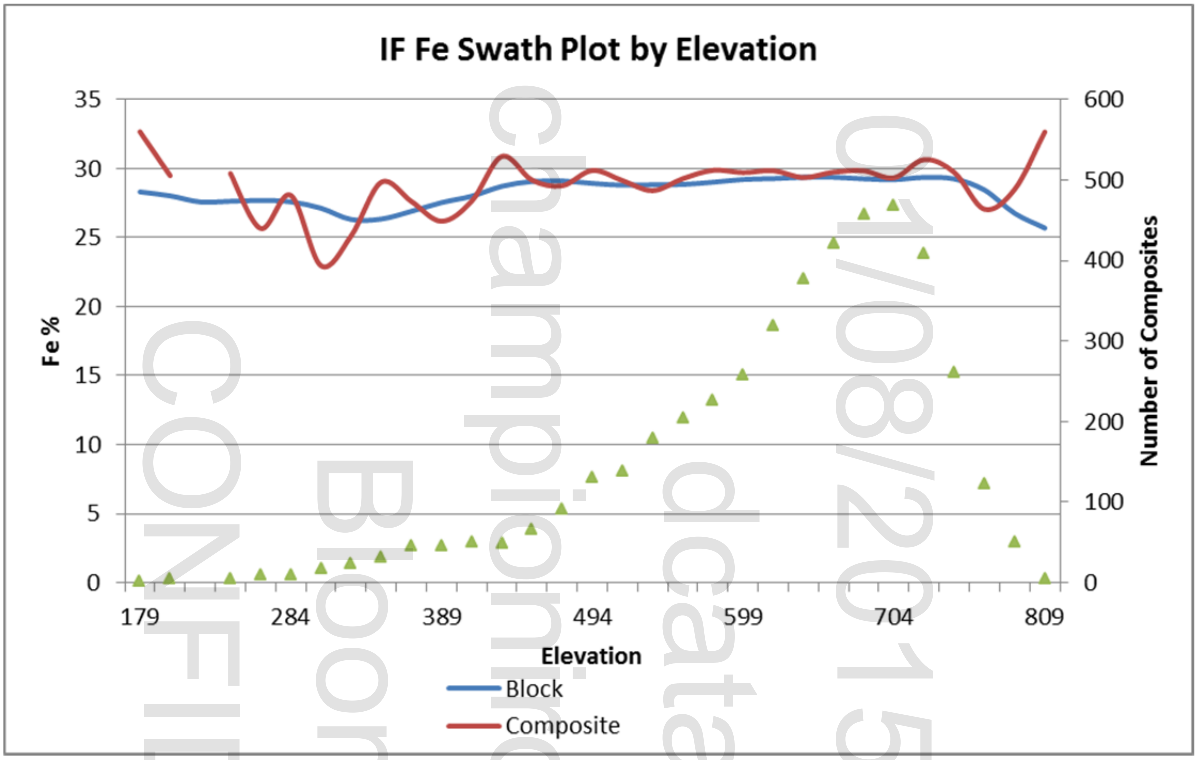
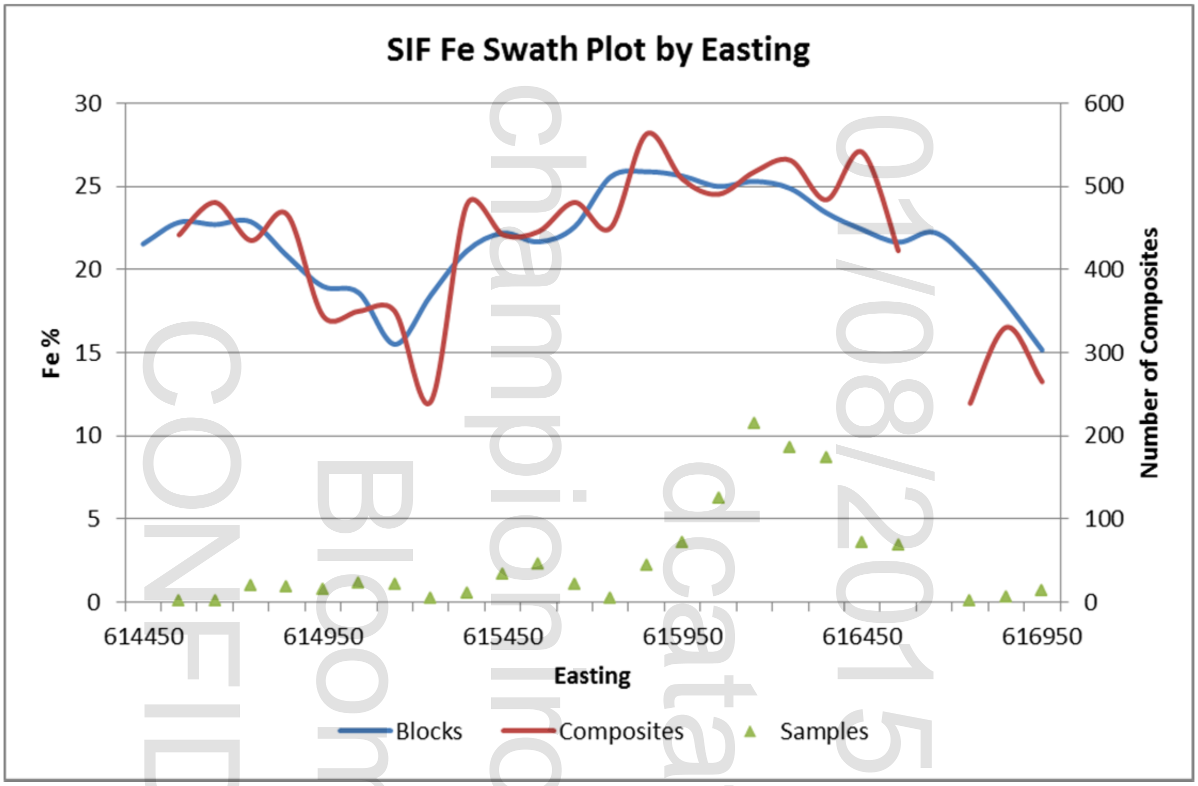
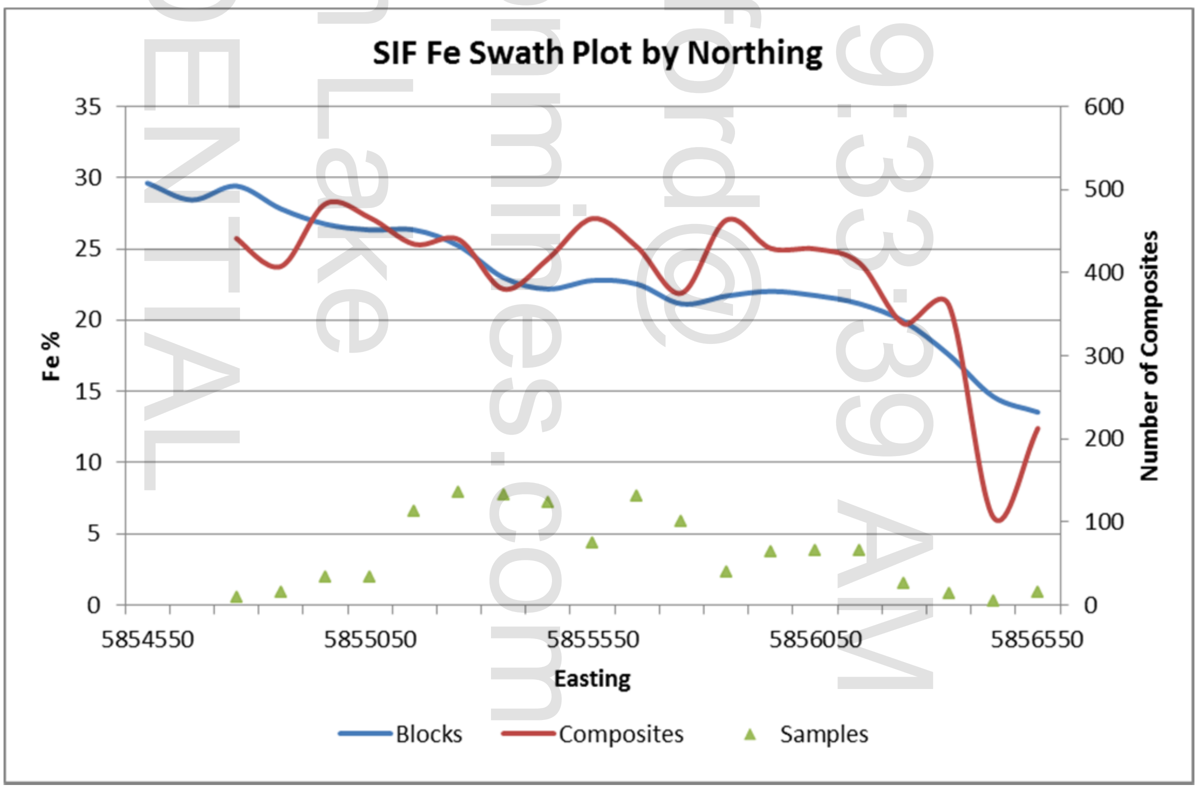


Figure 12.8.2: Iron Formation FeT Swath Plots


SIF Fe Swath Plot by Easting
Fe %
Number of Composites
Easting
0
5
10
15
20
25
30
600
500
400
300
200
100
614450
614950
615450
615950
616450
616950
Blocks
Composites
Samples


SIF Fe Swath Plot by Northing
Fe %
Number of Composites
Easting
0
5
10
15
20
25
30
35
600
500
400
300
200
100
5854550
5855050
5855550
5856050
5856550
Blocks
Composites
Samples

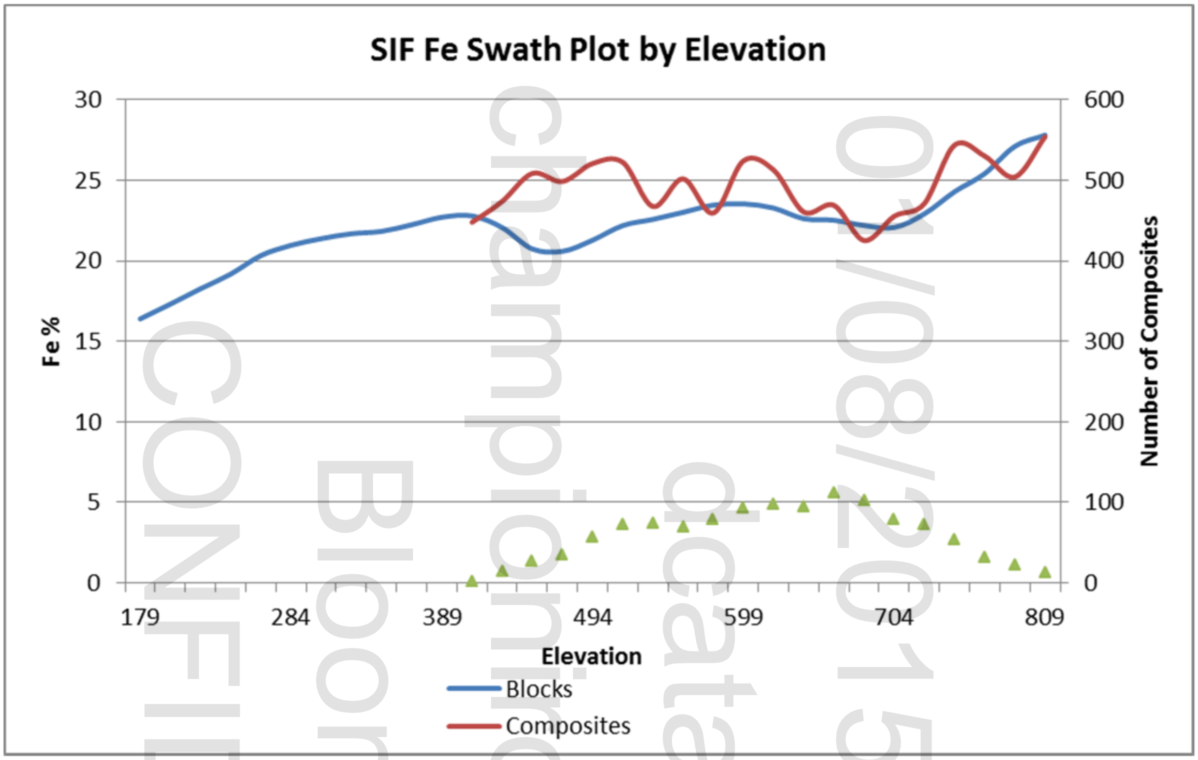


Figure 12.8.3: Silicate Iron Formation FeT Swath Plots

12.9 Resource Classification

The blocks were classified based on the pass in which they were estimated, the number of composites and drillholes and the distance to the composites used in the estimation. The classification method is shown in Table 12.9.1.

Table 12.9.1: Resource Classification

Classification	Pass	Minimum Composites	Minimum Drillholes	Distance
Measured	1	4	3	Average distance of 80 m to composites used in estimation
Indicated	1	4	2	200m to closest composite
Inferred	2	3	1	200m to closest composite
Inferred	3	1	1	300m to closest composite

12.10 Mineral Resource Statement

A pit optimization was run with the following parameters to constrain the resources as potentially mineable:

- Mass recovery (MR), where
    - o Concentrate Mass = [Mass Crude Ore*[Crude Fe(t) * Fe Recovery]]/Concentrate Fe(t), and
    - o Based on 79% Fe Recovery and a 66.2% Fe Concentrate.

- Average Product Value = US$120.00/t of concentrate;
- Bulk Mining Cost/t US$2.70/t;
- Surface Mining Cost/t AMT (All undisturbed <20 m upper surface) = US$4.10/amt;
- In situ Mining Recovery = 100%;
- Tailings and water management = US$1.00/t RoM;
- Processing Cost = US$6.62/t RoM;
- G&A Costs = US$4.50/t of concentrate;
- Logistics: Port and Transportation = US$25.00/t Concentrate; and
- Pit Slope Angle = 40.0^{o} for all material.

The Mineral Resources of the Bloom Lake deposit on a dry tonnage basis are presented in Table 12.10.1. The resources are stated at a cut-off grade of 15% for material inside the resource pit shell.

The classification meets the standards for resource classification set by CIM.

Table 12.10.1: Bloom Lake Mineral Resources, January 1, 2013*

Class	Material	Mt	FeT %	MagFe %	CaO %	MgO %
Measured	IF	393.2	30.1	3.2	0.47	0.58
	SIF	52.9	24.5	9.0	4.05	4.70
	Total	**446.1**	**29.5**	**3.9**	**0.90**	**1.07**
Indicated	IF	538.1	28.5	4.1	0.89	0.92
	SIF	381.7	25.1	8.8	3.96	3.79
	Total	**919.8**	**27.0**	**6.1**	**2.17**	**2.11**
Measured and Indicated		1365.8	27.8	5.38	1.76	1.77
Inferred	IF	143.2	29.8	2.5	0.50	0.59
	SIF	275.7	22.1	14.1	3.92	3.50
	Total	**419.0**	**24.8**	**10.1**	**2.75**	**2.50**

Effective Date: Resource Estimation October 31, 2012, Topography at July 1, 2012;
Tonnages and grade are rounded to reflect approximation; and
Resources are stated at a cut-off of 15% FeT and are contained within a pit optimization shell.

Mineral resources that are not mineral reserves do not have demonstrated economic viability. Mineral resource estimates do not account for mineability, selectivity, mining loss and dilution. These mineral resource estimates include inferred mineral resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. There is also no certainty that these inferred mineral resources will be converted to Measured and Indicated categories through further drilling, or into mineral reserves, once economic considerations are applied.

12.11 Mineral Resource Sensitivity

Grade tonnage curves were plotted separately for Total Measured and Indicated Resources and Inferred Resources and are presented Tables 12.11.1 and 12.11.2 and Figure 12.11.1. Tonnages are within the resource pit shell.

Table 12.11.1: Measured and Indicated Grade and Tonnage by FeT Cut-off Grade

Cut-off FeT %	Tonnage	FeT %	MagFe %	CaO %	MgO %
10	1,368.1	27.73	5.37	1.78	1.79
11	1,366.7	27.75	5.37	1.77	1.79
12	1,365.4	27.76	5.37	1.77	1.79
13	1,363.3	27.79	5.37	1.77	1.79
14	1,360.7	27.81	5.37	1.76	1.78
15	**1,356.3**	**27.86**	**5.37**	**1.76**	**1.78**
16	1,349.5	27.92	5.36	1.75	1.76
17	1,339.0	28.01	5.34	1.73	1.75
18	1,324.5	28.12	5.32	1.70	1.72
19	1,304.5	28.27	5.28	1.66	1.69
20	1,279.8	28.44	5.23	1.62	1.65

Table 12.11.2: Inferred Grade and Tonnage by FeT Cut-off Grade

Cut-off FeT %	Tonnage	FeT %	MagFe %	CaO %	MgO %
10	409.8	24.2	10.46	2.81	2.58
11	408.9	24.3	10.46	2.81	2.58
12	406.7	24.3	10.47	2.8	2.58
13	403.2	24.4	10.48	2.8	2.57
14	395.1	24.6	10.51	2.79	2.56
15	**389.4**	**24.8**	**10.51**	**2.78**	**2.55**
16	372.8	25.2	10.5	2.71	2.5
17	361.5	25.5	10.5	2.64	2.47
18	349.4	25.8	10.5	2.56	2.43
19	332.2	26.1	10.39	2.48	2.38
20	308.5	26.64	10.22	2.37	2.3

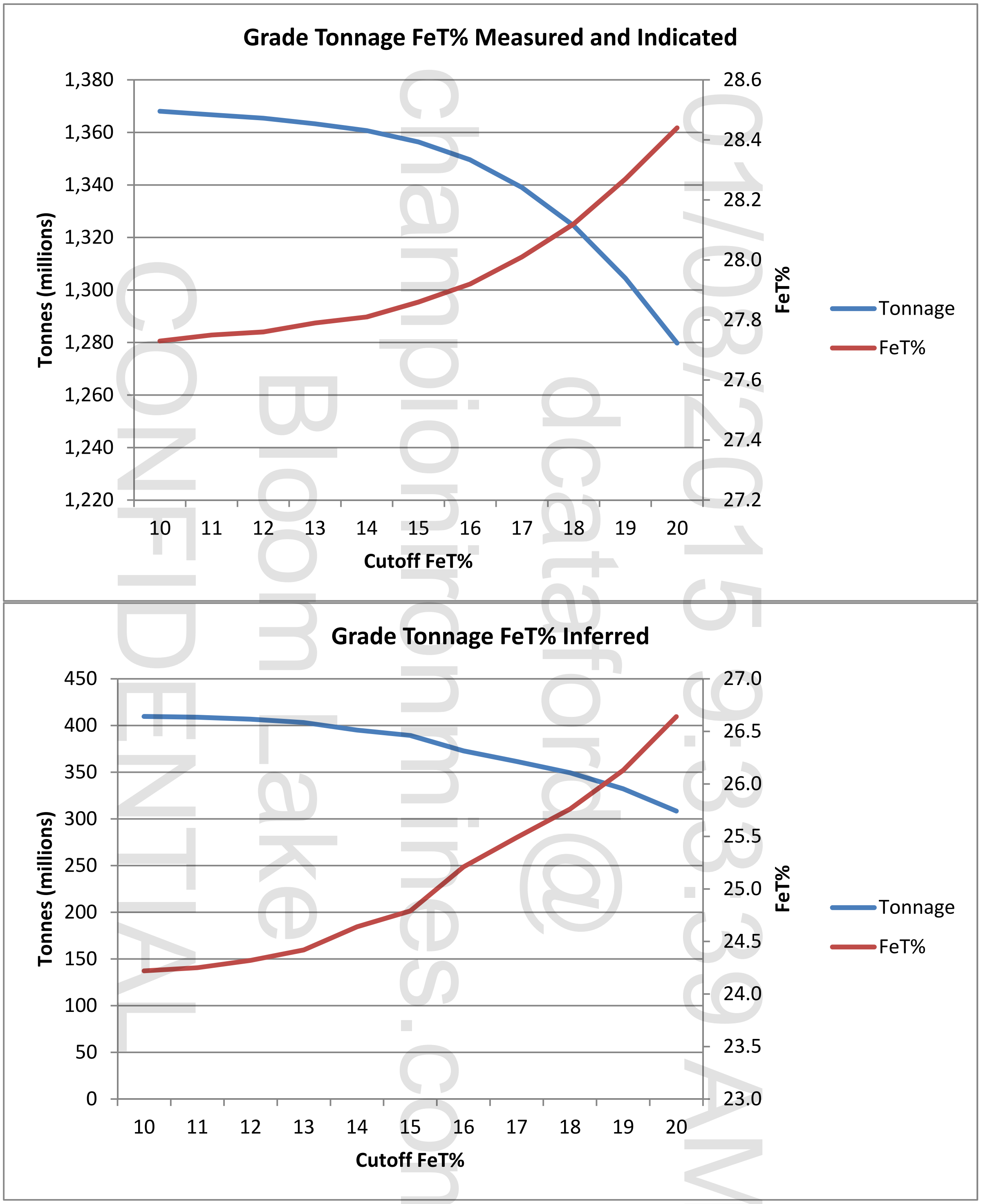


Figure 12.11.1: Grade Tonnage Curves for FeT %

13 Mineral Reserve Estimate

Using a cost structure based on 2012 actual production and the 2012 to 2016 budget, a new economic analysis was completed resulting in a new ore reserve statement for Bloom Lake. The proven and probable ore reserves for Bloom Lake based on this economic analysis are presented in Table 13.1.

Table 13.1: Bloom Lake Reserves as of January 1, 2013*

Classification	Tonnes	FeT	CaO	MgO	Mag Fe	Magnetite	Concentrate
	Mt (dry)	%	%	%	%	%	Mt (dry)
Proven	29.9	0.9	1.0	3.8	5.3	141.9	29.9
Probable	28.0	2.2	2.2	6.6	9.1	201.8	28.0
Proven + Probable	28.7	1.7	1.7	5.5	7.6	343.7	28.7

- Reserves effective date: July 1, 2012;
- All RoM and product tonnages reported are in dry metric tonne units.
- Reserves are reported not diluted;
- Reserves are reported within Mineral Lease Boundary;
- Average global strip ratio for the reserve was calculated to be 1.48 (waste to ore)
- Reserves based on July 1, 2012 topography;
- Cutoff used for Fet: 20%
- Concentrate Product is 66.2% Fe;
- Crude to Concentrate Mass Recovery Equation For all years based on the following equation: Concentrate Mass = [Mass Crude Ore*[Crude Fe(t) * Fe Recovery]]/Concentrate Fe(t). (Based on 79% Fe Recovery and a 66.2% Fe Concentrate). Phase I Fe recovery is calculated to be 72.2%. Phase II Fe Recovery is calculated to be 79%. Phase I and II Combined Fe recovery is calculated to be 75.3%.
- Reserves use US$112/t of concentrate selling price.
- Reserve material mined from July 1, 2012 through December 31st, 2012 is 6.7Mt of Crushed ore, 2.18Mt of concentrate, Crushed Fe grade of 32.4%. Waste material mined is 7.35Mt.

The 3-year benchmark trailing-average price adjusted for Bloom Lake site as per Cliff's corporate policy was used as the basis to determine proven and probable reserves. From this evaluation, a series of Whittle™ shells were generated based on revenue factors of the current 3-yr trailing price of US$112/t, f.o.b. port of Sept-Îles The selected shell was then used to guide a full engineered pit design, which was then phased appropriately and scheduled for mining activities for the duration of the project.. The costs used in this study represent all mining, processing, transportation and administrative costs, including loading of the sinter concentrate into lake freighters at Sept-Îles Port, Quebec.

The updated reserves of 1,051.3 Mt of proven and probable ore are similar to those reported in 2011. The new reserve is based on the updated 2012 Resource which was audited by SRK. SRK audited an upside case where Phase I and II mill throughput were increased from 2,200 t/h to 2,400 t/h. This upside mill throughput mine plan yielded a higher NPV however the reserves remain the same.

Detailed parameters used for the pit optimization are:

- Mass recovery (MR) = Concentrate Mass = [Mass Crude Ore*[Crude Fe(t) * Fe Recovery]]/Concentrate Fe(t). (Based on 75.3% Fe Recovery and a 66.2% Fe Concentrate);
- Average Product Value = US$112.00/t of concentrate;
- Bulk Mining Cost/t Ore = US$2.70/t; Waste = US$2.70/t;
- Surface Mining Cost/t AMT (All undisturbed <20 m upper surface) = US$4.10/amt;
- In situ Mining Recovery = 95% (by Mass) of Crude Ore;

- Tailings and water management = US$1.00/t RoM;
- Processing (Plant) Cost = US$6.62/t RoM;
- Admin/SG&A Costs = US$4.50/t of concentrate;
- Logistics: Port and Transportation = US$25.00/t Concentrate; and
- Pit Slope Angle = 40.0º for all material (including ramps).

13.1 Conversion Assumptions, Parameters and Methods

Pit optimization and pit design were conducted to convert the Mineral Resources to Mineral Reserves. The reserve estimation was conducted by Cliffs and audited by SRK. In accordance with NI 43-101 guidelines, the pit optimization has used only blocks classified as Measured or Indicated to generate revenue. Inferred resources are treated as waste. Blocks with Fe content less than 20% were not used in the Whittle™ pit optimization program to define the pit limits. To keep a steady iron feed grade to the plant, it was decided to use 20% Fe instead of 15% Fe used in the resource estimation. After the pit optimization was run, a detailed pit design with ramps was developed. The pit optimization and pit design were conducted at the end of December 2012.

The new July 1, 2012 ore reserve estimate has incremental differences from the preceding 2011 estimate, including similar sinter feed reserves and mine life, with an increased waste stripping as a function of a higher strip ratio. The updated 2012 resource model, which was used as the basis for the 2012 optimization and reserve calculation, had a large effect on the breadth and geometry of economic ore as compared to the 2011 Resource Model. The change is manifested primarily in the interpretation of the western Bloom area, where there was a considerable extension of a southwestern lens (at depth), though this was also accompanied by thinning of the previous interpretation in the central west pit area. The net effect of these changes has summarized the 2012 economic reserve position similar to 2011.

As mentioned above, the new 2012 resource model, along slight changes to core financial assumptions and design parameters, have contributed to the increased waste stripping from 1.33 to 1.48. These factors are:

- **Concentrate Price** –The 2011 ore reserve estimate used a pit shell reflecting a price of US$95/t of concentrate. The 2012 ore reserve estimate used a pit shell reflecting a price of US$112/t of concentrate which is the 2012 three-year trailing average price.
- **Operating Cost** –The 2011 ore reserve estimate used an overall operating cost of US$62/t. The operating costs in the 2012 estimate were larger by comparison, with operating costs on the order of US$73/t of concentrate - this more adequately represents the current as well as forecast unit cost profiles.
- **Revenue Margin** –The 2011 reserve operating profit margin was about US$65/t of concentrate. The 2012 profit margin, as a function of both increased price and increased cost assumptions, is US$38/t of concentrate.
- **Design Changes** – In order to maintain proper access through all mining push backs to multiple ore and waste delivery locations, all interim pit phases contain one or more high-wall ramp systems. Whilst phase ramp systems are temporary, the final high-wall ramp systems are not mined out. All pit walls and ramps (both phase and final) are engineered to the same design parameters. Inter-ramp angles were changed from 52º to 46º. The change is due to limited knowledge of slope stability. A slope stability study is planned for 2013.

13.2 Pit Optimization

As part of the resource evaluation, multiple Whittle™ pit optimizations were carried out on the full breadth of the Bloom Lake resource. As a part of the reserve derivation, the mathematically optimal pit optimization results have been used as a guide for detailed engineering pit and waste dump designs. Inputs used for the optimization do not necessarily conform with those quoted in the final economic cash flow model. Mathematical pit optimization economic parameters are shown in Figure 13.2.1.

			$/amt	$/rom t	$/conc t
COSTS	MCAF	Mining Cost ($/All Material Tonne)	2.70	7.05	21.43
	PCAF (1)	Processing Cost ($/Crude Ore Tonne Crushed)		6.62	19.41
		Tailings & Water Management Cost ($/Crude Ore Tonne Crushed)		1.00	2.93
	PCAF (2)	Logistics Cost ($/Product Tonne)			25.00
		Admin/SG&A Costs ($/Product Tonne)			4.50
		Total Cost/Tonne Concentrate			73.28

Source: Cliffs, 2012

Figure 13.2.1: Mathematical Pit Optimization Economic Parameters

13.2.1 Whittle™ Results and Analysis

As a result of the pit optimization, the relationship of potential pit shells (derived based on block scale economic analysis) is known and allows for subsequent shell selection by function of strip ratio comparison as subject to revenue factoring based on US$112/t concentrate product. This evaluation is conducted by analyzing the relationship of ore to waste, the associated best and worst case scheduled scenario cash flow for each incremental pit shell. With this information, the accommodating risk profile and revenue generating potential for the deposit can be estimated. Table 13.2.1.1 shows the nested pit results ranging from US$56/t concentrate to US$134/t concentrate. Table 13.2.1.2 shows the chosen pit for the 2012 Reserve (Pit 9), based on approximately US$100/t which was used to guide subsequent pit design and scheduling purposes. Pit 9 represents a mathematical shell based on a revenue factor of 0.90, and therefore complies with policy on the basis of being less than the current three year trailing average shell at revenue factor 1.00. The Pit 9 shell represents an equitable reserve scenario without the large inflexion of total waste burden which come with larger revenue factor shells above revenue factor 0.90, which allows the operation to subvert potential risk involved with additional capital costs.

Table 13.2.1.1: Whittle™ - Mathematical Pit Optimization Results – All Revenue Factors

Pit	Revenue Factor	US$	In situ Tonnes (MT dry)			Strip Ratio	FeT Grade	CaO Grade	MgO Grade	Mag Fe Grade
			Total	Ore	Waste		%	%	%	%
1	0.50	56.00	5	5	0	0.08	34.70	0.36	0.61	5.68
2	0.55	61.60	127	106	20	0.19	31.02	0.78	0.94	4.12
3	0.60	67.20	668	457	211	0.46	29.48	1.63	1.70	5.72
4	0.65	72.80	1,096	663	432	0.65	29.00	1.85	1.87	5.94
5	0.70	78.40	1,419	781	638	0.82	28.83	1.88	1.90	6.07
6	0.75	84.00	1,791	886	905	1.02	28.76	1.79	1.81	5.81
7	0.80	89.60	2,163	974	1,189	1.22	28.70	1.73	1.76	5.67
8	0.85	95.20	2,438	1,030	1,408	1.37	28.67	1.70	1.73	5.58
9	0.90	100.80	2,581	1,058	1,523	1.44	28.62	1.69	1.73	5.53
10	0.95	106.40	2,782	1,091	1,690	1.55	28.59	1.67	1.70	5.45
11	1.00	112.00	3,069	1,134	1,935	1.71	28.54	1.64	1.67	5.36
12	1.05	117.60	3,195	1,151	2,045	1.78	28.54	1.63	1.66	5.33
13	1.10	123.20	3,309	1,165	2,144	1.84	28.52	1.62	1.65	5.29
14	1.15	128.80	3,379	1,173	2,206	1.88	28.51	1.61	1.65	5.27
15	1.20	134.40	3,433	1,179	2,254	1.91	28.50	1.61	1.65	5.26

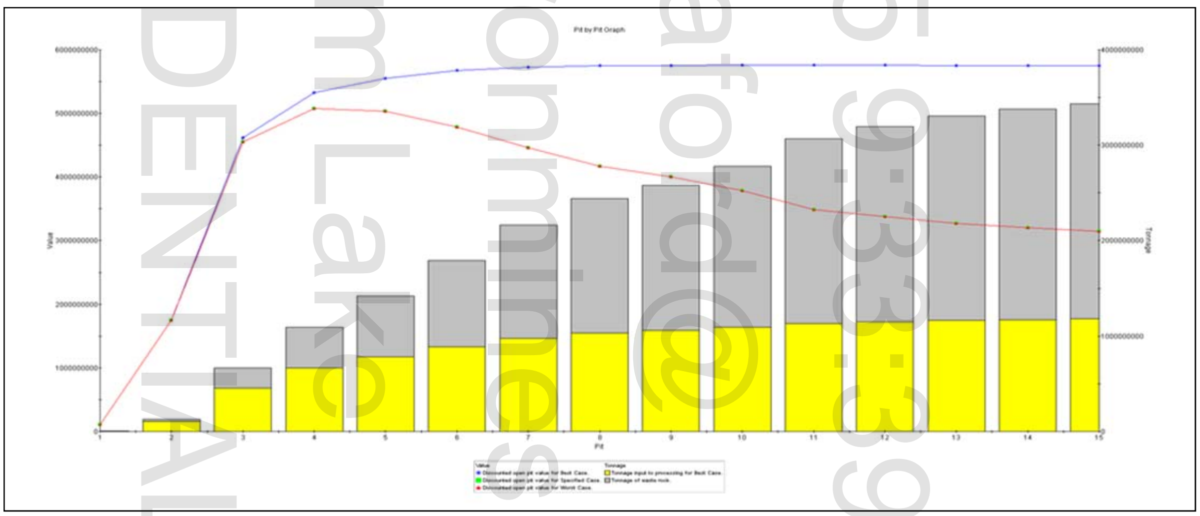


Figure 13.2.1.1: Whittle™ - Mathematical Pit Optimization Results – Pit by Pit Graph

Table 13.2.1.2: Whittle™ - Mathematical Pit Optimization Result – Optimal (Pit 9)

In situ Tonnes (MT dry)			Strip Ratio	RoM Grade			
Total	Ore	Waste		FeT (%)	Cao (%)	MgO (%)	MagFe (%)
2,581	1,058	1,523	1.44	28.62	1.69	1.73	5.53

13.3 Relevant Factors

The pit optimization shows that over 77% of the Economic Measured and Indicated Resources fall inside the optimal shell (Pit 9). The deposit remains open at depth in the western Bloom Lake pit (below the optimal shell) – this material may become financially viable in the future with improved economics which would support an increased strip ratio beyond the 2012 optimal pit shell.

The pit optimization has not been altered for mining width restriction at the very bottom of the shell, therefore extractable ore tonnes as per the engineered pit design which adhere to this restriction in practice will be slightly less and/or the realized strip ratio will be slightly higher than the Whittle™ result. Additionally, with increased ramp installations, the overall slopes of the actual design in certain key walls may be shallower than the Whittle™ pit result which was driven by slope arcs based on a common overall slope angle of 40 degrees –this will also incrementally increase the strip ratio from the mathematical result. The subsequent reserve quantities, therefore, include these considerations (see Open Pit Design section).

14 Mining Methods

14.1 Current Mining Methods

The Bloom Lake deposit is mined by surface mining methods. Cliffs uses a combination of contract and owner operated mining but owns the majority of the mining fleet. The expansion of 6.8 Mt/y (dry) to 14 Mt/y (dry) concentrate production includes the upsizing of the current equipment fleet. The surface operations include:

- Topsoil Stripping (including Surface/Till Removal) to expose bedrock surface;
- Bench Development: Ripping/Dozing/Drill/Blast, Loading and Hauling;
- Production (Bench) Drill and Blast;
- Production (Bench) Loading and Hauling; and
- General Maintenance and services to maintain pit operations.

The reserve is based on ongoing annual starting production of (at a minimum) 6.8 Mt/y from a blend of ores from the main Pignac and Chiefs pit areas as well as the new West Bloom Lake area. In 2014 the operation will commence the expansion to 14 Mt/y (dry) production, with a ramp-up schedule in the early years. Mining and Processing operations are scheduled for 24 hours/day, and the mine production is scheduled to directly feed the processing operations, with few notable exceptions as detailed in the production schedule (Section 14.3).

14.1.1 Open Pit Design

The pit design combines current site access, minimum mining width requirements, trafficability, and generalized geotechnical/engineered slope parameters to explore the possible full extraction of the resources through open pit techniques in a practical manner. As such, restrictions were not placed on any areas other than the current infrastructure locations.

14.1.2 Pit Design Parameters and Construction

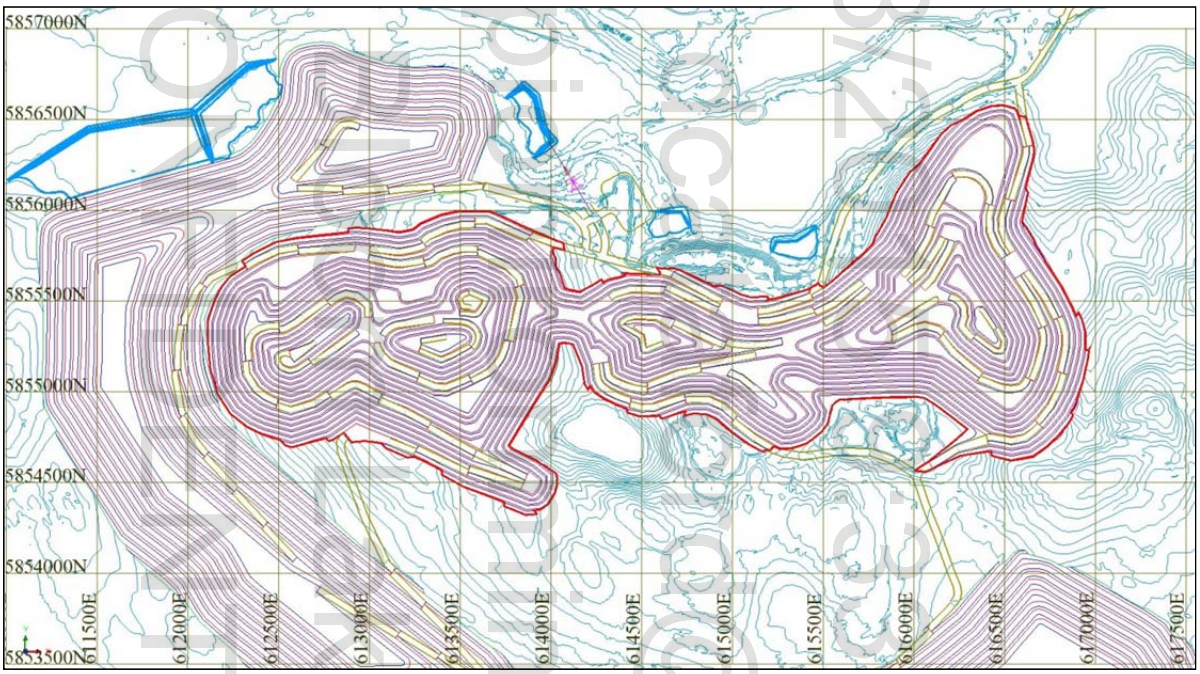
The pit has been engineered to maintain overall slopes at relatively conservative angles due to the limited local slope stability knowledge as a function of the early stage of the operation. The pit was designed in multiple nested phases, with the stage interactions and access being considered throughout the life of the phasing, and as it ties into the final design. All temporary walls (based on the pushback staging strategy) and final walls were designed to the same criteria. Additional space was allocated at various strategic points in the pit designs (such as at ramp switchbacks, etc.) to allow for future in-pit infrastructure (supporting power or dewatering facilities). Additionally, extra effort was made to reduce curvature and other inflexions in pit walls (although difficult to achieve due to the highly variable nature of the orebody geometry and the additional stripping required to straighten walls further.) See Figure 14.1.2.1 for a map of the final pit design.

The strategy for the pit design was to enhance the haulage and accessibility options of the pit throughout its development and stage interaction by the emplacement of multiple haulage ramps, which will facilitate access to both crushers and multiple dump locations simultaneously.

Table 14.1.2.1 indicates the design parameters used in the design generation.

Table 14.1.2.1: Pit Design Parameters

Parameter	Unit	Value	Note
Overall Slope Angle	Degrees	45	(wall with no ramps)
Overall Slope Angle	Degrees	40	(1 pass standard dual lane ramp)
Batter Angle	Degrees	70	
Bench Height	m	14	
Berm Width	m	20	
Wall Configuration	m/m	28/20	(double bench, single berm (repeated))
Ramp Width (std. dual)	m	35	
Ramp Width (std. single)	m	25	
Ramp Gradient (max)	%	9	



Source: Cliffs, 2012

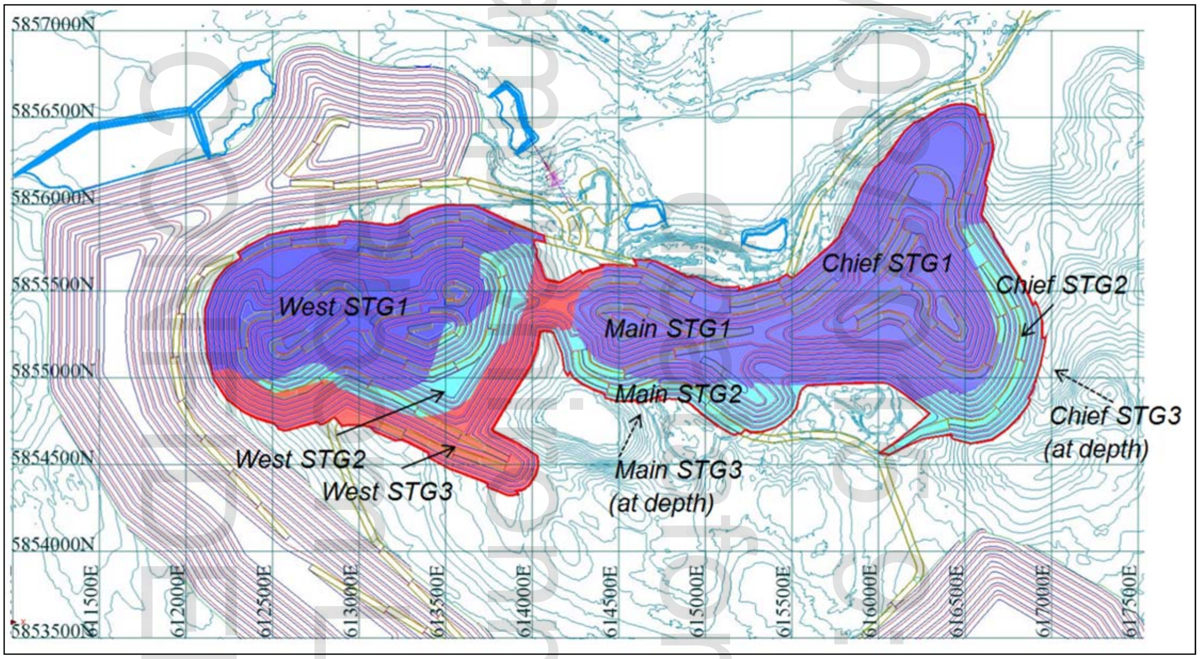
Figure 14.1.2.1: Final Pit Design – Plan View

14.1.3 Geotech

Bloom Lake has not performed any significant slope stability studies to validate the current pit design parameters; however they have plans for conducting a full study in 2013. Due to the lack of slope stability information, SRK decided to maintain an overall slope angle of 45º or less for all pit walls. As the majority of walls have permanent final ramps embedded into the final design, the majority of walls are 40º or less. SRK supports the plan for Cliffs to conduct a full slope stability study in 2013, and with this new information a re-design of the pit may be necessary to achieve an optimal and practical solution. Based on the geometry of the pit the change to the reserve quantum would not be material based on changes in slope angle. SRK is of the opinion that with proper detailed pit slope study, the inter-ramp angles may be increased.

14.2 Phase Design

Phase designs for the deposit are largely driven by effective mining widths and their influence on access to blendable proportions of the resource at any given time, whilst maintaining some ability to manage strip ratio over the life of the reserve. The same design parameters used in the final design are embedded in the stage designs to support the phasing strategy. Figure 14.2.1 indicates the relative positioning of the phase designs.



Source: Cliffs, 2012

Figure 14.2.1: Phase Pit Design Locations – Plan View

The Bloom Lake mining group has conducted initial blending studies in 2012, and the results of this information has identified blending regions West, Main, and Chief based on ore characteristics. These regions were used to guide to develop the phasing strategy, and each region was further subdivided into three stages, representing a total of nine phases used for the generation of production schedule inventory and ultimate reserve base.

14.3 Production Schedule

The production schedule is used as the basis of the economic model and comprises predicted material quantities and grades, as well as provided the required haulage profiles. A number of schedule iterations were run using the Surpac MineSched software program, manipulating various targeting strategies and production parameters.

In order to quantify and understand both the current 2012 baseline operational context (based on a 2,200 t/h mill throughput rate which is derived from historical operating capacities and 2012 enhancement assumptions), as well as the forecast operational context (based on a 2400 t/h throughput rate which is derived from a forward looking enhancement of the mills with respect to a step change in productivity from 2012 levels), two separate LoM mine schedules were generated based on the same reserve design. With this information, the economics of the various positions could be derived and the sensitivity to this particular assumption can be understood.

The two mine plans generated for this evaluation are the following:

<u>2,200 t/h Mill Throughput Mine Plan (2012 Baseline Case):</u> This mine plan assumed the stated reserves found in this document combined with the following assumptions:

- Phase I mill throughput: 2,200 t/h;
- Phase II mill throughput: 2,200 t/h;
- Phase II Mill startup: End of first quarter 2013 (since the creation of this mine plan, an announcement by Cliffs occurred where construction for Phase II were delayed. SRK did not have the chance to update the mine plan to show the new Phase II start date);
- Constant grade throughput;
- Maximum total tonnes movement of 100 Mt per year;
- Fe recovery for Phase I: 72.2%; and
- Fe recovery for Phase II: 79%.

<u>2,400 t/h Mill Throughput Mine Plan (Forecast Case):</u> This mine plan assumed the stated reserves found in this document combined with the following assumptions:

- Phase I mill throughput: 2,400 t/h;
- Phase II mill throughput: 2,400 t/h;
- Phase II Mill startup: End of first quarter 2013 (since the creation of this mine plan, an announcement by Cliffs occurred where construction for Phase II were delayed. SRK did not have the chance to update the mine plan to show the new Phase II start date);
- Constant grade throughput;
- Maximum total tonnes movement of 113 Mt per year;
- Fe recovery for Phase I: 72.2%; and
- Fe recovery for Phase II: 79%.

In both production schedules, the short term strategy was to closely match the current 5 year operational production plan physicals for the expansion and ramp-up period. This would ensure the LoM is aligned with the current development constraints which are more comprehensively modeled in the current short term mine plan. As the pit development progressed, and the ore feed rates reached target levels (based on the mill throughput rates), the production schedules diverge in order to meet the objectives of the long term strategy based on the respective mill throughput rate. In each case, however, certain realistic constraints remained constant, such as the maximum vertical advance rate of 7 benches per year, such that the practicalities of extraction are honored equally.

In both production schedules, current blending region access and proportions were maintained as long as possible. To counter ore supply variance year over year, a RoM stockpiling stream was included. There are some periods which over-produce ex-pit ore (and others which are deficient) so this acts as a buffer to that annual SR variability and allows total movement to remain relatively consistent (although the strip ratio varies). Therefore in some periods the ex-pit ore tonnes/grade and the crushed tonnes/grade differ and a balance is maintained in the stockpile. At the end of the LoM, however, all this material is blended into the crusher, and in most cases as soon as is possible as to avoid the penalties of depreciation.

Schedule 1 Results: 2200 t/h (2012 Baseline Operational Context)

The 2,200 t/h (Mill Throughput Rate) production schedule results indicate a steady crude ore feed rate for the first 20 years of approximately 38 MT (dry) / annum to the mills at approximately a 1.6 Strip Ratio (once the ramp-up period is complete) and a maximum all material movement of approximately 100 MT (dry) / annum. This is managed by the balancing of the in-pit staging (for development of all 9 phases of the reserve) for both strip ratio and grade. After these first periods, the production schedule fails to meet the total ore supply requirement for two years, even though ~2.6 MT (dry) stockpiled ore is blended in order to mitigate (this ore was stockpiled ahead of the known shortfall from excess in the 20 year stable period). Work is ongoing to further understand this ore supply issue (~2032), which is in part due to the geometry and grade/SR variability of the ore body at this achieved depth, and in part due to the staging interaction of the current phasing strategy (this event takes place after the completion of the Stage 3 Main phase). Further scheduling will be conducted by Cliffs in 2014 which will attempt to rectify this with further drilling information, optimization, stage re-design and schedule algorithm manipulation, with additional efforts to further smooth production physicals, grade variability, and reduction in stockpiling).

Mined Total Fe grade (FeT) for the first 20 years remains relatively stable at an average grade of 28.9% (ranging between 28.5% and 29.6%). CaO and MgO are consistent at approximately 1.3 -1.4 for the first ten years, and then a step change takes place (~2021) and the grades increase to approximately 1.7-1.8. Mag Fe exhibits a similar step change in this period, though with more annual variability, and a greater range (from 3.63 to 7.90). For two years (2025-2026) the maximum is achieved for this cycle (7.90), which is attributed to high Mag Fe content being mined out of the Stage 2 Chief area, which quickly drops off once this area is mined through.

After the 2032-2033 ore supply issue, FeT grades remain relatively consistent at about 28.5, CaO climbs to a relatively stable 2.13, MgO to an equally stable 1.97, and Mag Fe to 5.74.

These expit mined grades are blended with RoM stockpile material at times to meet ore feed grade targets, and so in some years the expit grades and the crushed grades differ slightly, and any balance is maintained in the RoM stockpiles.

See Table 14.3.1 for the 2200 LoM Production Schedule related to the ex-pit mining activity for quantities and grades.

See Table 14.3.2 for the 2200 t/h LoM Production Schedule related to the RoM stockpiling and Crushing quantities and grades.

See Figure 14.3.1 for the 2200 t/h AMT and Ore mining by phase schedule graph.

Table 14.3.1: Life of Mine Production Schedule (Mining) – 2200 t/h

	Mining (Expit)									
Year	**All Material MT (dry)**	**Ore (>20%Fet) MT (dry)**	**Subgrade (<20%Fet) MT (dry)**	**Waste MT (dry)**	**SR T/T**	**FeT %**	**CaO %**	**MgO %**	**MagFe %**	**Mag %**
2012fc	20.24	7.22	0.58	12.44	1.80	28.18	2.42	2.77	10.74	14.92
2013	58.40	20.87	1.02	36.51	1.80	29.45	1.52	1.91	5.79	8.04
2014	73.00	26.09	1.52	45.39	1.80	29.12	1.25	1.47	4.95	6.87
2015	96.72	37.17	1.03	58.52	1.60	28.88	1.06	1.37	4.26	5.91
2016	100.65	36.31	1.04	63.30	1.77	29.35	1.17	1.42	4.29	5.95
2017	100.37	38.64	1.70	60.04	1.60	28.58	1.11	1.27	3.69	5.13
2018	100.37	37.53	1.59	61.25	1.67	28.83	1.35	1.25	3.63	5.04
2019	100.37	37.42	2.62	60.34	1.68	29.03	1.19	1.31	4.93	6.85
2020	100.65	38.77	2.68	59.21	1.60	28.78	1.35	1.42	5.13	7.12
2021	100.37	38.44	2.71	59.22	1.61	29.48	1.15	1.29	4.84	6.72
2022	100.37	38.62	3.91	57.84	1.60	28.54	1.68	1.64	5.71	7.94
2023	100.37	38.63	3.84	57.91	1.60	28.66	1.98	1.92	6.25	8.69
2024	100.65	38.70	1.34	60.61	1.60	28.67	1.79	1.85	6.16	8.56
2025	100.37	38.63	2.13	59.61	1.60	29.03	1.74	1.79	7.90	10.97
2026	100.37	38.65	1.43	60.30	1.60	29.30	1.77	1.88	7.88	10.95
2027	100.37	38.63	1.42	60.33	1.60	29.48	1.47	1.56	6.16	8.56
2028	100.65	38.74	0.46	61.45	1.60	28.92	1.78	1.77	6.05	8.41
2029	100.37	38.65	0.96	60.76	1.60	28.81	1.88	2.00	4.20	5.83
2030	100.37	38.62	3.04	58.71	1.60	28.67	1.46	1.70	4.95	6.87
2031	100.37	37.77	1.42	61.18	1.66	28.73	1.55	1.50	5.65	7.85
2032	100.65	17.84	0.11	82.70	4.64	28.83	0.24	0.30	1.17	1.63
2033	100.37	18.15	0.13	82.09	4.53	28.74	0.23	0.29	1.06	1.48
2034	100.37	38.61	3.11	58.65	1.60	27.88	2.17	1.88	6.31	8.76
2035	100.37	41.40	5.41	53.57	1.42	28.26	2.05	1.94	6.54	9.08
2036	81.08	42.09	1.65	37.34	0.93	28.40	2.16	1.95	5.57	7.73
2037	73.29	41.98	1.55	29.77	0.75	28.69	2.13	1.95	5.63	7.82
2038	67.35	41.98	2.30	23.07	0.60	28.64	2.15	1.98	5.58	7.75
2039	56.44	41.98	1.45	13.01	0.34	28.70	1.91	1.90	6.18	8.58
2040	49.58	39.62	1.67	8.29	0.25	28.22	2.42	2.13	4.93	6.85
2041	28.73	23.64	0.74	4.35	0.22	26.06	3.96	3.49	8.95	12.43
LoM Reserve	**2613.72**	**1051.38**	**54.57**	**1507.77**	**1.49**	**28.72**	**1.69**	**1.70**	**5.48**	**7.62**

- SRK removed 2012 mine movement from cash flow due to only being half year production. This decision was made to correctly calculate the NPV.
- SRK has noticed that 2032 and 2033 ore availability is an issue but this issue will be mitigated with more drilling and/or a re-phasing exercise of the mine plan. See previous commentary.

Table 14.3.2: Life of Mine Production Schedule (Stockpiling and Crushing) 2200 t/h

	Mining (Expit)	RoM Stockpiling			Blending (Crushed Material)					
Year	Ore (>20%Fet)	RoM Stockpiled	RoM Reclaimed	RoM Stock Balance	Crude Tonnes Crushed	FeT	CaO	MgO	MagFe	Mag
	MT (dry)	MT (dry)	MT (dry)	MT (dry)	MT (dry)	%	%	%	%	%
2012fc	7.22	0.00	0.00	0.00	7.22	28.18	2.42	2.77	10.74	14.92
2013	20.87	0.00	0.00	0.00	20.87	29.45	1.52	1.91	5.79	8.04
2014	26.09	0.00	0.00	0.00	26.09	29.12	1.25	1.47	4.95	6.87
2015	37.17	0.00	0.00	0.00	37.17	28.88	1.06	1.37	4.26	5.91
2016	36.31	0.00	0.00	0.00	36.31	29.35	1.17	1.42	4.29	5.95
2017	38.64	0.31	0.00	0.31	38.33	28.58	1.11	1.27	3.69	5.13
2018	37.53	0.00	0.31	0.00	37.85	28.83	1.35	1.25	3.63	5.04
2019	37.42	0.00	0.00	0.00	37.42	29.03	1.19	1.31	4.93	6.85
2020	38.77	0.34	0.00	0.34	38.43	28.78	1.35	1.42	5.13	7.12
2021	38.44	0.12	0.00	0.45	38.33	29.47	1.15	1.29	4.84	6.72
2022	38.62	0.30	0.00	0.75	38.33	28.55	1.68	1.64	5.70	7.92
2023	38.63	0.30	0.00	1.05	38.33	28.66	1.98	1.91	6.24	8.67
2024	38.70	0.27	0.00	1.33	38.43	28.67	1.79	1.85	6.16	8.56
2025	38.63	0.30	0.00	1.63	38.33	29.02	1.74	1.79	7.84	10.89
2026	38.65	0.32	0.00	1.95	38.33	29.29	1.77	1.88	7.88	10.95
2027	38.63	0.31	0.00	2.26	38.33	29.47	1.49	1.58	6.24	8.67
2028	38.74	0.31	0.00	2.57	38.43	28.95	1.77	1.76	6.06	8.42
2029	38.65	0.33	0.00	2.89	38.33	28.82	1.87	1.98	4.32	5.99
2030	38.62	0.30	0.00	3.19	38.33	28.68	1.49	1.72	4.90	6.81
2031	37.77	0.00	0.55	2.64	38.33	28.73	1.55	1.52	5.59	7.77
2032	17.84	0.00	2.64	0.00	20.48	28.82	0.41	0.45	1.74	2.42
2033	18.15	0.00	0.00	0.00	18.15	28.74	0.23	0.29	1.06	1.48
2034	38.61	0.29	0.00	0.29	38.33	27.88	2.17	1.88	6.31	8.76
2035	41.40	3.07	0.00	3.36	38.33	28.26	2.05	1.94	6.54	9.08
2036	42.09	3.66	0.00	7.02	38.43	28.39	2.15	1.95	5.64	7.83
2037	41.98	3.65	0.00	10.67	38.33	28.65	2.14	1.95	5.63	7.82
2038	41.98	3.65	0.00	14.32	38.33	28.64	2.15	1.97	5.59	7.77
2039	41.98	3.65	0.00	17.97	38.33	28.68	1.97	1.92	6.03	8.37
2040	39.62	1.19	0.00	19.16	38.43	28.36	2.28	2.07	5.27	7.32
2041	23.64	0.00	14.68	4.48	38.33	27.09	3.21	2.85	7.30	10.15
2042	0.00	0.00	4.48	0.00	4.48	27.09	3.21	2.85	7.30	10.15
LoM Reserve	**1051.38**				**1051.38**	**28.72**	**1.69**	**1.70**	**5.48**	**7.62**

- SRK removed 2012 mine movement from cash flow due to only being half year production. This decision was made to correctly calculate the NPV.
- SRK has noticed that 2032 and 2033 ore availability is an issue but this issue will be mitigated with more drilling and/or a re-phasing exercise of the mine plan. See previous commentary.

Figure 14.3.1: Life of Mine Production Schedule Graph - All Material Mining Distribution by Stage (left) and Ore Mining Distribution by Stage (right)

- All units are expressed in dry metric tonnes.
- SRK has noticed that 2032 and 2033 ore availability is an issue but this issue will be mitigated with more drilling and/or a re-phasing exercise of the mine plan. See previous commentary.

Schedule 2 Results: 2400 t/h (Forecast Operational Context)

The 2400 t/h (Mill Throughput Rate) production schedule results indicate a steady crude ore feed rate for the first 20 years of approximately 42 MT (dry) / annum to the mills at approximately a 1.7 Strip Ratio (once the ramp-up period is complete) and a maximum all material movement of approximately 113 MT (dry) / annum. This is managed by the balancing of the in-pit staging (for development of all 9 phases of the reserve) for both strip ratio and grade. After these first periods, the production schedule fails to meet the total ore supply requirement for one year, even though ~3.5 MT (dry) stockpiled ore is blended in order to mitigate (this ore was stockpiled ahead of the known shortfall from excess in the 20 year stable period). This ore supply issue has manifested at the same point in the pit development as in the 2200 t/h case. Work is ongoing to further understand this ore supply issue (~2031), which is in part due to the geometry and grade/SR variability of the ore body at this achieved depth, and in part due to the staging interaction of the current phasing strategy (this event takes place after the completion of the Stage 3 Main phase). Further scheduling will be conducted by Cliffs in 2014 which will attempt to rectify this with further drilling information, optimization, stage re-design and schedule algorithm manipulation, with additional efforts to further smooth production physicals, grade variability, and reduction in stockpiling).

Mined Total Fe grade (FeT) for the first 20 years remains relatively stable at an average grade of 28.9% (ranging between 28.5% and 29.6%). CaO and MgO are more variable in this schedule, with a larger drop (1.0-1.2) in the early years followed by a steady climb to 1.7-1.8. Mag Fe exhibits a similar gradational change in this period, though with more annual variability, and a greater range (from 3.33 to 7.33). For two years (2025-2026) the maximum is achieved for this cycle (7.33), which is attributed to high Mag Fe content being mined out of the Stage 2 Chief area, which quickly drops off once this area is mined through. These grade profiles are largely consistent with the 2012 Baseline Schedule.

After the 2031 ore supply issue, FeT grades remain relatively consistent at about 28.5, CaO and MgO climb and remain variable above 2.00, and Mag Fe remains similar.

These expit mined grades are blended with RoM stockpile material at times to meet ore feed grade targets, and so in some years the expit grades and the crushed grades differ slightly, and any balance is maintained in the RoM stockpiles.

See Table 14.3.3 for the 2400 t/h LoM Production Schedule related to the ex-pit mining activity for quantities and grades.

See Table 14.3.4 for the 2400 t/h LoM Production Schedule related to the RoM stockpiling and Crushing quantities and grades.

See Figure 14.3.2 for the 2400 t/h AMT and Ore mining by phase schedule graph.

Table 14.3.3: Life of Mine Production Schedule (Mining) - 2400 t/h

Mining (Expit)										
Year	All Material MT (dry)	Ore (>20%Fet) MT (dry)	Subgrade (<20%Fet) MT (dry)	Waste MT (dry)	SR T/T	FeT %	CaO %	MgO %	MagFe %	Mag %
2012fc	20.24	7.22	0.58	12.44	1.80	28.18	2.42	2.77	10.74	14.92
2013	58.40	20.87	1.02	36.51	1.80	29.45	1.52	1.91	5.79	8.04
2014	73.00	27.06	1.49	44.45	1.70	29.20	1.25	1.45	4.96	6.88
2015	98.55	36.57	1.05	60.94	1.69	29.33	0.83	1.08	3.33	4.62
2016	109.80	40.78	1.13	67.89	1.69	29.20	1.06	1.33	4.12	5.73
2017	113.15	42.43	2.11	68.61	1.67	28.83	1.22	1.34	3.89	5.40
2018	113.15	41.85	1.46	69.84	1.70	28.65	1.26	1.27	3.79	5.27
2019	113.15	41.60	3.15	68.40	1.72	28.86	1.11	1.24	4.47	6.21
2020	113.46	39.88	2.65	70.93	1.85	28.97	1.44	1.51	7.00	9.72
2021	113.15	41.55	2.39	69.21	1.72	28.56	1.59	1.75	3.92	5.44
2022	113.15	41.98	3.81	67.36	1.70	28.53	1.69	1.72	4.95	6.88
2023	113.15	42.01	5.90	65.25	1.69	28.70	1.73	1.59	6.07	8.43
2024	113.46	42.09	1.67	69.69	1.70	29.00	1.68	1.68	7.18	9.97
2025	113.15	41.98	1.10	70.07	1.70	29.57	1.47	1.71	7.33	10.18
2026	113.15	41.98	1.54	69.63	1.70	29.41	1.67	1.86	7.30	10.14
2027	113.15	41.97	0.63	70.55	1.70	29.17	1.62	1.78	6.15	8.55
2028	113.46	45.52	0.92	67.02	1.49	28.45	1.91	1.93	5.40	7.51
2029	113.15	42.03	2.53	68.59	1.69	28.57	1.87	1.84	5.35	7.43
2030	113.15	42.01	2.21	68.93	1.69	28.61	1.71	1.50	5.36	7.45
2031	113.15	35.49	2.01	75.65	2.19	27.61	2.37	2.09	8.18	11.36
2032	113.46	41.34	2.86	69.26	1.74	28.28	2.13	2.04	5.34	7.42
2033	88.44	41.98	4.70	41.76	1.11	28.10	2.02	1.86	6.19	8.60
2034	88.57	41.98	1.92	44.68	1.11	28.12	2.16	1.94	5.02	6.97
2035	70.59	41.98	2.17	26.45	0.68	28.58	2.14	1.94	5.54	7.70
2036	70.36	42.09	1.35	26.92	0.67	28.56	2.19	2.06	5.80	8.05
2037	64.69	41.98	1.32	21.40	0.54	28.52	2.24	2.01	4.97	6.91
2038	53.90	38.38	0.93	14.59	0.40	28.44	1.81	1.69	4.78	6.64
2039	5.54	4.79	0.00	0.75	0.16	29.28	0.17	0.19	1.36	1.88
2040										
LoM Reserve	**2613.72**	**1051.38**	**54.57**	**1507.77**	**1.49**	**28.72**	**1.69**	**1.7**	**5.48**	**7.62**

- SRK removed 2012 mine movement from cash flow due to only being half year production. This decision was made to correctly calculate the NPV.
- SRK has noticed that 2032 and 2033 ore availability is an issue but this issue will be mitigated with more drilling and/or a re-phasing exercise of the mine plan. See previous commentary.

Table 14.3.4: Life of Mine Production Schedule (Stockpiling and Crushing) - 2400 t/h

Year	Mining (Expit) Ore (>20% Fet)	RoM Stockpiling			Blending (Crushed Material)					
		RoM Stockpiled	RoM Reclaimed	RoM Stock Balance	Crude Tonnes Crushed	FeT	CaO	MgO	Mag Fe	Mag
	MT (dry)	MT (dry)	MT (dry)	MT (dry)	MT (dry)	%	%	%	%	%
2012fc	7.22	-	-	-	7.22	28.18	2.42	2.77	10.74	14.92
2013	20.87	-	-	-	20.87	29.45	1.52	1.91	5.79	8.04
2014	27.06	0.78	-	0.78	26.28	29.20	1.25	1.45	4.96	6.88
2015	36.57	-	0.78	(0.00)	37.35	29.33	0.84	1.08	3.36	4.67
2016	40.78	-	-	(0.00)	40.78	29.20	1.06	1.33	4.12	5.73
2017	42.43	0.46	-	0.46	41.98	28.83	1.22	1.34	3.89	5.40
2018	41.85	-	0.12	0.34	41.98	28.65	1.26	1.28	3.79	5.27
2019	41.60	-	0.34	(0.00)	41.93	28.85	1.11	1.24	4.47	6.20
2020	39.88	-	-	(0.00)	39.88	28.97	1.44	1.51	7.00	9.72
2021	41.55	-	-	(0.00)	41.55	28.56	1.59	1.75	3.92	5.44
2022	41.98	0.00	-	0.00	41.98	28.53	1.69	1.72	4.95	6.88
2023	42.01	0.03	-	0.04	41.98	28.70	1.73	1.59	6.07	8.43
2024	42.09	0.00	-	0.04	42.09	29.00	1.68	1.68	7.18	9.97
2025	41.98	0.00	-	0.04	41.98	29.57	1.47	1.71	7.33	10.18
2026	41.98	0.01	-	0.05	41.98	29.41	1.67	1.86	7.30	10.14
2027	41.97	-	0.00	0.05	41.98	29.17	1.62	1.78	6.15	8.55
2028	45.52	3.43	-	3.48	42.09	28.45	1.91	1.93	5.41	7.51
2029	42.03	0.06	-	3.54	41.98	28.56	1.87	1.84	5.36	7.44
2030	42.01	0.03	-	3.57	41.98	28.60	1.72	1.53	5.36	7.45
2031	35.49	-	3.57	(0.00)	39.06	27.70	2.31	2.04	7.92	11.00
2032	41.34	-	-	(0.00)	41.34	28.28	2.13	2.04	5.34	7.42
2033	41.98	0.00	-	0.00	41.98	28.10	2.02	1.86	6.19	8.60
2034	41.98	0.00	-	0.00	41.98	28.12	2.16	1.94	5.02	6.97
2035	41.98	0.00	-	0.00	41.98	28.58	2.14	1.94	5.54	7.70
2036	42.09	0.00	-	0.00	42.09	28.56	2.19	2.06	5.80	8.05
2037	41.98	0.00	-	0.00	41.98	28.52	2.24	2.01	4.97	6.91
2038	38.38	-	0.00	(0.00)	38.38	28.44	1.81	1.69	4.78	6.64
2039	4.79	-	-	(0.00)	4.79	29.28	0.17	0.19	1.36	1.88
2040										
LoM Reserve	**1051.38**				**1051.38**	**28.72**	**1.69**	**1.7**	**5.48**	**7.62**

- SRK removed 2012 mine movement from cash flow due to only being half year production. This decision was made to correctly calculate the NPV.
- SRK has noticed that 2032 and 2033 ore availability is an issue but this issue will be mitigated with more drilling and/or a re-phasing exercise of the mine plan. See previous commentary.

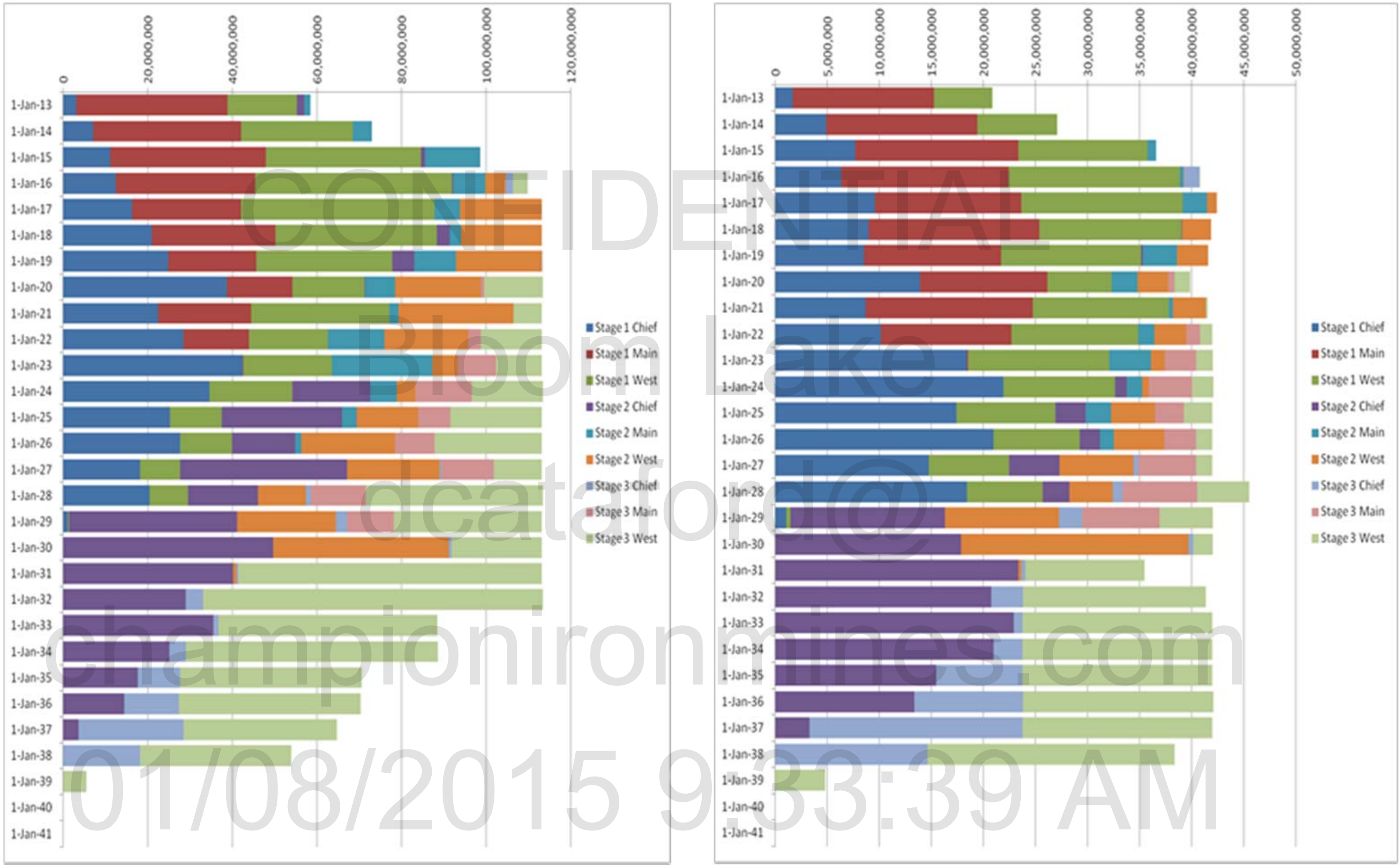
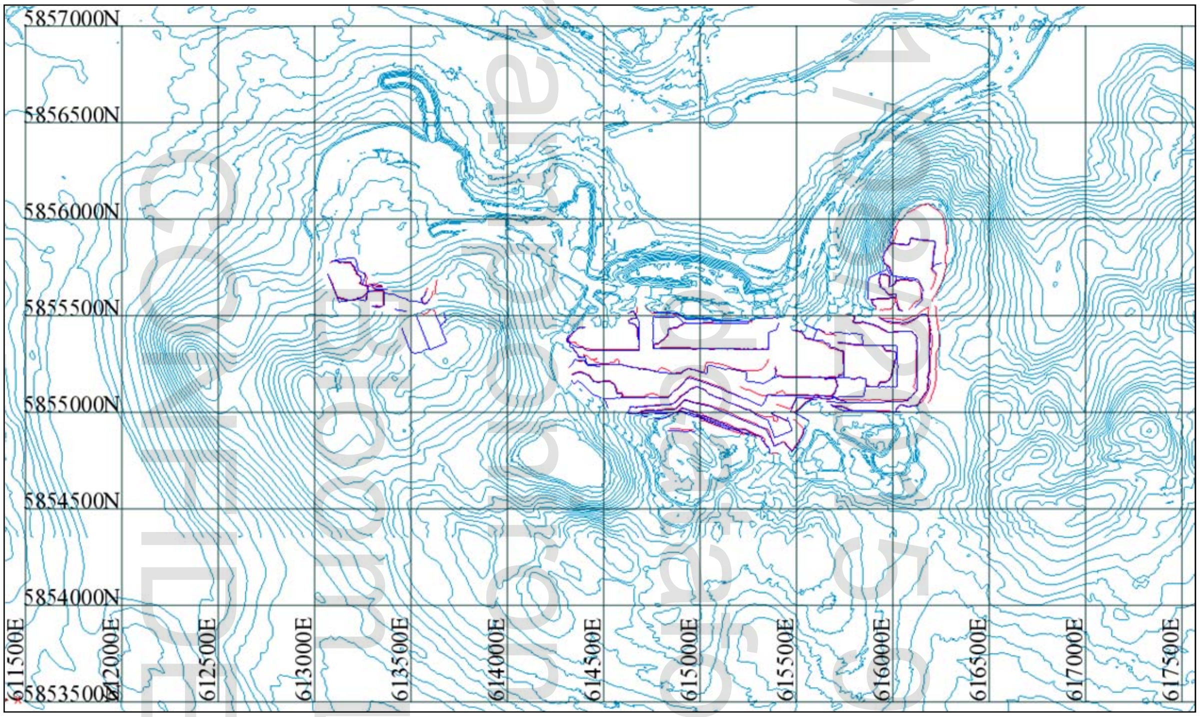


Figure 14.3.2: Life of Mine Production Schedule Graph - All Material Mining Distribution by Stage (left) and Ore Mining Distribution by Stage (right)

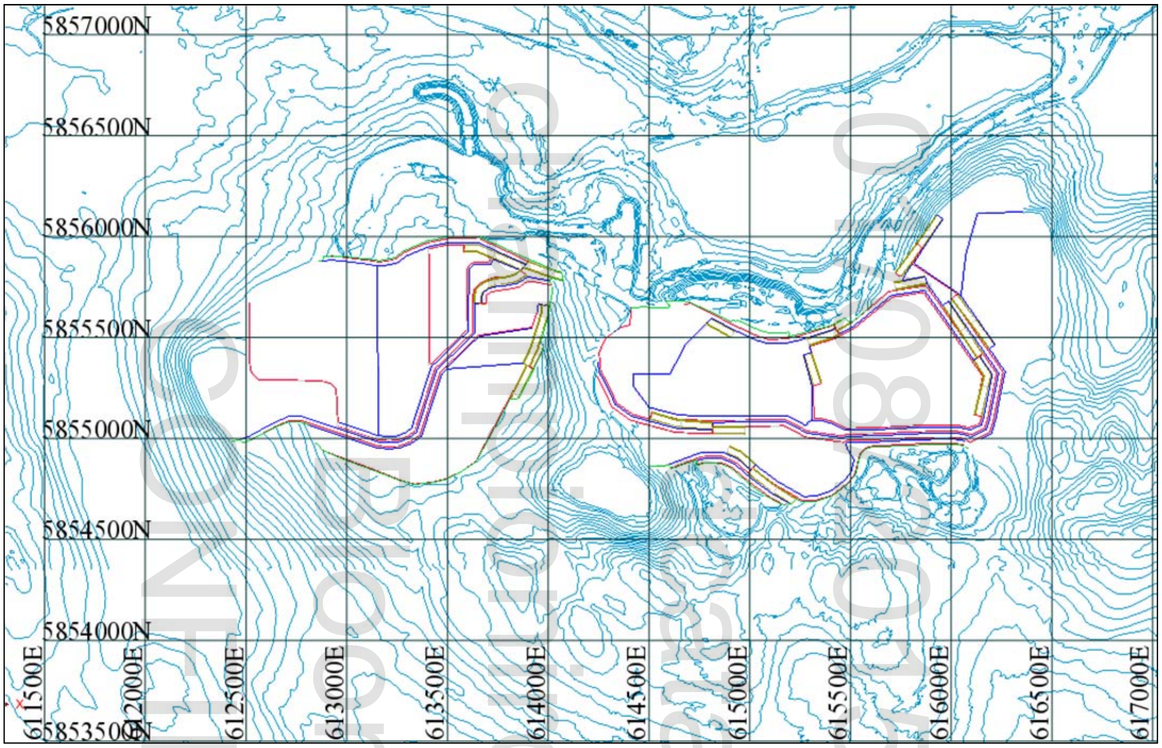
- All units are expressed in dry metric tonnes.
- SRK has noticed that 2031 ore availability is an issue but this issue will be mitigated with more drilling and/or a re-phasing exercise of the mine plan. See previous commentary.

The following maps indicate the development of the reserve through face position advance as the pit is mined by phase to its final position. These maps directly correspond to the 2012 Baseline Case (2200 t/h) schedule. It can be noted that the variance between both schedules is quite minimal at this scale of presentation, so for simplicity only one set of maps has been generated.



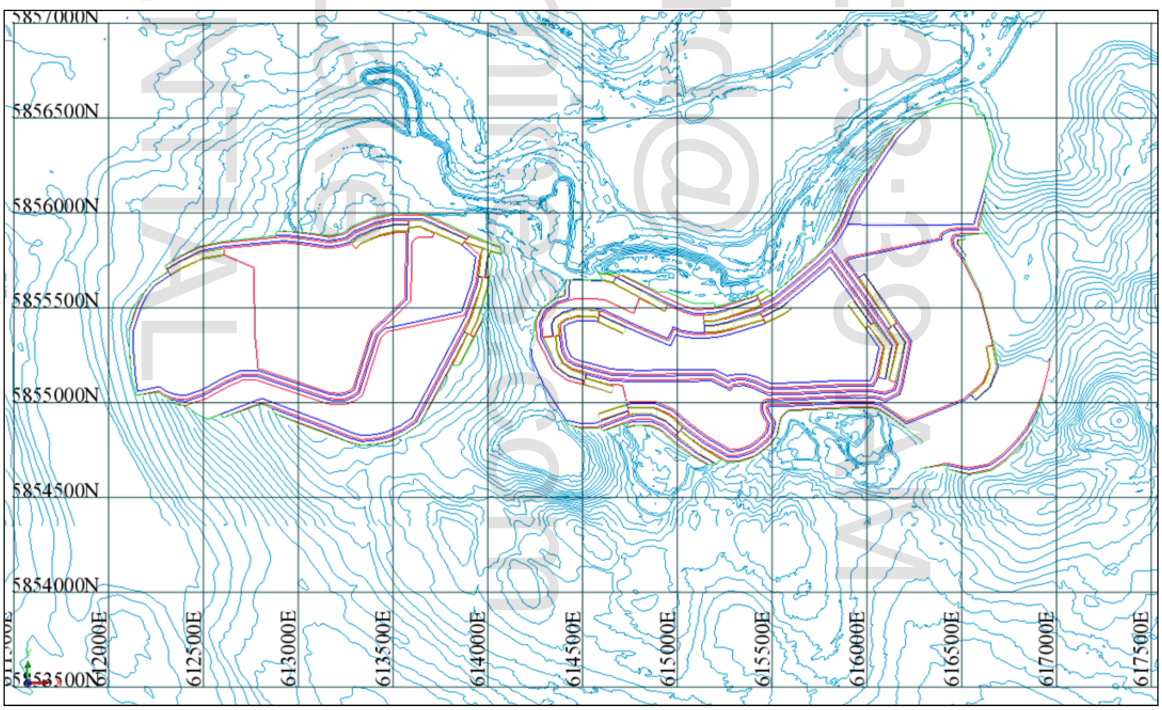
Source: Cliffs, 2012

Figure 14.3.3: Bloom Lake Final 2012 End of Year (EOY) Pit Topography



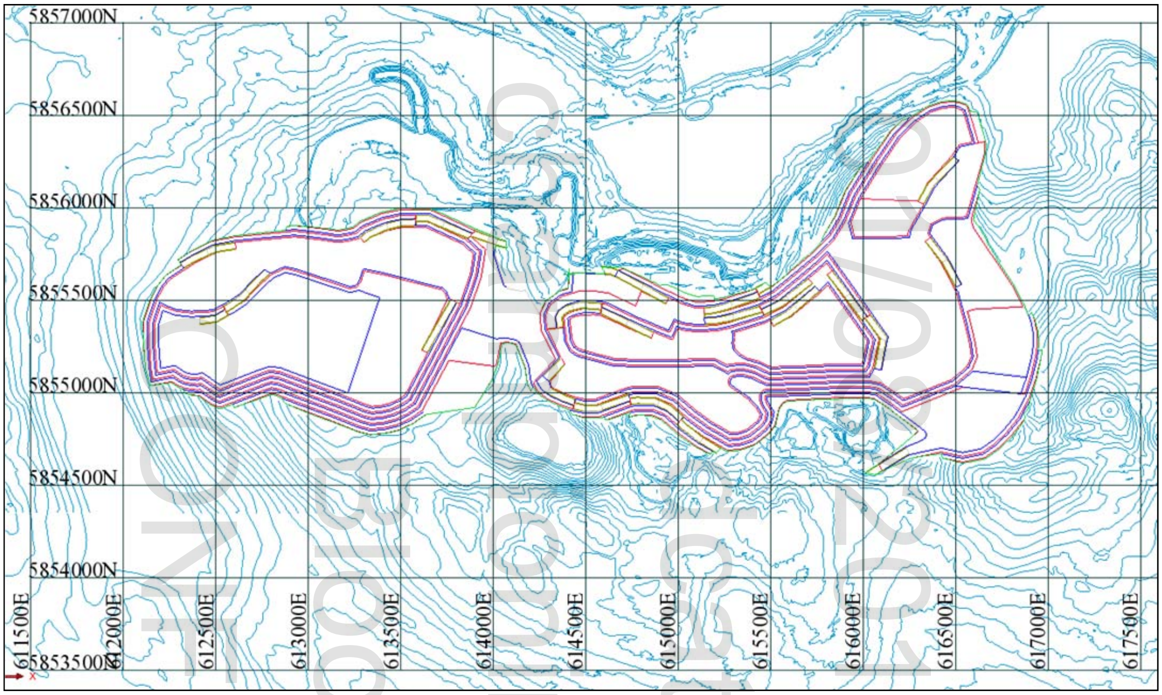
Source: Cliffs, 2012

Figure 14.3.4: Bloom Lake Final 2016 End of Year (EOY) Pit Topography



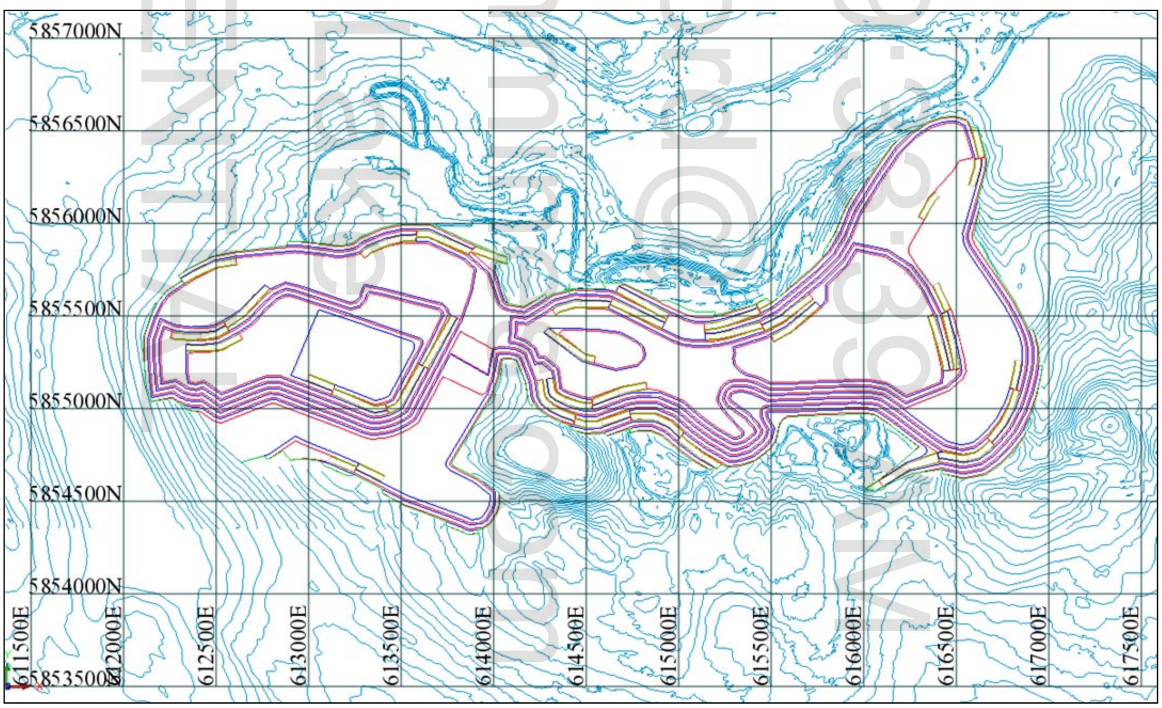
Source: Cliffs, 2012

Figure 14.3.5: Bloom Lake Final 2020 End of Year (EOY) Pit Topography



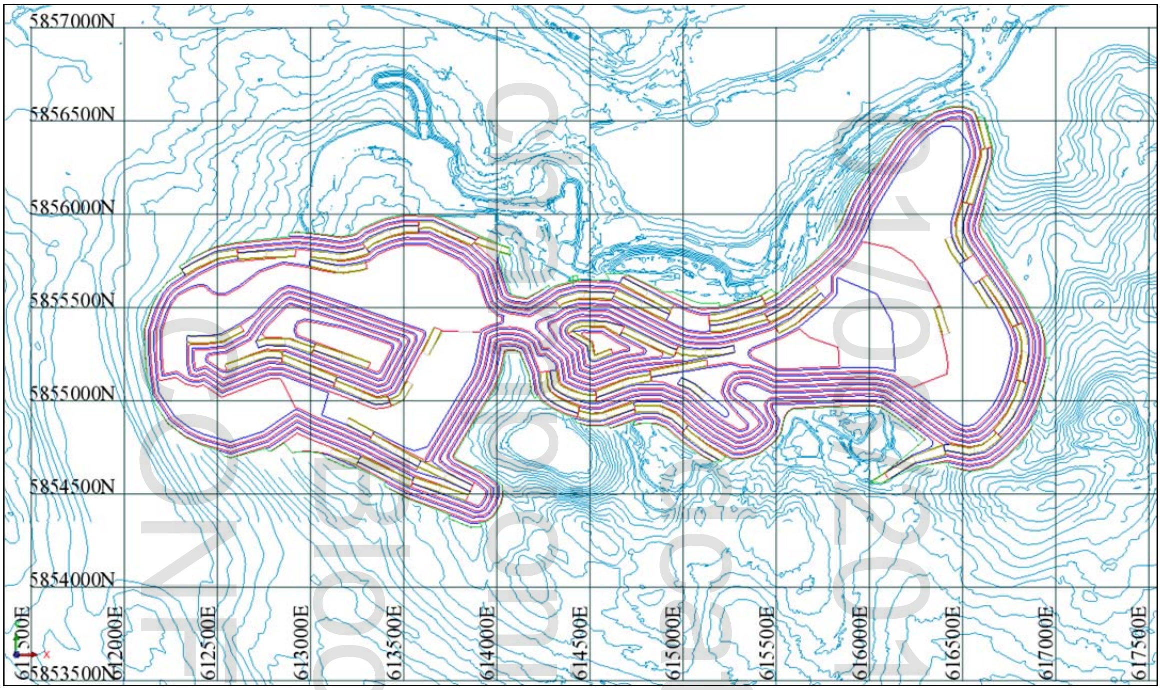
Source: Cliffs, 2012

Figure 14.3.6: Bloom Lake Final 2024 End of Year (EOY) Pit Topography



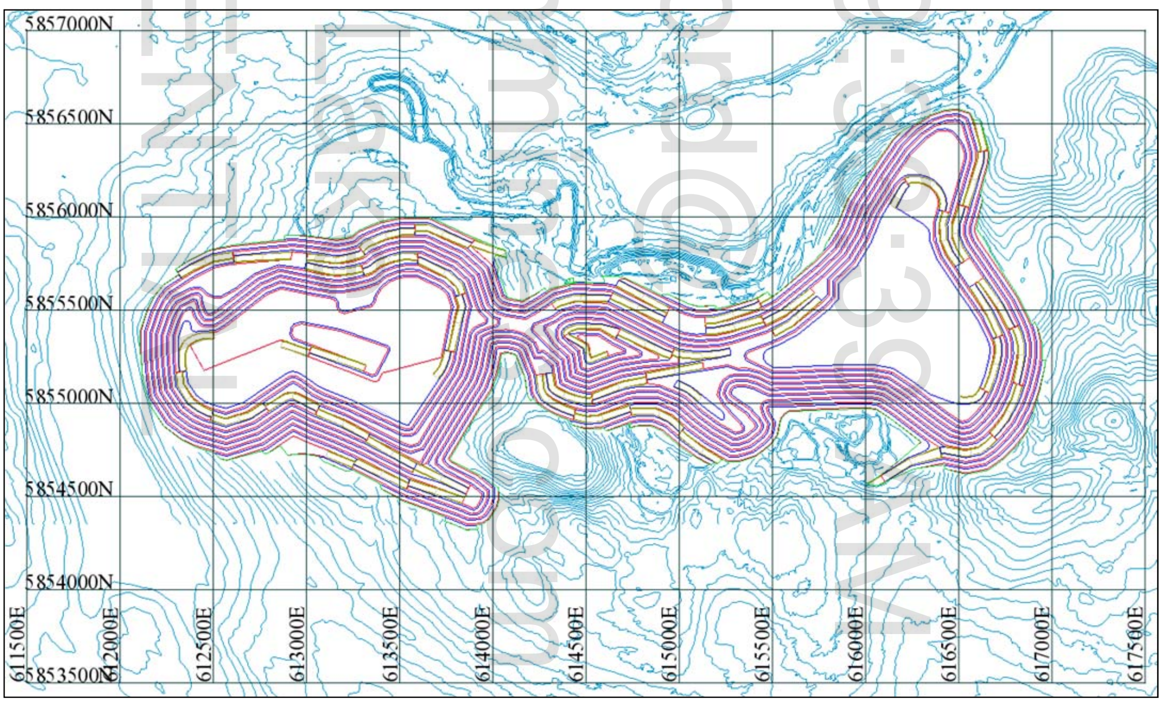
Source: Cliffs, 2012

Figure 14.3.7: Bloom Lake Final 2028 End of Year (EOY) Pit Topography



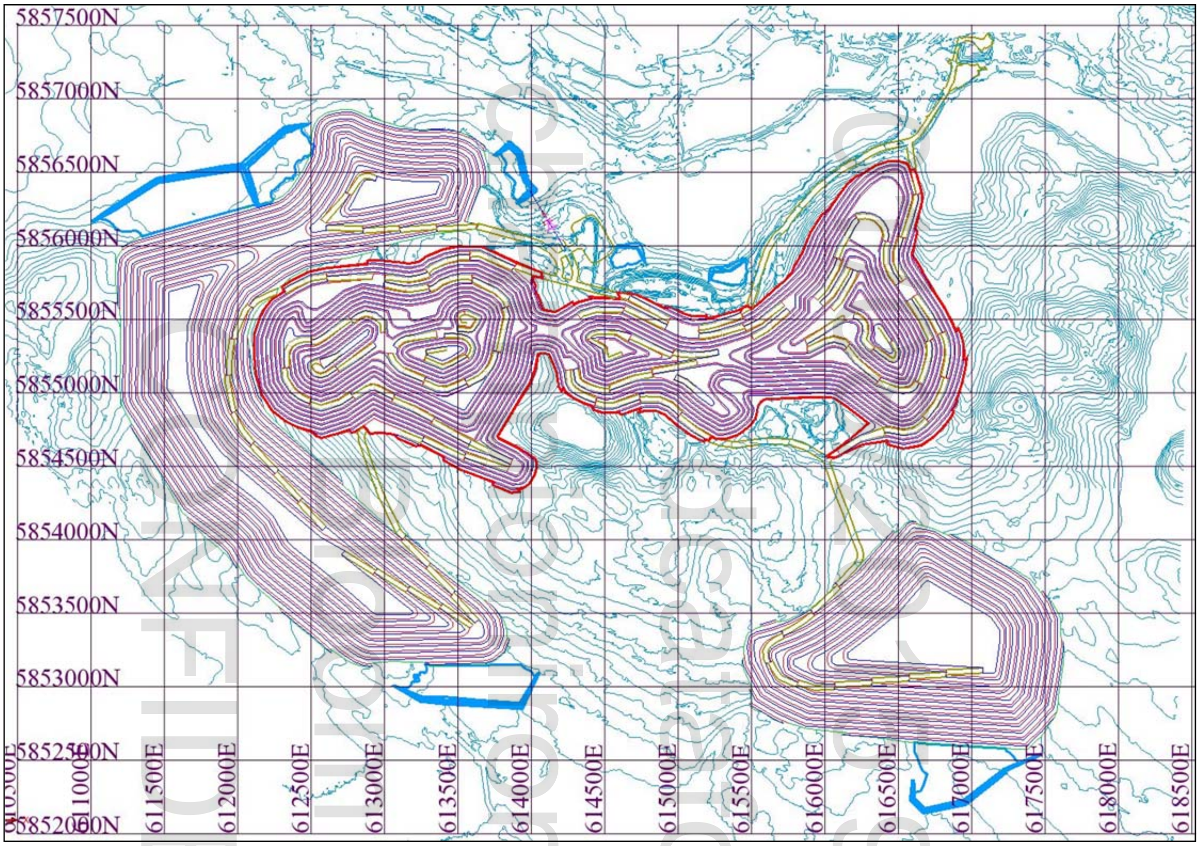
Source: Cliffs, 2012

Figure 14.3.8: Bloom Lake Final 2032 End of Year (EOY) Pit Topography



Source: Cliffs, 2012

Figure 14.3.9: Bloom Lake Final 2036 End of Year (EOY) Pit Topography



Source: Cliffs, 2012

Figure 14.3.10: Bloom Lake Final (2042) End of Life of Mine Topography

14.3.1 Dilution, SMU and Bench Configuration

The block model is based on 10 m x 10 m x 14 m blocks and represents the Selective Mining Unit (SMU) in relation to cut-off grade and subsequent dilution. Where the interpretation of the mineralized rock intersects a block model block centroid, the block within the mineralized shape is recorded. The flagging of ore type was based on block centroid and this accounts for the location and placement of the ore contact. Because the contact of waste and ore is clearly visible, minimal dilution is expected. If bigger equipment is selected, Bloom Lake may see an increase in dilution. For all the pit optimization exercises, a 5% material recovery loss was applied.

14.3.2 Grade Control

In order to assess grade control sample spacing requirements, Bloom Lake is planning to conduct a preliminary sample spacing analysis using the existing grade control data set from the mineralized material. SRK understands that Bloom Lake is planning to conduct grade control using standard drilling techniques and assumes that the resulting sample data will be statistically similar to the exploration data. SRK also assumes that the sample spacing requirements for the grade control process will be based on statistics of the Fe analyses. The optimal grade control sample spacing is 7 m x 7 m in ore on a 14 m bench, but currently 1 of every 4 holes is sampled. Assay results will be stored in a grade control database and subsequently analyzed by grade control engineers.

Engineers will then tape out waste and grade bin designation on the pit face for the excavators to dig to.

14.3.3 Reclamation

The overburden relocated by bulldozers or through re-handling, will be re-graded in preparation for reclamation. As mining operations are optimized during production, it will be necessary to control the reclamation to ensure disturbed land areas are kept below annual permitted disturbance levels. Soil from clear and grubbing operations in new pit strips will be directly transported to contoured spoil piles to finish reclamation.

14.3.4 Primary Production Fleet

The production equipment fleet is based on the life-of-mine (LoM) production schedules, and evaluated for each schedule. The equipment requirements are estimated based on the mining of 1,562.3 Mt of waste/subgrade and 1,051.4 Mt of ore as detailed in the economic model (in situ schedule). In addition, there will be small amounts of crusher stockpile rehandling. The primary equipment fleet consists of large front hydraulic shovels and off-road dump trucks. In addition to this primary equipment there will also be front end loaders (FEL's), dozers, rubber tire dozers (RTD's), graders, water trucks, and backhoe.

As the pre-production fleet will be utilized during full mine production, the production schedule and hence fleet estimation was designed to maintain a balanced haul fleet through the end of the life of the mine. After 2014, the stripping ratio begins to increase and required haul units increase proportionally along with longer hauls.

Haulage cycle times were estimated using Minesched Software Haulage Module. Table 14.3.4.1 illustrates the truck speeds used for the estimation of the primary truck fleet. A site maximum speed limit of 50 km/hr was employed. For the large equipment, SRK used a maximum speed limit of 38 km/hr based on safety and speed restrictions at similar size mines.

Table 14.3.4.1: Truck Speeds Applied (Komatsu 930E & 830E and CAT 793)

Gradient (%)	Loaded (km/hr)	Empty (km/hr)
+10	10.4	20.0
+8	11.5	22.0
+6	14.9	26.0
+4	17.0	30.0
+2	17.0	32.0
0	22.0	35.0
-2	24.0	38.0
-4	20.0	36.0
-6	19.0	32.0
-8	17.0	25.0
-10	15.0	15.0

Cliffs is currently building an in-pit crusher between the west and east pits and plans to send a minimum of 50% of the ore material to the in-pit crusher. Three routes were calculated: pit ore to in-pit crusher; pit ore to the current crusher near the processing plant and pit waste to two different dumps (west and south). The ore haulage estimation was averaged with approximately 50% of the material going to the in-pit crusher and 50% to the current crusher near the processing plant. The haulage estimation running cycle times do not include any loading, spotting and dumping time. The loading, spot and dumping time was added separately into the fleet estimator. Table 14.3.4.2 presents the truck running cycle times by year and material type for the 2012 Baseline Case (2200 t/h) Mine Production Schedule, and Table 14.3.4.3 for the Forecast (2400 t/h) Mine Production Schedule.

Table 14.3.4.2: 2012 Operational Baseline Case (2200 t/h) Truck Running Cycle Information

Year	Truck Cycles (*not including any loading or dumping time)										
	300T Equivalent Trucks	Trk Hours (000's)	Total Distance (000,000's)	Scheduled Hours/ Truck	Avg Cycle	Trk PL	Availability	Utilization	Efficiency	Productivity	Calc Truck Hours
	Trucks	Hr (K)	Km (M)	Hr	Min	T	%	%	%	%	Hr
2012fc	16	45.70	3.75	2933.11	41.19	300	0.82	0.90	0.90	0.66	2933
2013	20	119.00	3.59	5818.39	38.66	300	0.82	0.90	0.90	0.66	5818
2014	21	121.68	3.27	5818.39	31.27	300	0.82	0.90	0.90	0.66	5818
2015	24	142.31	3.06	5818.39	28.62	300	0.82	0.90	0.90	0.66	5818
2016	28	165.14	3.22	5834.34	31.48	300	0.82	0.90	0.90	0.66	5834
2017	27	157.73	3.14	5818.39	30.19	300	0.82	0.90	0.90	0.66	5818
2018	26	150.02	2.98	5818.39	28.92	300	0.82	0.90	0.90	0.66	5818
2019	28	162.24	3.30	5818.39	31.48	300	0.82	0.90	0.90	0.66	5818
2020	27	159.82	3.28	5834.34	30.68	300	0.82	0.90	0.90	0.66	5834
2021	28	163.48	3.24	5818.39	31.45	300	0.82	0.90	0.90	0.66	5818
2022	30	172.78	3.36	5818.40	32.99	300	0.82	0.90	0.90	0.66	5818
2023	31	180.44	3.22	5818.40	35.11	300	0.82	0.90	0.90	0.66	5818
2024	35	205.63	3.45	5834.33	36.69	300	0.82	0.90	0.90	0.66	5834
2025	33	192.58	3.47	5818.39	36.54	300	0.82	0.90	0.90	0.66	5818
2026	32	186.05	3.47	5818.38	35.12	300	0.82	0.90	0.90	0.66	5818
2027	34	200.24	3.88	5818.40	38.32	300	0.82	0.90	0.90	0.66	5818
2028	34	196.84	3.74	5834.33	37.09	300	0.82	0.90	0.90	0.66	5834
2029	34	196.89	3.80	5818.39	36.48	300	0.82	0.90	0.90	0.66	5818
2030	38	221.06	4.27	5818.39	41.26	300	0.82	0.90	0.90	0.66	5818
2031	40	232.26	4.35	5818.39	43.05	300	0.82	0.90	0.90	0.66	5818
2032	38	219.63	4.34	5834.33	39.22	300	0.82	0.90	0.90	0.66	5834
2033	40	233.15	4.72	5818.39	42.09	300	0.82	0.90	0.90	0.66	5818
2034	40	230.67	4.57	5818.40	43.35	300	0.82	0.90	0.90	0.66	5818
2035	42	243.51	4.87	5818.39	46.04	300	0.82	0.90	0.90	0.66	5818
2036	34	200.45	5.04	5834.34	48.12	300	0.82	0.90	0.90	0.66	5834
2037	32	187.12	5.10	5818.38	49.64	300	0.82	0.90	0.90	0.66	5818
2038	30	172.34	5.08	5818.40	49.34	300	0.82	0.90	0.90	0.66	5818
2039	21	121.57	4.14	5818.38	47.32	300	0.82	0.90	0.90	0.66	5818
2040	19	109.47	4.31	5834.33	45.80	300	0.82	0.90	0.90	0.66	5834
2041	13	73.01	4.86	5818.39	49.14	300	0.82	0.90	0.90	0.66	5834
2042	-	0.00			0.00	300	0.82	0.90	0.90	0.66	5834

*SRK used the average cycle time to re-calculate mining OPEX and CAPEX.

Table 14.3.4.3: Forecast Case (2400 t/h) Truck Running Cycle Information

Year	Truck Cycles (*not including any loading or dumping time)										
	300T Equivalent Trucks	Trk Hours (000's)	Total Distance (000,000's)	Scheduled Hours/ Truck	Avg Cycle	Trk PL	Availability	Utilization	Efficiency	Productivity	Calc Truck Hours
	Trucks	Hr (K)	Km (M)	Hr	Min	T	%	%	%	%	Hr
2012fc	16	45.70	3.75	2933.11	41.19	300	0.82	0.90	0.90	0.66	2933
2013	20	119.00	3.59	5818.39	38.66	300	0.82	0.90	0.90	0.66	5818
2014	21	121.40	3.27	5818.40	31.31	300	0.82	0.90	0.90	0.66	5818
2015	25	145.49	3.07	5818.39	28.23	300	0.82	0.90	0.90	0.66	5818
2016	30	177.15	3.17	5834.33	30.72	300	0.82	0.90	0.90	0.66	5834
2017	30	174.21	3.04	5818.39	29.57	300	0.82	0.90	0.90	0.66	5818
2018	30	176.69	3.14	5818.39	30.09	300	0.82	0.90	0.90	0.66	5818
2019	31	180.67	3.23	5818.39	31.08	300	0.82	0.90	0.90	0.66	5818
2020	33	190.81	3.46	5834.32	32.95	300	0.82	0.90	0.90	0.66	5834
2021	32	186.07	3.25	5818.40	31.03	300	0.82	0.90	0.90	0.66	5818
2022	35	200.93	3.53	5818.39	33.72	300	0.82	0.90	0.90	0.66	5818
2023	38	220.78	3.61	5818.39	37.34	300	0.82	0.90	0.90	0.66	5818
2024	43	252.80	3.53	5834.33	40.29	300	0.82	0.90	0.90	0.66	5834
2025	37	217.72	3.45	5818.39	36.31	300	0.82	0.90	0.90	0.66	5818
2026	37	216.30	3.61	5818.39	36.67	300	0.82	0.90	0.90	0.66	5818
2027	40	234.13	3.90	5818.39	39.50	300	0.82	0.90	0.90	0.66	5818
2028	40	234.77	3.90	5834.34	38.83	300	0.82	0.90	0.90	0.66	5834
2029	41	239.93	4.15	5818.39	39.83	300	0.82	0.90	0.90	0.66	5818
2030	45	264.67	4.51	5818.39	43.53	300	0.82	0.90	0.90	0.66	5818
2031	42	242.85	4.25	5818.39	39.90	300	0.82	0.90	0.90	0.66	5818
2032	44	258.01	4.58	5834.33	42.71	300	0.82	0.90	0.90	0.66	5834
2033	36	208.69	4.72	5818.39	45.21	300	0.82	0.90	0.90	0.66	5818
2034	39	227.32	5.16	5818.39	48.76	300	0.82	0.90	0.90	0.66	5818
2035	31	179.66	5.05	5818.40	48.59	300	0.82	0.90	0.90	0.66	5818
2036	31	182.71	5.21	5834.32	49.77	300	0.82	0.90	0.90	0.66	5834
2037	29	169.21	5.21	5818.39	50.59	300	0.82	0.90	0.90	0.66	5818
2038	19	111.34	4.03	5818.41	48.47	300	0.82	0.90	0.90	0.66	5818
2039	2	12.89	4.59	5818.52	48.72	300	0.82	0.90	0.90	0.66	5818
2040	-	0.00			0.00	300	0.82	0.90	0.90	0.66	5834

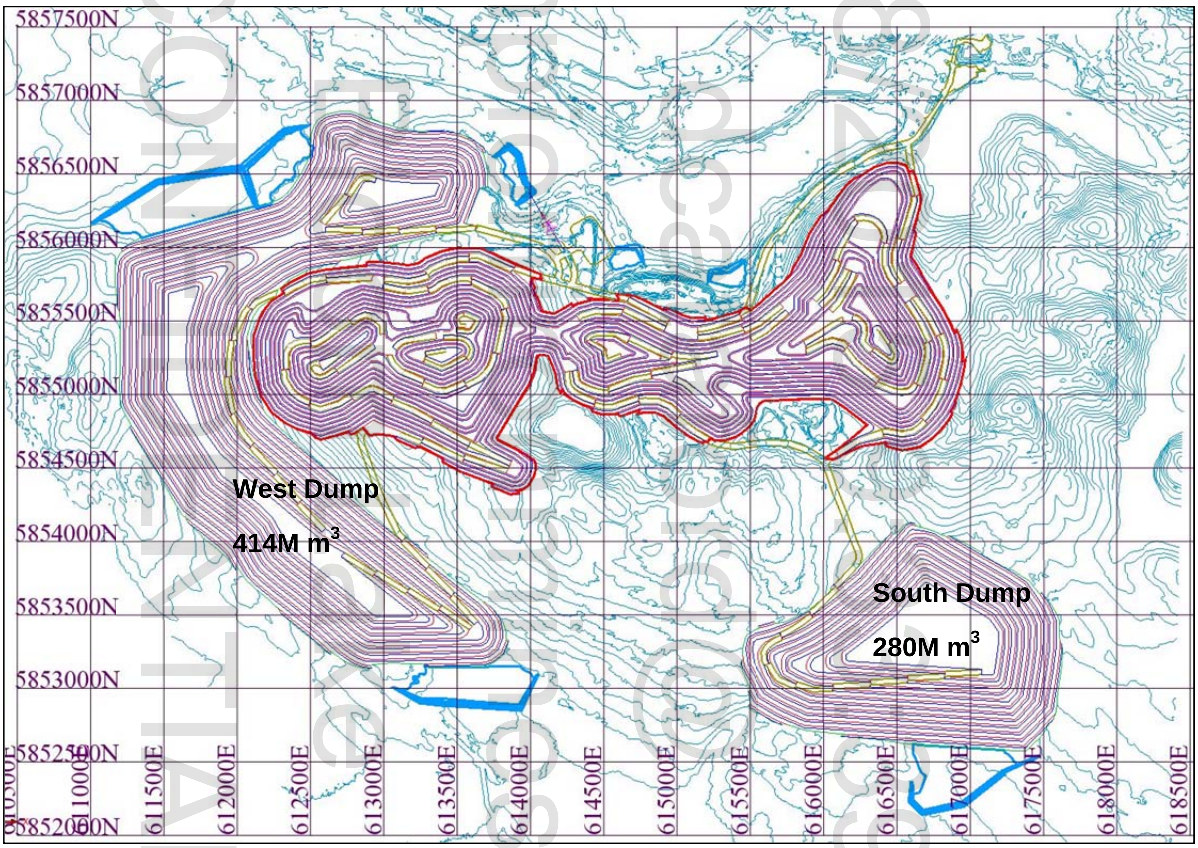
*SRK used the average cycle time to re-calculate mining OPEX and CAPEX.

14.3.5 Waste Dumps

Due to the increase in strip ratio and larger pit foot print, and in consideration of water management issues involved with the operating environment, two waste stockpile (dump) designs were created to accommodate the waste and low grade material mined. Swell factors (in situ to dumped) are estimated to be 25%. Further swell tests are planned to confirm assumptions. Presently, there is an additional 5% contingency capacity designed for accommodate positive variances in waste requirements in the current dump designs. All waste dump designs are connected to the final and stage pit designs by a master plan road concept, as well as a surface water management plan. Table 14.3.5.1 shows the waste dump parameters used for the design.

Table 14.3.5.1: Bloom Lake Waste Dump Parameters

Parameter	Unit	Value
Overall Slope Angle	Degrees	26
Batter Angle	Degrees	35
Bench Height	m	20
Berm Width	m	15
Ramp Width – 2 way	m	35
Ramp Width – 1 way	m	25
Ramp Gradient (Shortest)	%	8
Dump Density	g/cm^3	2



Source: Cliffs, 2012

Figure 14.3.5.1: Bloom Lake Waste Dump Design and Capacities

The waste dump design and staging strategies are consistent between both production schedule scenarios.

14.3.6 Current Mining Fleet

Table 14.3.6.1 shows the current mining fleet (January 9, 2013).

Table 14.3.6.1: Bloom Lake Current Mining Equipment (January 9, 2013)

Type	Model	Capacity	Hours
Shovel	Terex RH340	33 yd³	16,118
Shovel	Terex RH340	33 yd³	16,480
Shovel	CAT 6060	33 yd³	2,682
Shovel	Komatsu PC4000	28 yd³	1
Loader #304	LeTourneau 1850	22 yd³	13,006
Loader #305	LeTourneau 1850	22 yd³	1,711
Loader #306	Komatsu WA1200	18.5 yd³	2,240
Loader #307	Komatsu WA1200	18.5 yd³	2,057
Truck #101	CAT 793D	240 tons (short)	17,458
Truck #102	CAT 793D	240 tons (short)	17,678
Truck #103	CAT 793D	240 tons (short)	18,216
Truck #104	CAT 793D	240 tons (short)	18,722
Truck #105	CAT 793D	240 tons (short)	18,087
Truck #106	CAT 793D	240 tons (short)	18,211
Truck #107	CAT 793D	240 tons (short)	18,261
Truck #108	CAT 793F	240 tons (short)	9,133
Truck #109	CAT 793F	240 tons (short)	7,667
Truck #110	CAT 793F	240 tons (short)	8,577
Truck #111	Komatsu 830E-AC	240 tons (short)	1,975
Truck #112	Komatsu 830E-AC	240 tons (short)	2,472
Truck #113	Komatsu 830E-AC	240 tons (short)	2,140
Truck #114	Komatsu 830E-AC	240 tons (short)	2,077
Truck #115	Komatsu 830E-AC	240 tons (short)	2,227
Truck #180	CAT 777F	100 tons	16,494
Truck #181	CAT 777D	100 tons	29,866
Truck #182	CAT 777D	100 tons	29,567
Truck #183	CAT 777D	Water/sand truck	22,856
Truck #184	CAT 785B	Fardier	88,716
Drill #52	BUCYRUS 49HR	-	10,602
Drill #53	BUCYRUS 49HR	-	6,550
Dozer #201	CAT D10T	-	14,064
Dozer #202	CAT D10T	-	13,350
Dozer #203	CAT D9T	-	16,352
Dozer #204	CAT 854K	-	5,439
Grader #251	CAT 16M	-	13,073
Grader #252	CAT 16M	-	12,886

14.3.7 Life of Mine Mining Fleet

Table 14.3.7.1 shows the life of mine estimated mining equipment for the 2012 Operational Base Case (2200 t/h) Production Schedule.

Table 14.3.7.2 shows the life of mine estimated mining equipment for the 2012 Operational Base Case (2400 t/h) Production Schedule.

Table 14.3.7.1: Life of Mine Bloom Lake Estimated Mining Equipment: 2200 t/h Schedule

Year	Drill Large	Drill Small	Loader 18yd	Loader 22yd	Shovel 28 yd	Shovel 33yd	Electric Shovel 40yd	Truck 240 st	Truck 300 st	Dozer Large	Dozer Wheel	Dozer Small	Grader
2013	4	0	2	2	1	2	0	15	0	2	2	0	3
2014	3	2	2	0	1	3	0	15	16	2	2	1	3
2015	4	2	2	1	1	3	1	15	17	2	2	2	4
2016	5	2	1	2	1	3	2	15	23	3	2	2	4
2017	5	2	1	2	1	3	2	15	27	3	3	2	4
2018	5	2	1	2	1	3	2	15	26	3	3	2	4
2019	5	2	1	2	1	3	2	15	25	3	3	2	4
2020	5	2	1	2	1	3	2	15	27	3	3	2	4
2021	5	2	1	2	1	3	2	15	26	3	3	2	4
2022	5	2	1	2	1	3	2	15	27	3	3	2	4
2023	5	2	1	2	1	3	2	15	28	3	3	2	4
2024	5	2	1	2	1	3	2	5	38	3	3	2	4
2025	5	2	2	0	1	3	2	5	39	3	3	2	4
2026	5	2	2	0	1	0	4	5	38	3	3	2	4
2027	5	2	2	0	1	0	4	5	37	3	3	2	4
2028	5	2	2	0	1	0	4	5	40	3	3	2	4
2029	5	2	2	0	1	0	4	5	39	3	3	2	4
2030	5	2	2	0	1	0	4	5	38	3	3	2	4
2031	5	2	2	0	1	0	4	5	43	3	3	2	4
2032	5	2	2	0	1	0	4	5	45	3	4	2	4
2033	5	2	1	0	1	0	5	5	40	3	4	2	4
2034	5	2	1	0	1	0	5	5	42	3	3	2	4
2035	5	2	2	0	1	0	4	5	45	3	3	2	4
2036	5	2	2	0	1	0	4	5	48	2	3	1	4
2037	4	2	2	0	1	0	3	5	40	2	2	1	3
2038	4	2	2	0	1	0	2	5	37	2	2	1	2
2039	4	2	0	0	1	0	2	5	34	1	2	1	1
2040	3	2	0	0	1	0	1	5	27	1	1	1	1
2041	3	2	0	0	1	0	1	5	23	1	1	1	1
2042	2	2	0	0	1	0	1	5	13	1	1	1	1

- Required equipment calculation was used to estimate yearly equipment needs.
- SRK did not optimize the fleet purchase and end of life rebuilds. This estimate was calculated using the current fleet, 5 year previous mine plan budget and SRK filled in the missing equipment requirement. SRK did not calculate any of the smaller mining equipment. The missing smaller mining equipment capital is accounted in the sustaining line item in the cash flow.

Table 14.3.7.2: Life of Mine Bloom Lake Estimated Mining Equipment – 2400 t/h Schedule

Year	Drill Large	Drill Small	Loader 18yd	Loader 22yd	Shovel 28 yd	Shovel 33yd	Electric Shovel 40yd	Truck 240 st	Truck 300 st	Dozer Large	Dozer Wheel	Dozer Small	Grader
2013	3	2	2	0	1	1	0	15	16	2	2	1	3
2014	4	2	2	1	1	1	1	15	17	2	2	1	3
2015	5	2	1	2	1	1	2	15	23	3	2	2	4
2016	5	2	1	3	1	1	2	15	29	3	2	2	4
2017	5	2	1	2	1	1	2	15	29	3	4	2	4
2018	5	2	1	2	1	1	2	15	29	3	4	2	4
2019	5	2	1	2	1	1	2	15	30	3	4	2	4
2020	5	2	1	2	1	1	2	15	32	3	4	2	4
2021	5	2	1	2	1	1	2	15	30	3	4	2	4
2022	5	2	1	2	1	1	2	15	33	3	4	2	4
2023	5	2	1	2	1	1	2	15	45	3	4	2	4
2024	5	2	2	0	1	1	3	5	49	3	4	2	4
2025	5	2	2	0	1	1	4	5	43	3	4	2	3
2026	5	2	2	0	1	1	4	5	44	3	4	2	3
2027	5	2	2	0	1	1	4	5	47	3	4	2	3
2028	5	2	2	0	1	1	4	5	46	3	4	2	3
2029	5	2	2	0	1	1	4	5	47	3	4	2	3
2030	5	2	2	0	1	1	4	5	51	3	4	2	3
2031	5	2	2	0	1	1	5	5	47	3	4	2	3
2032	5	4	2	0	2	1	5	5	50	3	4	2	4
2033	4	4	2	0	2	1	3	5	41	2	3	1	3
2034	4	2	2	0	1	1	3	5	44	2	3	1	3
2035	4	2	2	0	1	1	2	5	35	2	2	1	2
2036	4	2	2	0	1	1	2	5	36	2	2	1	2
2037	3	2	2	0	1	1	2	5	33	2	2	1	2
2038	3	2	0	0	1	1	1	5	27	1	2	1	1
2039	1	1	0	0	1	1	1	1	10	1	1	1	1

- Required equipment calculation was used to estimate yearly equipment needs.
- SRK did not optimize the fleet purchase and end of life rebuilds. This estimate was calculated using the current fleet, 5 year previous mine plan budget and SRK filled in the missing equipment requirement. SRK did not calculate any of the smaller mining equipment. The missing smaller mining equipment capital is accounted in the sustaining line item in the cash flow.

To achieve a lower mining operating cost, the following mining capital expenditure will need to be allocated:

Table 14.3.7.3: LoM Estimated Mining Equipment Capital – 2200 t/h and 2400 t/h Schedules

Year	Mining Capital (US$000's)	
	2,200 t/h	2400 t/h
2013	1,500*	56,337*
2014	64,600*	69,942*
2015	66,700*	45,932*
2016	13,700*	19,315*
2017	700*	35,323*
2018	16,738	22,643
2019	14,408	26,961
2020	7,126	7,880
2021	17,739	39,109
2022	110,994	106,531
2023	15,593	53,108
2024	56,508	37,271
2025	204,694	233,399
2026	38,336	23,203
2027	4,629	20,382
2028	20,326	9,631
2029	15,280	16,093
2030	77,790	105,638
2031	47,020	24,582
2032	3,263	20,197
2033	2,461	10,931
2034	72,901	5,884
2035	39,813	34,314
2036	3,798	4,879
2037	6,139	4,079
2038	3090	n/a
2039	203	n/a
2040	5875.2	n/a
2041	0	n/a
Total	**$931,924**	**1,033,563**

*SRK used Cliffs 5 year budget for mine equipment. SRK recalculated the equipment requirements after 5 years.
*The first five years mine plan has the flexibility of using the estimated 2200 t/h mine capital for the 2400 t/h mine plan. The difference between the 2200 t/h and 2400 t/h five year mine capital is US$17 million.

14.3.8 Mining Operating Costs

Table 14.3.8.1 shows the historical and estimated future mining operating cost (includes all contractor mining assistance).

Table 14.3.8.1: Bloom Lake Historical and Future Estimated Mining Costs – 2200 t/h and 2400 t/h Schedules

Year	Mine Cost US$ (All Material Movement)	
	2,200 t/h	2,400 t/h
2010	3.48	3.48
2011	4.65	4.65
2012 (January to September)	4.10	4.10
2013	2.90	2.89
2014	2.72	2.70
2015	2.55	2.53
2016	2.65	2.59
2017	2.66	2.58
2018	2.62	2.60
2019	2.73	2.67
2020	2.70	2.71
2021	2.75	2.65
2022	2.81	2.76
2023	2.88	2.95
2024	2.92	3.05
2025	2.81	2.79
2026	2.67	2.71
2027	2.82	2.85
2028	2.79	2.85
2029	2.77	2.88
2030	2.96	3.03
2031	3.04	2.90
2032	2.88	3.07
2033	2.99	3.21
2034	3.08	3.31
2035	3.18	3.36
2036	2.62	2.70
2037	2.69	2.75
2038	2.64	n/a
2039	2.59	n/a
2040	2.56	n/a
2041	2.78	n/a

2010 – 2012 (Actual costs)
2013 – 2041 (Calculated costs based on new mine plans and reserves, assuming that less contractors will be used and mining capital expenditure is allocated)

Table 14.3.8.2 shows the unit cost breakdown for mining operating costs for both cases (2,200 t/h and 2,400 t/h).

Table 14.3.8.2: Bloom Lake Estimated Life of Mine Unit Cost Breakdown – 2200 t/h and 2400 t/h

Description	Units	US$	
		2,200 t/h	2,400 t/h
Drilling	$/t	0.16	0.16
Blasting	$/t	0.49	0.49
Loading	$/t	0.19	0.20
Hauling	$/t	1.63	1.64
Roads & Dumps	$/t	0.16	0.17
Ancillary	$/t	0.11	0.11
Total		**2.74**	**2.77**

SRK: Lower mining costs are expected due to the replacement of large electric shovels and larger 300 t trucks.

15 Recovery Methods

15.1 Processing Methods

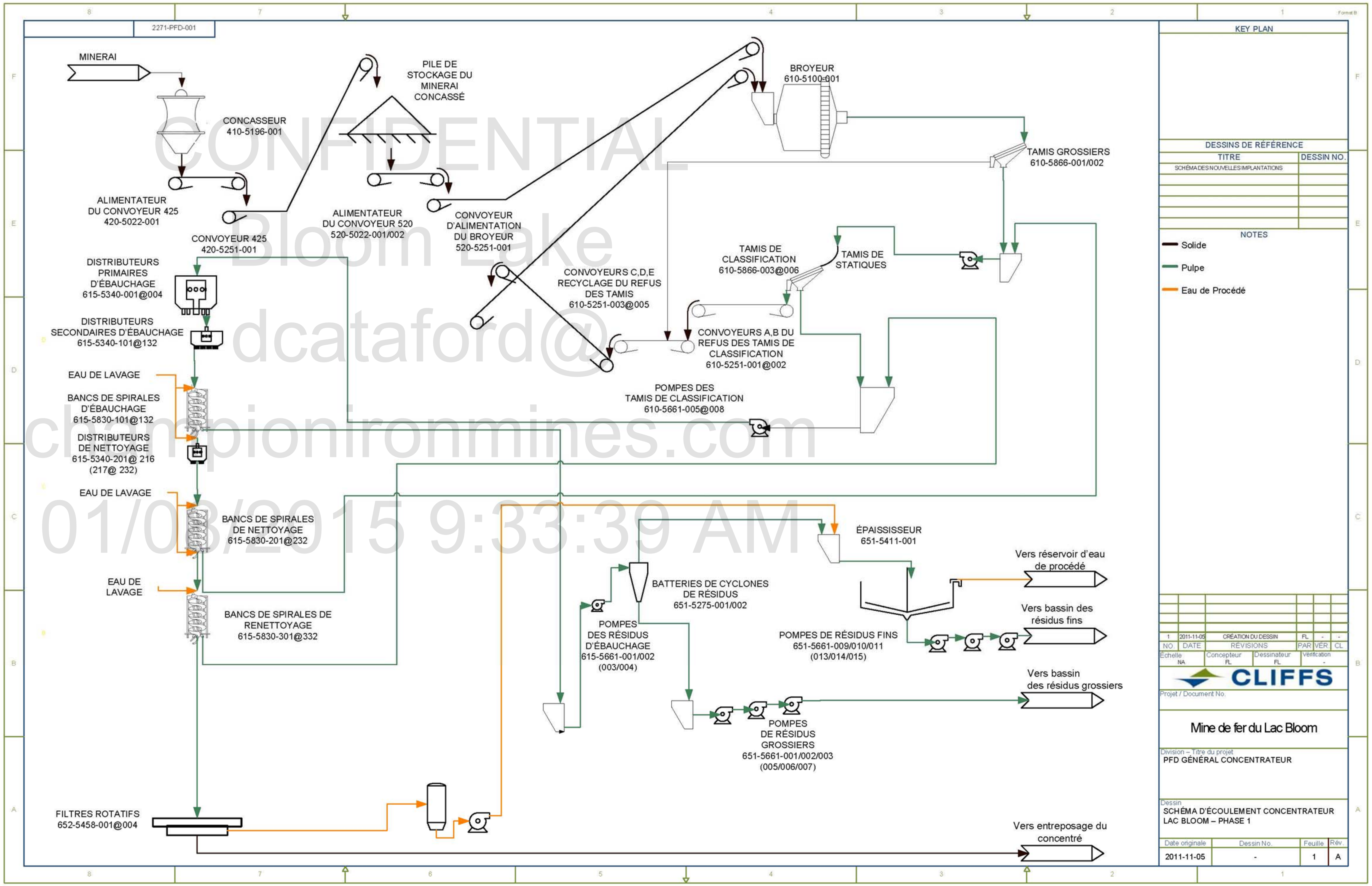
The Phase I concentrator was commissioned during March, 2010. The current Bloom Lake process flowsheet is shown in Figure 15.1.1 and the major design criteria are shown in Table 15.1.1. Ore from the mine is delivered by truck to the primary gyratory crusher and crushed to -8 inch. Crushed ore is then conveyed to a 26,000 t (live capacity) crushed ore stockpile. Crushed ore is reclaimed through a coarse ore stockpile with two 1.8 m x 7.0 m apron feeders that discharge onto a conveyor that feeds the 11 m diameter x 6.1 m long autogenous grinding (AG) mill driven by dual 5,590 kW motors. The AG mill discharges onto two pebble screens with the screen oversize being recirculated back the AG mill. Screen undersize is pumped to four banana-style vibrating screens to produce an 850 micron (20 mesh) separation. It should be noted that the original design criteria called for a 425 micron (35 mesh) separation, but the installed screens do not have the capacity to make the 425 micron separation and a 850 micron screen decks were installed after commencement of operation to prevent the screens from flooding. Bloom Lake staff has stated that the iron ores currently being processed appear to be well liberated at the 850 micron grind size, and plan to continue operating at this grind size.

Table 15.1.1: Major Process Design Criteria - Phase I and Phase II Concentrator

Item	Units	Criteria	
		Phase 1	Phase II
Design Feed Rate	t/h	2,372	2,614
Operating days	day	365	365
Operating Availability	%	93	93
Fe Recovery	%	84	83
Concentrate Production	t/y	8,000,000	8,000,000
Concentrate Grade			
Fe	%	66.2	66
SiO_2	%	4.5	4.5
Grind	microns	410	n/a
Grind	mesh	35	n/a
Classification Screen	microns	n/a	600
Effective Screen U'size	microns	n/a	425

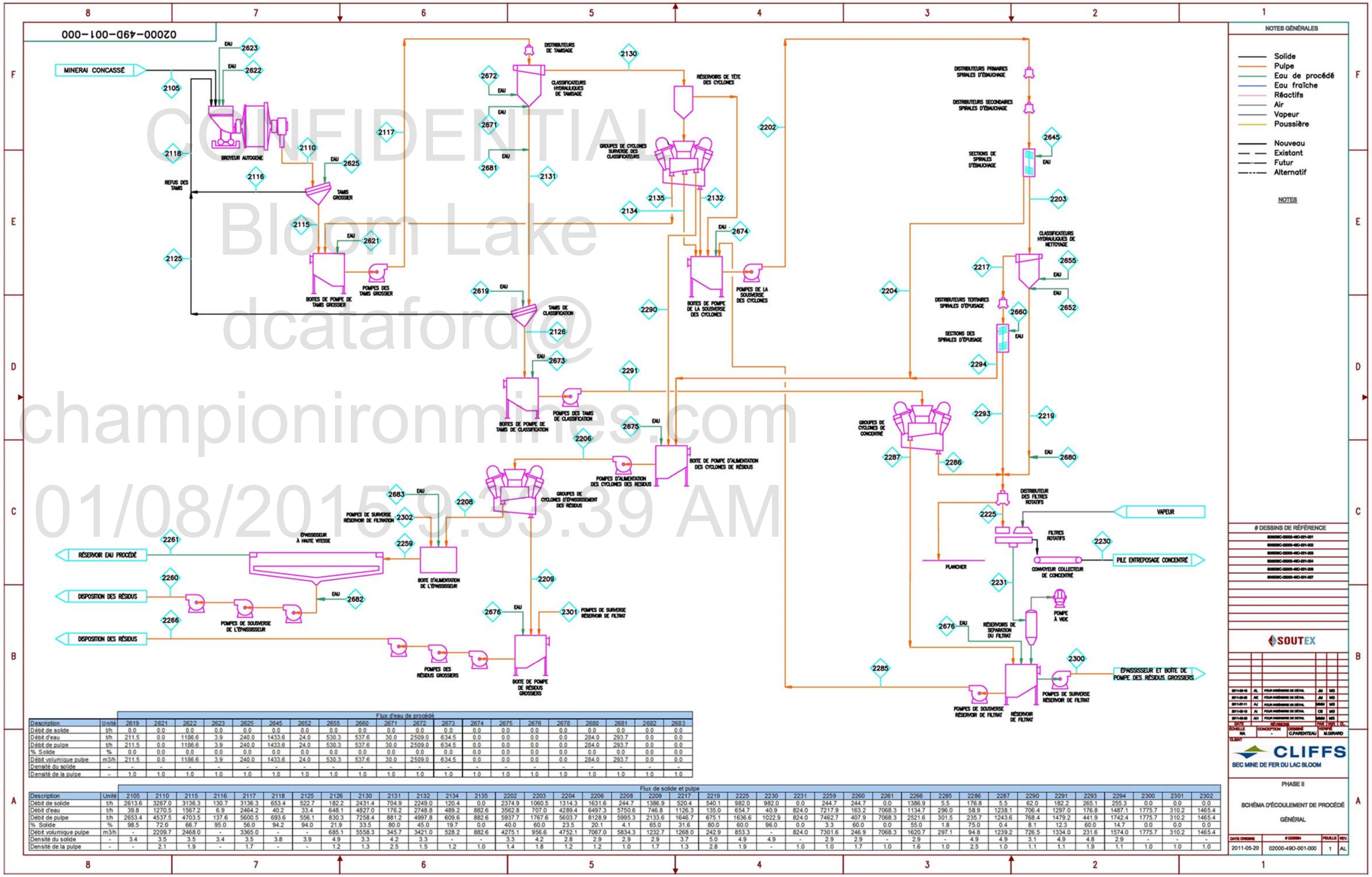
Screen undersize from the grinding circuit is advanced to the rougher gravity concentration circuit, which consists of 32 banks of rougher spirals arranged in four lines. Each bank has two rows of eight back to-back double-start spirals. Concentrate from each bank of rougher spirals is combined and distributed by a 24-way rubber line cleaner spiral distributor to feed a bank of 12 double-start cleaner spirals. Concentrate from each cleaner spiral feeds directly to a recleaner spiral for final upgrading. The rougher, cleaner and re-cleaner have five turns. In total, there are 1,024 rougher spirals, 768 cleaner spirals and 768 recleaner spirals. Concentrate from the recleaner spirals is dewatered on four horizontal pan filters fitted with steam hoods to dry the concentrate to about 2.5% moisture during the winter months and 4.5% during the summer months. Filtered concentrate is discharged to a 42 inch wide belt conveyor and transported to a 24,000 t capacity silo at the train load-out station. The silo is designed to hold 24 hours of production and load-out requirement is for one 240-car (100 t capacity cars) train per day.

The Phase II concentrator plant is currently under construction and scheduled for completion during 2014. The Phase II concentrator flowsheet is shown in Figure 15.1.2, and will incorporate improvements in equipment selection and sizing based on pilot plant testing and experience gained from the operation of the Phase I concentrator. The main difference between the two concentrators is the incorporation of hydrosizers in the Phase II concentrator flowsheet, which will be used, in conjunction with spiral concentrators, to produce an iron gravity concentrate. In the Phase II concentrator the AG mill discharge will be screened and the screen undersize will be advanced to a primary classification hydroclassifier circuit. The underflow from the classification hydrosizers will be screened at 600 microns, with the screen oversize being returned to the AG mill and the screen undersize being further dewatered with hydrocyclones. The hydrocyclone underflow will be final iron concentrate that will then be advanced to the filtration circuit. The classification hydrosizer circuit overflow will also be dewatered by hydrocyclones with the underflow advancing to the rougher spiral concentrators and the overflow being recirculated. The tailing from the rougher spiral concentrators will be final tailing and the rougher spiral concentrate will be further classified and concentrated in a cleaner stage of hydrosizers. The underflow from the second stage hydrosizers will also be final concentrate and advanced to filtration. The cleaner hydrosizer overflow will be sent to the scavenger spiral concentrators, with the concentrate advancing to the final concentrate filters and the tailing being discarded to tailings. Incorporation of hydrosizers into a gravity concentrating circuit will serve to improve the concentrating efficiency of the spiral concentrators by presenting a properly sized feed material to the spirals, and can be expected to result in improved iron recoveries and concentrate grades.



Source: Cliffs

Figure 15.1.1: Bloom Lake Phase I Process Plant Flowsheet



Flux d'eau de procédé																					
Description	Unité	2619	2621	2622	2623	2625	2645	2652	2655	2660	2671	2672	2673	2674	2675	2676	2678	2680	2681	2682	2683
Débit de solide	t/h	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Débit d'eau	t/h	211.5	0.0	1186.6	3.9	240.0	1433.6	24.0	530.3	537.6	30.0	2509.0	634.5	0.0	0.0	0.0	0.0	284.0	293.7	0.0	0.0
Débit de pulpe	t/h	211.5	0.0	1186.6	3.9	240.0	1433.6	24.0	530.3	537.6	30.0	2509.0	634.5	0.0	0.0	0.0	0.0	284.0	293.7	0.0	0.0
% Solide	%	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Débit volumique pulpe	m3/h	211.5	0.0	1186.6	3.9	240.0	1433.6	24.0	530.3	537.6	30.0	2509.0	634.5	0.0	0.0	0.0	0.0	284.0	293.7	0.0	0.0
Densité du solide	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Densité de la pulpe	-	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0

Flux de solide et pulpe																				
Description	Unité	2105	2110	2115	2116	2117	2118	2125	2126	2130	2131	2132	2134	2135	2202	2203	2204	2206	2208	2209
Débit de solide	t/h	2613.6	3267.0	3136.3	130.7	3136.3	653.4	522.7	182.2	2431.4	704.9	2249.0	120.4	0.0	2374.9	1060.5	1314.3	1631.6	244.7	1386.9
Débit d'eau	t/h	39.8	1270.5	1567.2	6.9	2464.2	40.2	33.4	648.1	4827.0	176.2	2748.8	489.2	882.6	3562.8	707.0	4289.4	6497.3	5750.6	746.8
Débit de pulpe	t/h	2653.4	4537.5	4703.5	137.6	5600.5	693.6	556.1	830.3	7258.4	881.2	4997.8	609.6	882.6	5937.7	1767.6	5603.7	8128.9	5995.3	2133.6
% Solide	%	98.5	72.0	66.7	95.0	56.0	94.2	94.0	21.9	33.5	80.0	45.0	19.7	0.0	40.0	60.0	23.5	20.1	4.1	65.0
Débit volumique pulpe	m3/h	-	2209.7	2468.0	-	3365.0	-	-	685.1	5558.3	345.7	3421.0	528.2	882.6	4275.1	956.6	4752.1	7067.0	5834.3	1232.7
Densité du solide	-	3.4	3.5	3.5	3.4	3.5	3.8	3.9	4.9	3.3	4.2	3.3	-	-	3.3	4.2	2.8	2.9	2.9	2.9
Densité de la pulpe	-	-	2.1	1.9	-	1.7	-	-	1.2	1.3	2.5	1.5	1.2	1.0	1.4	1.8	1.2	1.2	1.0	1.7

Description	Unité	2217	2219	2225	2230	2231	2259	2260	2261	2266	2285	2286	2287	2290	2291	2293	2294	2300	2301	2302
Débit de solide	t/h	520.4	540.1	982.0	982.0	0.0	244.7	244.7	0.0	1386.9	5.5	176.8	5.5	62.0	182.2	265.1	255.3	0.0	0.0	0.0
Débit d'eau	t/h	1126.3	135.0	654.7	40.9	824.0	7217.9	163.2	7068.3	1134.7	296.0	58.9	1238.1	706.4	1297.0	176.8	1487.1	1775.7	310.2	1465.4
Débit de pulpe	t/h	1646.7	675.1	1636.6	1022.9	824.0	7462.7	407.9	7068.3	2521.6	301.5	235.7	1243.6	768.4	1479.2	441.9	1742.4	1775.7	310.2	1465.4
% Solide	%	31.6	80.0	60.0	96.0	0.0	3.3	60.0	0.0	55.0	1.8	75.0	0.4	8.1	12.3	60.0	14.7	0.0	0.0	0.0
Débit volumique pulpe	m3/h	1268.0	242.9	853.3	-	824.0	7301.6	246.9	7068.3	1620.7	297.1	94.8	1239.2	726.5	1334.0	231.6	1574.0	1775.7	310.2	1465.4
Densité du solide	-	3.7	5.0	4.9	4.9	-	2.9	2.9	-	2.9	-	4.9	4.9	3.1	4.9	4.8	2.9	-	-	-
Densité de la pulpe	-	1.3	2.8	1.9	-	1.0	1.0	1.7	1.0	1.6	1.0	2.5	1.0	1.1	1.1	1.9	1.1	1.0	1.0	1.0

Figure 15.1.2: Bloom Lake Phase II Process Plant Flowsheet

Tailings Disposal

Approximately 230 Mt of mill tailings will be produced over 20 years of operation and will be stored in a 741 ha tailing storage facility (TSF) located north of the pit and west of the plant site. The spiral rougher tailing is the final plant tailing and is pumped to cycloning station near the TSF.

The cyclone underflow is discharged to a drainage basin where is allowed to drain. After draining to a workable density it is loaded into haul trucks and hauled to the TSF or embankment construction areas. The cyclone overflow is pumped directly to the TSF. It is SRK's understanding that the current tailing disposal procedures are temporary and that as part of the Phase II expansion, plant tailings will be cycloned at the plant with cyclone underflow being pumped directly to the tailing pond and the cyclone overflow being thickened and then pumped to the tailing pond. This will eliminate the expensive tailing rehandling operation that is currently being practiced.

15.2 Phase I Concentrator Performance

Phase I concentrator 2012 performance statistics is summarized in Table 15.2.1. As can be seen, the Phase I concentrator has not yet achieved design production rates or iron recovery projections. During 2012 the plant has averaged 2,033 t/h versus a design production rate of 2,372 t/h. Iron recovery has averaged 72% and concentrate weight percent recovery has averaged 34.7% into final iron concentrates that averaged 66.2% Fe and 4.6% SiO_2.

Table 15.2.1: Phase I Concentrator Performance - 2012

	January	February	March	April	May	June	July	August	September	October	November	December	Year to Date
Concentrate													
Weight Recovery, %	34.0	36.7	35.4	35.3	34.9	36.0	33.0	34.5	32.6	33.0	35.9	35.7	34.7
Moisture Content, %	2.8	2.5	2.4	2.7	3.8	4.1	3.9	3.7	4.0	3.8	3.7	3.3	3.4
Fe %	66.2	66.3	66.4	66.4	66.3	66.4	66.3	66.1	66.1	66.0	66.2	66.2	66.2
Silica, %	5.0	5.0	4.8	4.7	4.3	4.2	4.4	4.5	4.5	4.4	4.4	4.5	4.6
Fe Recovery, %	73.1	76.7	74.8	74.0	72.3	72.5	67.8	68.9	67.5	69.7	75.2	72.1	72.0
Concentrate Rate (tons/hour)	695.6	663.4	753.0	703.9	631.3	735.1	713.1	743.9	677.6	752.9	734.0	850.0	721.1
- 150 µm in Concentrate	30.4	29.0	28.7	30.6	33.9	32.8	34.7	35.5	34.9	29.4	28.2	28.9	31.4
Magnetite %	7.1	4.5	9.2	6.2	7.7	8.1	14.1	14.6	14.5	17.4	14.0	9.8	10.6
Tailings													
Coarse Tailings Fe %	11.5	10.9	11.5	11.9	13.1	13.6	15.0	15.0	14.8	12.9	11.5	13.3	12.9
Fine Tailings, Fe %	21.5	18.4	19.1	19.0	20.4	19.7	20.8	20.3	20.8	18.6	18.6	20.1	19.8
Feed													
tons/hour	2,026	1,807	2,104	1,967	1,768	1,956	2,082	2,109	2,015	2,192	2,021	2,352	2,033
Fe % (Calc)	30.7	32.7	31.5	31.6	32.0	32.9	32.3	30.8	30.2	31.2	31.1	32.6	31.6
Fe% (Sampled)	30.8	31.7	31.5	31.6	32.1	33.1	32.3	32.9	32.0	31.3	31.4	33.2	32.0
Mill Availability, %	97	57	99	84	98	72	97	90	97	94	72	99	88

Source: Cliffs

The process plant measures feed tonnage and iron concentrate tonnage with belt scales that are calibrated on a regular basis. In addition concentrate and final tailing (coarse and fine fractions) grades are sampled throughout each production shift. Feed grade to the plant is calculated on the basis of measured feed and concentrate tons and final tailing grades. Overall iron recovery is then calculated on the basis of the actual iron concentrate production and the calculated feed grade to the plant. It should be noted that Bloom Lake also obtains plant head sample from the scalping screen undersize, but does not use this value in recovery calculations. Table 15.2.1 shows the average iron grades for both the calculated and sampled plant feed. Sampled feed grades tend to be slightly lower than the calculated feed grades.

SRK would make the following observations regarding the performance of the Phase I concentrator:

- Plant throughput has averaged 2,033 t/h which is significantly less than the design capacity of about 2,372 t/h. Based on the Technical Report issued by BBA, November 2008, it appears that the AG mill may have been designed on the basis of a gross power requirement of 3.6 kWh/t with a 20 mesh classification screen (4.2 kWh/t with a 35 mesh classification screen). During 2012, the Phase I concentrator reported gross AG mill power ranging from about 3.8 – 7.9 kWh/t with an overall average of 4.8 kWh/t;
- The samples used to for the comminution studies reported by BBA in 2008 were from mini-bulk composites 10 and 11 from surface outcrop samples from the west zone of the ore deposit. Some additional SMC (SAG mill comminution) testing appears to have been done on mini-bulk composites 4 and 8, also from surface samples. It is not clear, however, what area of the deposit these composites represented. It should be noted that there is the potential of the ore being harder in the eastern zones of the ore deposit due to the higher levels of magnetite;
- In order to obtain a better understanding of the variability of the ore hardness, SRK would recommend that additional comminution testing be done on selected drill core from throughout the deposit. This will enable a better understanding of throughput capacity achievable with the existing AG mill;
- The Bloom Lake operating staff has experimented with AG mill discharge grates with both ½ inch and ¾ inch openings. At the present time ¾ inch grates are installed, and the circulating load of the pebbles back to the AG mill is reported to be in the range of 35-40%. In order to improve the throughput capacity of the present AG mill it may be necessary implement a secondary comminution circuit for this recycle streams. Options that should be evaluated include:
    - o Installation of a pebble crusher;
    - o Installation of an HPGR;
    - o Installation of a secondary ball mill; and
    - o AG to SAG mill conversion.

Iron recovery has averaged about 72% which is about 12.5% below the original iron recovery target. It is unlikely that iron recovery can be significantly improved without major equipment modifications in the Phase I concentrator. In order to predict iron recovery from the Phase I concentrator, SRK has applied a 12.5% iron recovery discount to the iron recovery equation that was originally used for Phase I.

The adjustment methodology is presented in Table 15.2.2 and is based on the following:

- Predicted Wt% recovery = 0 .3788 x Feed Fe % - 3.1746 – (0.194 x MgO% x100/Conc Fe%
- Predicted Fe Recovery = (Wt% recovery x Conc Fe %)/Feed Fe %
- Adjusted Fe Recovery = Predicted Fe Recovery – 12.5%

Table 15.2.2: Adjusted Iron Recovery for the Phase I Concentrator

	January	February	March	April	May	June	July	August	September	October	November	December	Year to Date
Feed Grade, Fe %	30.7	32.7	31.5	31.6	32.0	32.9	32.3	30.8	30.2	31.2	31.1	32.6	31.6
Concentrate Grade, Fe%	66.2	66.3	66.4	66.4	66.3	66.4	66.3	66.1	66.1	66.0	66.2	66.2	66.2
Predicted Weight % Recovery (1), (2)	39.1	41.9	40.3	40.4	40.9	42.2	41.4	39.3	38.5	39.8	39.7	41.8	40.4
Predicted Fe Recovery (3)	84.4	84.9	84.9	84.9	84.8	85.1	84.9	84.3	84.1	84.3	84.5	84.8	84.7
Adjusted Recovery (4)	71.9	72.4	72.4	72.4	72.3	72.6	72.4	71.8	71.6	71.8	72.0	72.3	72.2

(1) Based on 2005 Recovery Equation: Weight % Recovery = 1.3788 * Feed Fe% - 3.1746 - (0.194 *MgO % * 100/Conc Fe %)
(2) MgO % is fixed at 2.1%
(3) Predicted Recovery Equation: Fe Recovery = (Weight % Recovery * Conc Fe %)/Feed Fe %
(4) Adjusted Fe Recovery: Predicted Fe Recovery - 12.5

15.3 Phase II Concentrator Projected Performance

The Phase II concentrator has been designed with the same size AG mill that was installed in the Phase I concentrator. However, the Phase II concentrator is projected to process 2,600 t/h. Given that the Phase II concentrator will be processing the same ore as the Phase I concentrator, it is very unlikely that a throughput capacity of 2,600 t/h can be achieved. SRK would make the following additional observations regarding projected performance of the Phase II concentrator:

- Phase II concentrator throughput may be incrementally better than the Phase I concentrator due to:
  - Provision for better blending of the ore prior being fed to the AG mill; and
  - Installation of hydrosizers in combination with spiral concentrators, which may serve to off-load of portion of the ore at a coarser size split.
- Based on the observations presented above, Phase II concentrator throughput capacity may likely be in the range of 2,200 to 2,400 t/h;
- As with the Phase I concentrator, the installation of a secondary comminution circuit may ultimately be necessary to achieve targeted throughput rates. This should be evaluated after the Phase II concentrator has been commissioned, de-bottlenecked and actual throughput capacity confirmed; and
- Projected iron recovery at the Phase II concentrator continues to be based on the same recovery equation that that was used to project iron recovery at the Phase I concentrator. However, significant additional pilot testing was undertaken to design an improved gravity concentration circuit that includes significantly different equipment than was installed in the Phase I concentrator. The Phase II concentrator circuit is based on a combination of hydrosizers and spiral concentrators, which SRK believes will significantly improve the efficiency of gravity concentration circuit. In addition, more efficient spiral concentrators were subjected to pilot plant testing and have been installed in the Phase II concentrator. The Phase I concentrator does not have any hydrosizers and relies entirely on less efficient spiral concentrators.

SRK is of the opinion that iron recovery in the Phase II concentrator will be significantly improved, but most likely not to the level projected in the recovery equations used for the project. It is SRK's opinion that iron recovery into acceptable grade concentrates will be in the range of 78-80%. In order to predict iron recovery from the Phase II concentrator, SRK has applied a 5% iron recovery discount to the iron recovery equation that was originally used for both the Phase I and Phase II concentrators. The adjustment methodology is presented in Table 15.3.1 and is based on the following:

- Predicted Wt% recovery = 0 .3788 x Feed Fe % - 3.1746 – (0.194 x MgO% x100/Conc Fe%;
- Predicted Fe Recovery = (Wt% recovery x Conc Fe %)/Feed Fe %; and
- Adjusted Fe Recovery = Predicted Fe Recovery – 5%.

Table 15.3.1: Adjusted Iron Recovery for the Phase II Concentrator

Unit	Parameter
Feed Grade, Fe %	30.0
Concentrate Grade, Fe%	66.0
Predicted Weight % Recovery (1), (2)	38.2
Predicted Fe Recovery (3)	84.0
Adjusted Recovery (4)	79.0

(1) Based on 2005 Recovery Equation: Weight % Recovery = 1.3788 * Feed Fe% - 3.1746 - (0.194 *MgO % * 100/Conc Fe %)
(2) MgO % is fixed at 2.1%
(3) Predicted Recovery Equation: Fe Recovery = (Weight % Recovery * Conc Fe %)/Feed Fe %
(4) Adjusted Fe Recovery: Predicted Fe Recovery - 5

15.4 Operating Cost

Phase I concentrator operating costs for 2012 (January to September) are summarized in Table 15.4.1. The average cost per ton of concentrate is reported at US$18.94, which is equivalent to US$6.20/t of plant feed. It is likely that Phase II concentrator operating costs will be similar.

Table 15.4.1: Summary of Bloom Lake Process Plant Operating Costs (January - December, 2012)

Unit	2012 Total	January	February	March	April	May	June	July	August	September	October	November	December	Ore Milled (US$/t)	Concentrate (US$/t)
Administration	3,911,370	26,530	347,418	167,149	404,997	203,550	994,344	588,902	231,329	335,867	260,238	128,005	223,042	0.25	0.72
Crusher	9,225,187	488,477	174,201	897,439	472,072	1,874,662	596,036	1,210,568	665,774	220,897	1,028,396	1,266,935	329,730	0.58	1.69
Surge Pile	2,769,451	553,859	636,400	45,634	400,441	285,284	145,805	318,803	(340,677)	144,014	261,113	146,922	171,854	0.17	0.51
Concentrator	25,810,781	2,135,175	1,536,076	1,291,926	1,695,684	2,890,621	3,017,728	1,529,740	1,330,017	2,185,871	2,909,264	2,846,066	2,442,612	1.63	4.74
Drying	4,267,986	284,104	304,506	300,871	252,117	1,472,841	61,963	105,053	261,467	332,894	205,917	494,951	191,302	0.27	0.78
Load out	4,225,275	486,557	312,487	1,340,417	479,743	258,261	162,102	508,903	(267,568)	202,969	97,991	391,555	251,859	0.27	0.78
General Maintenance	22,012,943	1,536,930	2,098,377	3,388,721	4,034,878	906,759	1,204,253	1,629,867	1,243,231	929,532	1,957,702	2,043,649	1,039,044	1.39	4.04
Auxiliary Services	9,709,186	1,493,593	1,105,340	1,433,620	1,094,730	547,247	64,111	191,136	121,697	237,460	572,453	1,203,278	1,644,522	0.61	1.78
Laboratory	3,609,434	346,671	273,850	281,428	247,646	293,011	193,492	255,790	327,393	230,087	222,019	335,953	602,096	0.23	0.66
Shared Services	2,290,411	1,033,905	300,106	302,106	163,152	32,372	22,496	(64,735)	2,983	105,683	147,082	153,501	91,760	0.14	0.42
Waste Treatment	7,754,411	44,903	696,931	(647,100)	784,977	998,433	582,988	549,628	1,450,462	1,486,078	333,165	100,248	1,373,697	0.49	1.42
Water Management - Mill	904,025	-	-	-	-	-	-	-	-	107,028	105,802	170,407	520,788	0.06	0.17
Steam & Drainage	371,998	(3,170)	11,253	3,025	902	36,731	100,336	2,585	177,961	11,601	1,075	-	29,700	0.02	0.07
Total Mill	**96,862,458**	**8,427,534**	**7,796,944**	**8,805,236**	**10,031,340**	**9,799,772**	**7,145,655**	**6,826,239**	**5,204,069**	**6,529,978**	**8,102,216**	**9,281,468**	**8,912,005**	**6.12**	**17.77**

Source: Cliffs/Bloom Lake

16 Project Infrastructure

Following is a discussion of infrastructure and logistics requirements for the Project.

16.1 Site Access Road

The 6 km improved gravel access road to Bloom Lake is from Highway 389, between the Mont-Wright mine and the town of Fermont. Other roads service the concentrator, mine, crusher, and the tailings line to the freshwater pump houses and Bloom Lake. The access road is scheduled to be widened and guard rails added in the five year capital budget.

16.2 On-Site Services

Services are housed in a 2,485 m^2 (35 m x 71 m) structure which is attached to the Phase I concentrator building. The following services/functions are performed within this structure:

- Warehousing area for parts and supplies;
- Location of plant steam boiler; for heating and filter cake drying;
- Electrical/instrument repair shop;
- First-aid room;
- Administration office, including purchasing, human resources, technical services, training and plant operating personnel; the Phase I laboratory equipped for metallurgical testwork, lunchroom, men's and women's change rooms, sanitary and locker facilities; a communications room; compressor room;
- Fresh water storage tank and water treatment facilities, and
- Electrical room.

Additional service facilities are included in the expansion project to add support facilities for the expanded mine and mill capacity.

16.2.1 Shop & Warehouse

The warehouse floor area covers an area of 630 m^2 (21 m x 30 m) and is 9.5 m high. Trucks to be unloaded descend a ramp to bring the truck bed level with the loading dock and floor inside the warehouse. A fenced outdoor warehouse yard is also used to stock material.

The existing truck shop or garage has four bays. One bay has a high door for mine trucks but does not allow drive through exit for the large mine trucks as the opposite door is smaller and blocked by conex boxes. A set of bays with a smaller door exists, but it also does not have drive-through capability. A tool crib is installed in the shop with a welding area being established at the time of the site visit. A number of trailer and modular units are attached to the shop to provide office space for engineering and mine operations/maintenance supervision.

The PHII expansion will provide a larger facility with capability of supporting the larger trucks and the greater number of pieces of equipment. Additionally, the design is being modified to allow multiple high bay doors to allow more efficient movement of repaired vehicles.

16.2.2 Utilities Area

The 820 m^2 utilities area includes the boiler room, fresh water storage tank and water treatment, blower and compressor rooms and the emergency MCC room.

Two 50,000 lb/h capacity fire tube boilers supply low pressure steam to the concentrate filters and to the hot water heat exchangers for building heating.

16.2.3 Emergency Vehicle Station

The emergency vehicle station is sized for an ambulance and one fire truck. The first-aid station is also located in this area.

16.2.4 Offices, Dry and Lunch Room

An office space of 1,379 m^2 for administration, human resources, accounting, purchasing, engineering, plant operating and maintenance personnel is provided on the second floor of the service building. Washrooms and a first aid room are also located on this floor.

Change rooms, showers and toilets for men are located on the ground floor and on the first floor for women. The lunch room is on the first floor.

16.2.5 Laboratory

The Phase I laboratory is located on the ground floor and 266 m^2 of floor space for the preparation and analysis of samples by wet methods and XRF. The preparation area is equipped for splitting, drying, crushing, grinding, screening and filtering of samples from both the mine and the Phase I concentrator.

The laboratory facilities and equipment capacity will be expanded to meet the additional analysis requirements of the 16 Mt/y project. Given the distance between the two concentrators, an additional sample preparation lab will be added. These facilities will include a preparation area for weighing, filtering, drying, splitting and screening of samples the second concentrator only.

16.3 Electric Power

16.3.1 Primary Power Supply

Phase I installed capacity is 35 MW. Two 17 km power lines constructed by HQ provide the required power to the Project. The first line is 11.5 km in length and run from the HQ Normand station alongside Highway 389 to the mine power metering station at the junction of Highway 389 road and the plant access road. The second section, 6.5 km in length, runs alongside the mine access road to the Bloom 34.5 kV main station.

22.5 km of 34.5 kV aerial lines distribute power to various plant and mine loads.

The mine site is supplied at 7.2 kV via two portable substations 34.5 kV to 7.2 kV, 7.5 MVA located at each end of the pit.

The main 4.16 kV switchgear fed from the two 21/28 MVA transformers is located in the main electrical room of the service building and feeds various 4.16 kV motor control centers, 4.16 kV variable frequency drives (VFD) and 4.16 kV to 600 V substations through the plant. The AG Mill twin 7,500 HP motor are connected to the main 4.16 kV switchgear.

Motor loads of 300 HP and higher, using across-the-line starters, are supplied at 4.16 kV. Smaller motors and motors with a Variable Frequency Drive (VFD), with loads up to 1000 HP, are supplied from 600 V substations.

The 600/347 V lighting and building services use small isolating transformers, thus limiting the 600 V fault level to a safe level of less than 10 kA, which allows for the use of distribution of equipment currently used in large industrial complexes.

16.3.2 Phase II Expansion

Construction of a new 34.5 kV high voltage power line will be required for the new facilities. The installed capacity for Phase II will be an additional 35MW.

The Normand sub-station is presently fed a 315kV line with a tap at 161 kV. The 161 kV is at maximum capacity but the 315kV line has the required capacity to feed the new mill.

Additional power will originate from the 315 kV line coming from the Montana substation which passes at a distance of 1.9 km from the property. A 315 kV to 34.5 kV sub-station will be erected and the mill would be fed by a second 34.5 kV line originating from this new substation.

The erection of a new 1.9 km 315 kV line will be required by HQ.

16.3.3 Emergency Back-up System

The emergency power supply for the concentrator and service building is backed by a 1000 kW diesel generator set which is installed adjacent to the Service Building. The system will supply critical loads such as lighting and the steam plant during a power outage. A 600 kW diesel generator set will supply emergency power to the crusher building.

An additional 1000 kW stand-by diesel generator will provide emergency power to operate critical items of equipment for the Phase II concentrators in case of failure of the main power supply.

16.4 Water Supply

16.4.1 Fresh Water

Fresh water is used as make-up to the boilers and for domestic consumption. The average hourly consumption on an annual basis is approximately 6 m^3. Fresh water is supplied to the fresh water reservoir the Phase I concentrator by gravity flow from Bloom Lake. From there, distribution pumps pump water either to the domestic water circuit or to the steam boilers.

For the Phase II mill, the fresh water circuit will be extended by the addition of a new set of pumps.

16.4.2 Process Water

Process water is supplied to Phase I concentrator from an in-ground reservoir by three vertical turbine pumps (three in operation).

Process water will be supplied to Phase II concentrator from an above ground tank by three horizontal pumps (two in operation). As for Phase I plant, thickener overflow will be the principal supply of water to Phase II process water reservoir. Reclaim water will be a secondary source.

16.4.3 Reclaim Water

Reclaim water is pumped from the decanted water in the tailings retention area. There are two vertical turbine type reclaim pumps with 400 HP drives, one operating and one standby. Both pumps have a combined capacity of 1385 m^3/hr.

Reclaim water from the polishing pond located at the tailings disposal area will be pumped back to the Phase II concentrator by the addition of three new barge-mounted vertical turbine pumps (two operating, one stand-by). Part of the flow will be used for equipment cooling, flocculant dissolution and gland seals, the remainder will be discharged into Phase II process water reservoir.

16.4.4 Sanitary Treatment

The sewage disposal system accommodates the concentrator and service building and the truck shop. The sewage treatment system includes collecting and pumping stations, a septic tank and aerobic treatment stages.

16.4.5 Fire Water

The fire protection system includes fire water pumps, a fire water distribution network, fire water hose stations and water sprinkler systems.

Fire water pumps are located in a pump house at Bloom Lake. There are two main pumps and one jockey pump to maintain the pressure in the fire water pipe network. For Phase II, two additional vertical pumps will supply water from Confusion Lake to Phase II plant process water reservoir and industrial water circuit in the event that there is insufficient or no reclaim water coming from the polishing pond. In total, there are four vertical pumps to supply industrial water to Phase I and Phase II concentrators.

No additional fire pumps will be added for the Phase II expansion. There will a total of three horizontal turbine pumps (one electric, one diesel, one jockey) to provide fire water to the whole site. These pumps are installed at the same location as the industrial water pumps.

16.5 Tailings Disposal

The doubling of the production rate from the project expansion requires doubling the water usage for tailing handling. Since the increase in production will be using the same reserves, the amount of tailings produced will be the same. The tailings will be produced at a higher rate. Since the 2011 acquisition, Cliffs determined that the area permitted for tailings disposal was insufficient to meet the long term needs of the operation given the expanded reserve base. Cliffs has developed a long range tailings development plan that is able to meet the life of mine requirements.

16.5.1 Design Criteria

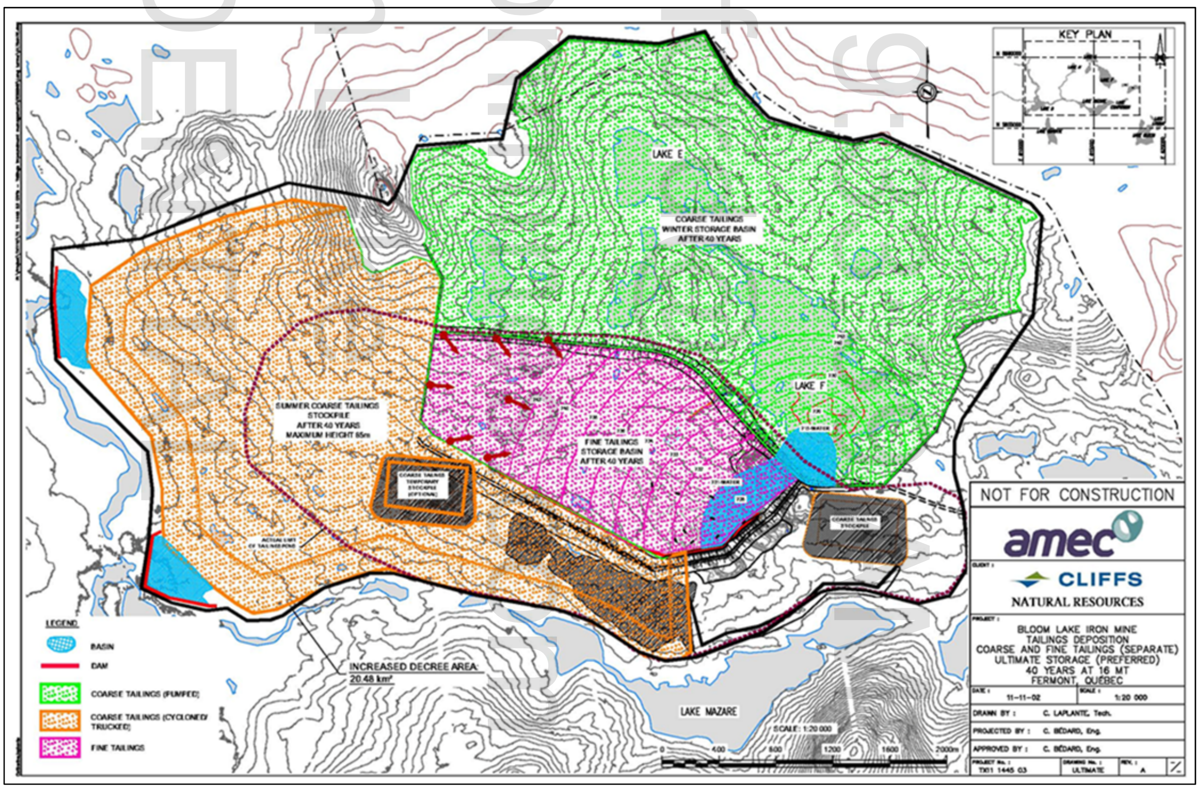
- The plant will reject two types of residue; fine residue (characterized as silt) which must be entirely confined and coarse residue (characterized as sand) dried which requires no confinement and can be utilized as a construction material;

- The residue will be produced in the following proportions: 15% fines and 85% coarse;
- Mass volume of fine residue pumped: 1.3t/m^3;
- Mass volume of coarse residue cyclone 1.3 t/m^3, trucked 1.7 t/m^3;
- Repose angle of fine residue: 0.75%;
- The process water transported with the fine residue will be recovered in a sedimentation ponds (treatment for suspended materials) and decanted toward the polishing pond to be later recycled to the plant;
- The process water transported with the coarse residue will be removed from the residue (by cyclone action or other) and decanted toward the polishing pond to be later recycled to the plant; and
- No wetland and/or lake areas may be used to store residue or process water.

16.5.2 Development Plan

Essentially the methodology is summarized in three steps:

- Considering the base assumptions relative to the freezing of water in the residue storage areas, it is necessary to double the capacity of the polishing pond in order to contain sufficient water to supply the plant during winter at the new production rate;
- Validation of the work for deviation of water runoff at the proximity of Lake Mazaré; and
- Calculate quantities and the investment schedule.



Source: Cliffs

Figure 16.5.1: Bloom Lake Tailings Storage Facility Plan

The previously permitted tailings footprint shown by the red dotted line in Figure 16.5.1 has the capability to store six to eight years of production from the start of operations in 2010. Due to water management requirements only half of the area is usable for tailings disposal. The western portion is not usable as runoff and reuse water could not be properly controlled. In 2012, Cliffs received the decree modification to allow for the expansion of the tailings footprint to the west shown in yellow. The additional western footprint increases the tailings storage up to 14 to 16 years from the start of operations. The topography of the area is better suited for runoff and reuse water management. Tailings work in 2012 and 2013 is located within the yellow and pink shaded areas.

The Northeast expansion area shown in green is currently being reviewed. This area has the potential to expand tailings disposal capacity to 40 years. Costs and structures required for operations in this area have yet to be determined but are expected not to exceed the current footprint. This area will impact numerous lakes so the timeline for permitting could be greater than three years.

It should be noted that government water management requirements (Directive 19) have recently been revised in 2012. These new requirements may impact the cost, timing, and construction requirements for storage water dams in order to handle additional capacity. Previous requirements called for the collection, storage and treatment of all runoff water up to the volume from a 1:100 year rain event in combination with the snow melt from a 1:100 year winter snow fall. The new requirement has revised the rain fall levels to a 1:1000 year event.

16.6 Phase II Mancamp

A permanent facility to accommodate additional workers at the mine will be built within the town of Fermont. Additional facilities including a wellness/recreation facility and new cafeteria are planned in town. Potable water supply, sewage treatment, fire protection and electric utilities are available at the Townsite.

The town has indicated that the construction camp currently located in town should be moved away from the town. Cliffs is making arrangements to provide a construction camp along the mine access road between highway 389 and the mine site. All services will need to be provided including potable water supply, sewage treatment, fire protection and electric utilities.

16.7 Rail Transport

At the mine site, empty open top railcars pass beneath the concentrate load-out silo and are automatically loaded with 100 t of ore. These cars are part of a 240 car train that can arrive daily at the site to begin the transportation process to the port. Mine site facilities include conveyors, storage silo, load-out tower, and a rail loop. For phase II an additional conveyor and storage silo will be added to handle the increased volume.

Interconnecting railways are then utilized for the transportation of the iron ore concentrate from the mine site to port. The 36 km long Bloom Lake Railway line connects the mine site with the Wabush Lake Railway near Labrador City. From there the ore is hauled over 400 km to reach the port. There are 750 insulated ore cars dedicated to move Bloom Lake concentrate as part of the rail fleet consisting of three trains.

As part of Phase II, 750 more railcars will be added to the fleet. It will also include the addition of rail sidings near the mine site, on the mainline, and at the port to handle the increased volume from the mine.

16.8 Port Facilities

Concentrate is unloaded from railcars at the Pointe Noire and can be either loaded directly onto a vessel or stockpiled to be reclaimed and loaded at a later date. The existing Bloom Lake concentrate stockpiling and shipping system is comprised of a rotary car dumper, dump hopper, stockpiling and reclaiming conveyors, a stacker-reclaimer, and ship loaders. Storage capacity is currently 550,000 t of concentrate in the stockpile yard.

There are two marine terminals at Pointe Noire consisting of Dock 30 and Dock 31. Both docks are owned by the Port of Sept-Iles and leased and operated by Cliffs. Iron ore concentrate can be shipped via Dock #31 using a stationary 6,000 t/h stationary shiploader. Dock 30, which is part of the Wabush Mine facilities, has two 3,500 t/h traveling shiploaders and the associated conveyor systems. The two marine terminals are now tied together via a cross conveyor that allows concentrate that is stored or unloaded on the Dock 31 side to be loaded into vessels at Dock 30.

The water draft at Dock 31 is limited and cannot be significantly increased. Vessel size is limited and vessel loading operations are hindered by the stationary shiploader. Several improvements have been made to the structure at Dock 30 and dredging will be completed to fully load capesize vessels independent of tidal conditions. A new shiploader is planned for Dock 30 that will significantly increase the loading rates of concentrate.

As part of Phase II, concentrate stockpile capacity will be increased through an addition of a new conveyor from the Bloom Lake system to the Wabash stockpile yard. This will allow for sufficient inventory to improve shipping and allow for reclaiming concentrate for vessel loading from both facilities.

17 Environmental Studies, Permitting and Social or Community Impact

17.1 Related Information

There is no related information. The following information was summarized in the Technical Report issued on December 31, 2011, and has been updated with more recent information developed since that time, as relevant and appropriate.

17.2 Environmental Study Results

An environmental impact study was prepared for the Project expansion. Environmental impacts and mitigation information provided in the request for modification are summarized below. Some information was taken from the previous NI 43-101 reports (Consolidated Thompson Iron Mines Ltd. and BBA Inc., 2008; CIMA, 2010; SRK, 2011). The modification request was granted in September 2011 and authorization for construction of the new infrastructure was granted in the same month.

17.2.1 Air

Potential non-point sources of dust include the tailings pond, the waste rock piles and the ore and concentrate stockpile areas. The mine has dust mitigation measures for fine particle emissions, such as dust collectors, at some of the crushing and transportation facilities. The laboratory has a dust collection system in the preparation area and fume hoods in the wet assay room. Current efforts at the mine include the construction of an A-Frame enclosure over the primary ore crusher and stockpile, which should reduce overall particulate emissions substantially.

17.2.2 Biological

No plants designated with a protection status are known in the area. Impacts to local lakes and water streams were identified in the initial environmental impact study. Bird habitat loss has occurred; however, the availability of similar habitats nearby is believed to provide suitable replacement habitat. The company was cited for non-compliance with a condition of a certificate of authorization for the construction of ponds C and D at Bloom Lake for improper deforestation following an inspection on October 16, 2012.

17.2.3 Cultural and Historic Resources

No archaeological or historic resources are known within the mine site; however, it is possible that artifacts or relics may be encountered because terrestrial environments within this lake area are the most probably sites for archaeological resources.

17.2.4 Water

The Bloom Lake Mine has a concession to withdraw water from Bloom Lake for domestic (potable) use and for boiler make-up water. There is no restriction regarding the volume extracted. Lake of Confusion water may be used during outage of the reuse water circuit, which recycles tailings water back to the plant for reuse as process water.

Potential sources of impacts to water quality are the tailings impoundment and sedimentation basins, the pit and waste rock dumps. Iron ore and iron concentrate stored at the stockpiles, plus fine particles in the tailings, could be windblown to surface water bodies, or infiltration from the facilities could reach groundwater. Specific potential impacts are increased metals concentrations, especially iron, and turbidity.

Impacted surface water and decant water from the tailings are collected in a settling pond. The settling pond water level is maintained by pumpback to the process water reservoir at the Concentrator or by intermittent pumping to a clarification pond. The water in the clarification pond is treated to flocculate fine solids and remove the red color, if needed, before discharge to the environment. Solids from the bottom of the clarification pond are pumped back to the tailings impoundment. The site did not provide information regarding compliance of the discharge to regulatory requirements. As noted in Section 2 of this report, the operator has been issued notices of violation for the release of water to the environment which did not meet the discharge criteria for total suspended solids.

17.2.5 Hazardous Materials Management

Drainage systems from wash bays in the truck shop and warehouse have sand and sludge interceptors, oil interceptors and a water/soap settling tank with oil skimmers. Drainage from the shop repair bays pass through an oil separator. Effluents from the wash bays and repair bays are discharged into the drainage ditch surrounding the plant site. Water in this ditch passes through two cleaning basins before it is discharged into Confusion Lake.

Fuel oil tanks have integral holding sections to retain leakage and prevent contamination to the ground. Used oil and lubricants and oil skimmed from ponds are burned in the boiler or disposed of off-site by a contractor. As noted in Section 2 of this report, the operator has been cited for several minor, and correctable, violations for improper storage of hazardous materials.

17.2.6 Tailings and Waste Rock

The tailings are mainly composed of silica and of some metals. The reclamation plan indicates that the waste rock and tailings have no acid mine drainage potential and that the beneficiation process does not use many chemicals.

Water from the Bloom Lake tailings pond was tested to determine whether treatment was needed prior to discharge. Based on a 14 day settling period, the water would not pass requirements for total suspended solids, color, pH, turbidity, conductivity, EMF and heavy metals under MDDEP Directive 019 limits. Two treatment plants were constructed to treat tailings water and impacted stormwater run-off. A third treatment plant is planned for 2013.

17.3 Environmental Issues

There were two environmental releases during May 2011. On May 22, 2011, an estimated 2,600 gal of ferric sulfate, a water treatment chemical used to treat mine water, spilled onto the ground from an above ground storage tank leak at the mine. An unknown quantity of the chemical migrated into Mazare Lake. Pumps were set up to pump contaminated water from the Lake into the mine wastewater treatment ponds, as well as recovered material that had spilled on the ground. The environmental impact associated with this spill is believed to be minimal. Two days later, on May 24, 2011, in an unrelated event, stormwater overflowed a basin, which caused the failure of a recently constructed dike. Untreated surface water run-off was subsequently discharged directly to Mazare Lake.

Both releases were properly reported to the Ministry of Sustainable Development, Environment and Parks (MDDEP). The MDDEP conducted an inspection and issued a Notice of Infraction that alleged the releases of a contaminant (TSS) and a dangerous substance (ferric sulfate) into the environment. MDDEP conducted a follow-up investigation in late 2012, but has not indicated further enforcement actions are warranted. In the meantime, Cliffs has implemented new measures to try and reduce the risk of such releases in the future.

There are no known significant issues that are believed to materially impact the mine's ability to operate.

17.4 Operating and Post Closure Requirements and Plans

Environmental monitoring during operations includes water and air quality monitoring.

The site monitors mine effluent, domestic effluent and groundwater. Effluent is continuously monitored (visually) with more specific analyses required on routine basis that ranges from three times a week to annually per federal and provincial requirements. There are nine groundwater monitoring points that are sampled twice per year for arsenic, calcium, copper, iron, potassium, magnesium, sodium, nickel, lead, zinc, petroleum hydrocarbons, total cyanide, bicarbonate, sulfate, pH and electrical conductivity.

There is one dust collector that is tested every three years and must be maintained in good operating condition. No other air emission requirements are in place.

17.5 Required Permits and Status

The mine has been authorized for operation under the federal environmental authority including Fisheries and Oceans Canada, Transport Canada, Natural Resources Canada and Environment Canada. The assessment process was overseen by the Canadian Environmental Assessment Agency.

Construction of an 8 Mt/y expansion for the Bloom Lake Iron Mine project was started in 2008 and commenced operation in March 2010. The project was subject to an environmental impact assessment and review process under Article 31 of the Environment Quality Act. Authorization of the project was given by the Quebec government in 2008. Additional certificates of authorization, in compliance with Articles 22 and 32 of the Environmental Quality Act, were approved for the construction of different infrastructures during Phase I. The certificate of authorization for the mine exploitation, ore treatment, waste rock and tailings disposition was granted in March 2010.

An expansion (Phase II) was proposed (CIMA, 2010) and approved by the Ministry of the Environment in a decree modification in September 2011, which approved the project in principle. An authorization for construction of new infrastructure associated with Phase II was also granted at that time. However, authorization to proceed with operation of Phase II is not expected by Cliffs until mid-2013. Under the current permits and authorizations, the LoM is only 20 years. At some point in the near future, the operation will need to seek authorization for the expansion of the waste rock dumps and tailings impoundment footprints in order to reach the expected 37-year LoM based on iron ore reserves. This may include expansion to the north, which would involve additional permitting by the Department of Fisheries and Oceans.

The mine has received operational permits for the mine, dust collection systems, railroad and the wastewater treatment systems (see Section 2.4.2). In addition, numerous other permits have been granted for construction of ancillary facilities such as dams and port installations, plus operation of quarries (Cliffs, 2013).

Known permit violations or pending actions associated with the Bloom Lake mining operations are discussed above.

17.5.1 Post-performance or Reclamation Bonds

The approved Closure Plan includes a financial guarantee of CAN$17,494,019, as described in Section 18.7. This instrument does not include the Phase II expansion; as such, an internal review of the new financial guarantee was completed and sent to the MDDEFP in January 2013 for review and approval (see below for details).

17.6 Social and Community

The area is used for limited holiday and recreational activities. The mine operator has made agreements with different users in order to compensate for impacts on community use of the lands within the mining lease. An agreement was signed between Consolidated Thompson and Innu Takuaikan Uashatmak Mani-Utenam representing the aboriginal people. Because these are non-public agreements, site did not provide detailed information regarding commitments made in the agreement, or whether those commitments have been fulfilled to date.

A noise study was conducted to assess impacts on the nearby areas. Noise levels at nearby Daigle Lake and Fermont are lower than the 40 dBA threshold allowed.

17.7 Mine Closure

The Bloom Lake Mine has prepared a reasonably comprehensive closure plan for Phase I of the operation (AMEC, 2009), which details closure objectives, baseline conditions, project description, closure actions, progressive rehabilitation, treatment of contaminated soils, closure and post-closure monitoring. This plan was approved in 2010 (Québec Ministère des Ressources Naturelles et de la Faune, 2010). The 2010 approval includes the requirement for a financial guarantee for reclamation which is to be funded through annual deposits starting in 2013 through 2023. The 2010 reclamation costs for the Phase I impacted areas were estimated at CAN$24,991,455, plus a contingency of CAN$3,259,744. The financial guarantee, per regulation, is estimated at 70% of the estimated reclamation cost (excluding the contingency cost); thus totaling CAN$17,494,019. This amount and the amortized funding schedule were included in the October 15, 2010 approval by the Ministry of Natural Resources and Wildlife.

The reclamation program includes the following activities:

- Progressive reclamation of the tailings area and waste rock dumps with local vegetation except the polishing pond, which will be converted into a marsh;
- All buildings, facilities and equipment will be dismantled. Recycling will be promoted. All material not sold/recycled will be deposited in an authorized site located within the property

limits. Contaminated material will be sent offsite for proper disposal in respect to applicable regulations;

- CLM's buildings located in Fermont City will be sold to the city or any other interested entity. If the city is going to close at the mine closure, buildings will be dismantled and restored;
- The concentrate pad (lined) will be emptied and no contamination is expected. The surface will be graded and revegetated with native species;
- All petroleum tanks and associated pipes will be disposed accordingly to applicable regulation; and
- Contaminated soil will be treated on site or disposed offsite in respect with regulations.

It was estimated that closure activities will take approximately three years. A monitoring program covering facilities stability, agronomy and environmental aspects will be performed for the five years following completion of the reclamation.

AMEC is currently developing an updated closure plan to include Phase II of the project; a copy of the preliminary report (AMEC, 2013) was provided for review, was completed and submitted for regulatory approval. The closure actions are described at a conceptual level but the approaches are sound and reflect current industry practices. Some important issues of note include:

- Material characterization for ARD and metal leaching potential, as reported by Golder (2012) indicate that waste, ore and tailings will be non-acid generating;
- The closure plan indicates that a pit lake will form post closure; however, a prediction of the future water quality is not currently available; and
- Preliminary indications are that surface water management will be a key issue post closure, but this part of the preliminary report has not yet been completed.

As the site still has another 30 years of operation (end of mine life is stated as 2043 based on current data), none of the aforementioned issues appear to be material at this time. Inclusion of Phase II into the project will, however, increase the overall reclamation costs for the project by over 150%, the financial guarantee for which would again be funded with annual installments between 2013 and 2024.

18 Capital and Operating Costs

18.1 Capital Cost Estimates

The life of mine total capital costs for the project are based on historical data obtained through on-going operations and estimated future investments of sustaining capital in addition to the Phase II expansion. Table 18.1.1 shows the Life of Mine capital costs.

Table 18.1.1: LoM Capital Costs (US$000's)

Description	2200 t/h Case	2400 t/h Case
Expansion	836,760	836,760
Productive	88,013	88,013
Safety	5,238	5,238
Environmental	395,555	395,555
Health	75	75
Quality	225	225
Infrastructure	51,497	51,497
Mobile Equipment (From previous 5 Yrs. Budget)	147,200	147,200
Mobile Equipment (After 5 Yrs. based on updated Mine Plan)	778,824	823,915
Other	55,556	55,556
Sustaining Capital	347,893	358,541
Mine Closure	132,000	132,000
Total Capital	**$2,844,736**	**$2,894,457**

The five year capital plan is presented in Table 18.1.2. The expansion portion of the capital is explained in detail Section 18.1.1-Phase II Expansion Capital.

Table 18.1.2: Current Bloom Lake Five Year Capital Plan (US$000's)

Area	2013	2014	2015	2016	2017	Total
Expansion	367.2	468.5	1.1	-	-	836.8
Production	-	83.0	5.0	-	-	88.0
Sustaining	100.9	254.7	145.0	93.2	61.4	655.2
Total Capital	**468.1**	**806.2**	**151.1**	**93.2**	**61.4**	**1,580.0**

Source: Cliffs

The total Phase II Expansion Capital totals US$1.4 billion as reported in the August, 2012 PHII forecast. See Table 18.1.3.

The five year budget includes remaining PHII expansion costs for 2013, 2014, and 2015 which will complete the Phase II Expansion.

Table 18.1.3: Phase II Expansion Capital Budget (US$ Million)

Capital Category	Total Capital
Development	68.7
Mobile Equipment	125.3
Plant	818.6
Housing	7.5
Expansion	**$1,020.0**
Garage	48.0
Tailings Disposal System	195.0
Mine Infrastructure Upgrades	49.2
Port and Rail	-
Common Infrastructure	**$292.2**
Grand Total	1,312.2
Contingency	106.7
Total with Contingency	**$1,418.9**

A detailed discussion of the Phase II Expansion Capital is found in Section 19.1.1.

18.1.1 Phase II Expansion Discussion

The Phase II expansion is intended to double production. The construction has been ongoing. The project has experienced a number of project capital cost revisions over the time that the project has been developed. The intent of this section of the report is to provide a brief reconciliation of the capital cost increases and provide an overview of some of the drivers of the changes.

The overall increase in project capital can be most easily seen in Figure 18.1.1.1 Project Capital Cost Increase Review. Note that the cost reported in the figure is reported in Canadian dollars (CAN$). This is useful and necessary for only this section (18.1.1) of the report to allow clear understanding of total cost and to be consistent with Cliff's internal reports and discussions with partner WISCO. The figure is based on summary data provided by Cliff's with details from review of project documents, previous technical reports, and meetings with the Cliffs and other project parties. The numerical basis for the figure is provided in Table 18.1.1.1.

Table 18.1.1.1: Bloom Lake Capital Reconciliation (CAN$ Millions)

CAN $million's	Acquisition Changes			Cliffs 2011 Modifications						Cliffs 2012 Modifications						
Capital Category	Pre-CLF Acquisition Estimate from CT	Scope / Design Change CLF Board	First Estimate after Acquisition	Broadening PH II Requirements	Scope / Design Change	OPEX to CAPEX	Additional Housing	Adjustments for Port and Rail	Cliffs BOD Approved January 2012	Move to Sustaining or Productive CAPEX	Move to OPEX	Change in Forecast	Scope Change	August 2012 Forecast	Variance To CT 2010 Estimate	Variance To BOD Approved
Development	0.0	0.0	0.0	113.8	0.0	0.0	0.0	0.0	113.8	0.0	(28.0)	(19.9)	2.8	68.7	68.7	(45.1)
Mobile Equipment	0.0	0.0	0.0	44.3	0.0	127.4	0.0	(60.0)	111.7	0.0	0.0	13.5	0.0	125.2	125.2	13.5
Plant	503.2	63.3	566.5	0.0	84.7	0.0	0.0	0.0	651.2	0.0	0.0	167.4	0.0	818.6	315.4	167.4
Housing	0.0	0.0	0.0	0.0	0.0	0.0	7.0	0.0	7.0	0.0	0.0	0.0	0.5	7.5	7.5	0.5
Expansion	503.2	63.3	566.5	158.1	84.7	127.4	7.0	(60.0)	883.7	0.0	(28.0)	161.0	3.3	1,020.0	516.8	136.3
Garage	0.0	0.0	0.0	20.7	4.8	0.0	0.0	0.0	25.5	0.0	0.0	22.5	0.0	48.0	48.0	22.5
Tailings Disposal System	0.0	0.0	0.0	57.2	73.9	0.0	0.0	0.0	131.1	0.0	0.0	11.1	52.7	195.0	195.0	63.8
Mine Infrastructure Upgrades	0.0	0.0	0.0	68.5	0.0	0.0	0.0	0.0	68.5	(38.9)	0.0	16.6	3.0	49.2	49.2	(19.3)
Port and Rail	48.4	0.0	48.4	100.0	2.9	0.0	0.0	(151.3)	0.0	0.0	0.0	0.0	0.0	0.0	(48.4)	0.0
Common Infrastructure	48.4	0.0	48.4	246.5	81.6	0.0	0.0	(151.3)	225.2	(38.9)	0.0	50.2	55.7	292.2	243.8	67.0
Total	551.6	63.3	614.9	404.6	166.3	127.4	7.0	(211.3)	1,108.9	(38.9)	(28.0)	211.2	59.0	1,312.2	760.6	203.3
Contingency	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	106.7	106.7	106.7
Total with Contingency	551.6	63.3	614.9	404.6	166.3	127.4	7.0	(211.3)	1,108.9	(38.9)	(28.0)	211.2	59.0	1,418.9	867.3	310.0
Entities Involved	CT	CT and Cliffs		Cliffs												

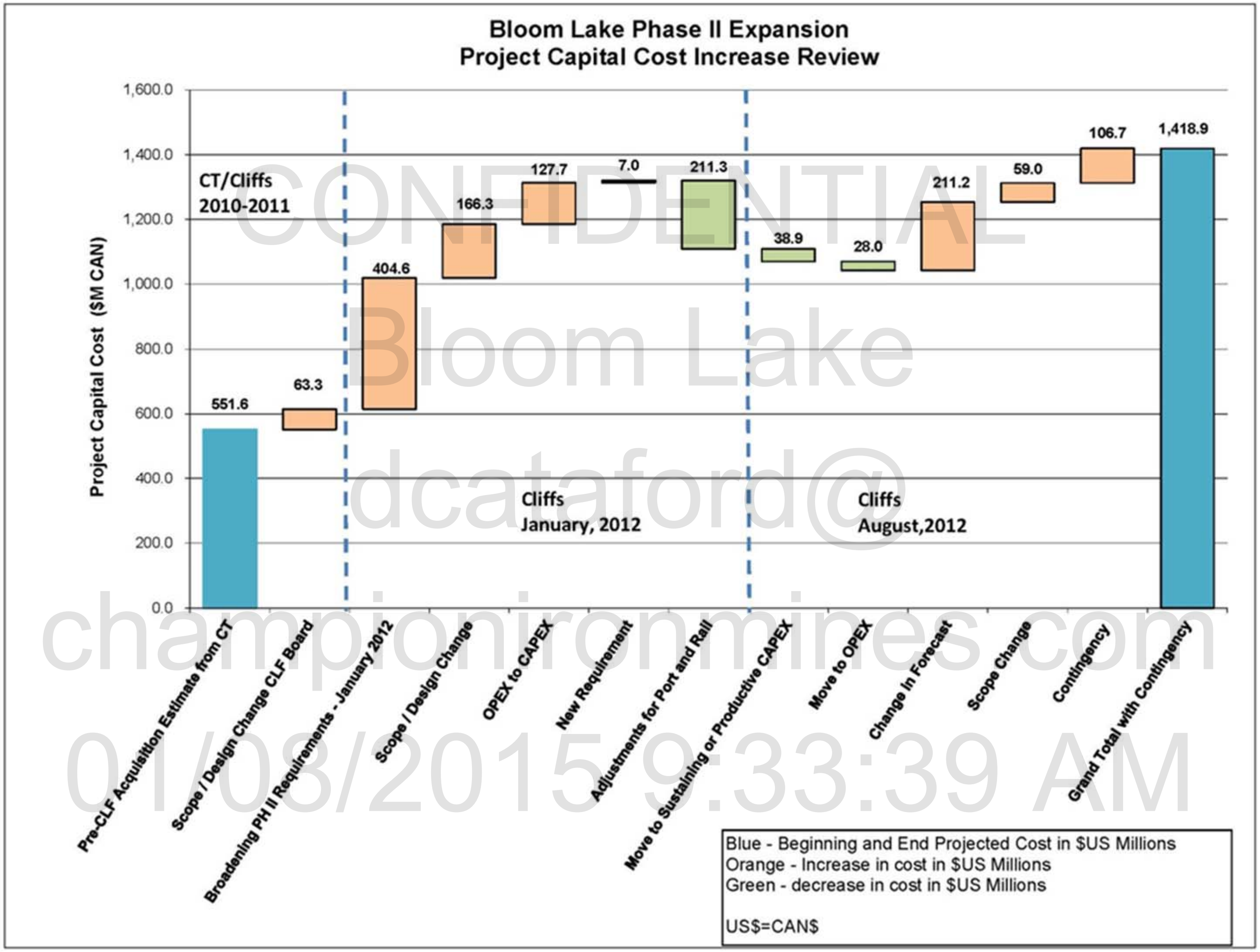


Figure 18.1.1.1: Expansion Project Capital

The first section on the left hand side of the figure represents the time that Consolidated Thompson Iron Mines Ltd. (CT in chart) had control of the project and was developing the Phase II portion of the project. The second section represents the time immediately after the acquisition by Cliffs (CLF in figure) and modifications/adjustments made during that initial time and approved by Cliffs Board of Directors in January 2012. The third box represents the most recent update reviewed which is dated August 2012. The discussion of the figure will describe the costs and basis for them to build to the total current capital estimate.

Pre-Cliffs Acquisition Estimate from Consolidated Thompson

The initial budget described in the June 22, 2010 Technical Report on Bloom Lake (prepared for CT) is summarized in Table 18.1.1.2 and shows both the Canadian dollar and US dollar equivalents at an exchange rate of CAN$1.05 per US$1.

Table 18.1.1.2: 2010 Project Capital Estimate

Unit	US$ Millions	CAN$ Millions
Direct Costs	Year 1,2,3	Year 1,2,3
Construction Phase - Initial CAPEX	427.6	449.0
Owner's Cost	11.6	12.2
EPCM	39.4	41.4
Contingency (10% TIC)	46.7	49.0
Total Project Cost	**$525.3**	**$551.6**

Source: Bloom Lake Technical Report – June 22, 2010

The capital project at this time consisted of a new crusher, 4 km of overland belt conveyor, with a luffing radial stacker to place material near the existing stockpile that allowed feed to both mills. Additionally, infrastructure and support systems included only an upgrade of the steam plant and increased fuel storage capacity. A new 34.5 kV power line was noted and additional apartment buildings in Fermont. Although not in the Technical Report, Port and Rail modifications were estimated at CAN$48.4 million.

Mining equipment and rail cars were to be leased and then bought back starting in 2016.

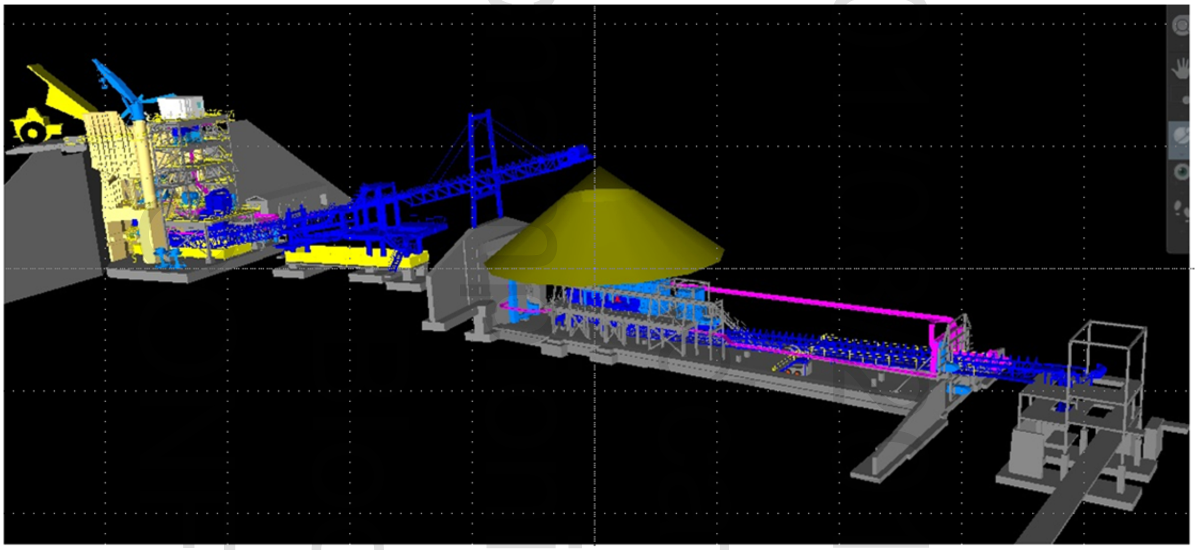
The new concentrator at this time was designed to be very similar to the existing Phase I concentrator. The concentrate would be discharged to stockpile and then reclaimed to a transfer conveyor to the load out tower. Tailings were to be disposed of at the same location as Phase I. Fine and coarse tailing would be pumped separately. After the initial period coarse tails dewatering cyclones would be located north of the fine tailings area.

Changes Before Acquisition by Consolidated Thompson and Cliffs

Scope/Design Change by CTL with participation of CLF board

CTL recognized certain deficiencies in the Phase II design and suggested modifications to the project prior to and during the acquisition of the mine by Cliffs. The modifications totaled CAN$63.3 million and included modifications to the crusher and overland conveyor, crude ore surge pile, concentrator recovery circuit, and concentrate storage.

The new crusher was modified to correct a surge bin issue that proved to be a bottleneck on the old crusher. A high speed 13,000 tonne per hour conveyor/feeder system was added in a tunnel under the crusher to feed onto a surge pile that empties to feeders that load the overland conveyor. The new system protects the feed unit onto the overland conveyor and improves reliability. A larger rock breaker was added at the crusher. A crane for repair work was also added to the crusher (Figures 18.1.1.2 and 18.1.1.3).



Source: Cliffs

Figure 18.1.1.2: New Crusher and Overland Conveyor Feeder

Source: Cliffs

Figure 18.1.1.3: Construction Photograph of New Crusher

The overland conveyor route was modified to its currently constructed location and required modifications to the support foundations and additional earthwork. It also required additional transfer chutes and modifications to the drive motors (Figure 18.1.1.4). The conveyor cost estimated in the NI 43-101 was determined to be too low.

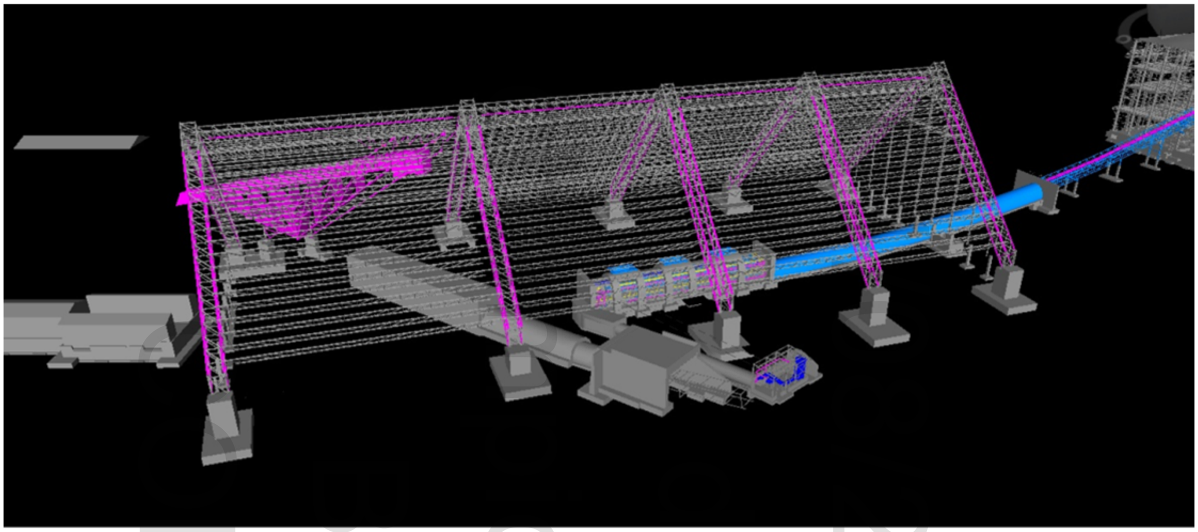
Source: Cliffs

Figure 18.1.1.4: Photograph of Overland Conveyor

A covered ore storage building with a tripper unloader and a modified feeder arrangement under the ore stockpile were added to address dust issues, improve stockpile handling, and allow blending of ore from pits to feed both concentrators (Figure 18.1.1.5).

Source: SRK

Figure 18.1.1.5: Photo of Ore Storage Area



Source: Cliffs

Figure 18.1.1.6: Ore Storage and Feeder System

The concentrator design was modified to add hydroclassifiers into the process flow. The spirals were raised roughly 8 meters to accommodate improvements to the discharge of the Filter Tables. The Filters were upsized from the 24 ft tables in Phase I to 32 ft. The thickener size was also increased to 45 meters in order to improve process water quality coming back to the plant.

Improvements to the final product handling system included the addition of another product conveyor and storage silo tied into the existing rail load out.

The Phase II budget totals CAN$614.9 million after this adjustment.

Source: Cliffs

Figure 18.1.1.7: Concentrate Storage

Cliffs 2011 Modifications

Broadening Phase II Requirements

After the acquisition the project was reviewed in detail by Cliffs and substantial improvements and deficiencies were noted. The previous project budget did not include adequate capital to cover costs for proper mine development, adequate mining equipment for production requirements, and common infrastructure for the current operation and expansion. The majority of the increase was for projects necessary to support the project as a whole and not tied directly to the second concentrator. Table 18.1.1.3 details the CAN$304.0 million additional capital.

Table 18.1.1.3: Broadening Phase II Requirements

Area	CAN$ Millions
Development	113.8
Mobile Equipment	44.3
Plant	-
Housing	-
Expansion	**158.1**
Garage	20.7
Tailings Disposal System	57.2
Mine Infrastructure Upgrades	68.5
Port and Rail	100.0
Common Infrastructure	**246.5**
Total	**CAN$404.6**

Development projects including additional overburden removal, internal development cost, and topographic drilling and blasting accounted for CAN$98.1 million or 86% of the expense. Other projects included: exploration of Bloom West Pit, mine expansion roads, and tree clearing in the mining area.

The need for an expanded shop facility was identified (garage) due to the increase number of pieces of equipment in the mine fleet and limited facilities on site.

The tailings delivery system was modified to pump both coarse and fine tailings to the tailings area. The old method of coarse tailings disposal was to pump to a drainage area, where the tails dewatered naturally. The coarse tails were then hauled by truck to build the tailings embankments and to final placement location. The new method is proposed to reduce operating costs. The additional cost includes dams and dikes, pump house tailings booster station and tailings lines, roads, and a polishing pond dam.

Source: Cliffs

Figure 18.1.1.7: Photo of the New Tailings Booster Station

Mine infrastructure upgrades include mine services and buildings, processing plant infrastructure, processing plant services and utilities, and on-site infrastructure. Table 18.1.1.4 provides the cost by area.

Table 18.1.1.4: Broadening Phase II – Mine Infrastructure Upgrades

Area	CAN$ (Millions)
Mine Services & Buildings	7.5
Processing Plant Infrastructure	21.0
Processing Plant Services & Utilities	19.6
On-Site Infrastructure	20.4
Indirects & Owner's Costs	-
Mine Infrastructure Upgrades Total	**CAN$68.5**

- The mine services and buildings include a fueling station and mine offices;
- Processing plant infrastructure includes addition of a third stockpile and stacker/reclaimer, stockpile reclaim hopper, HMI redesign and control room reconfiguration, addition of a metal magnet and dust mitigation repairs;
- The modifications to processing plant services and utilities included the addition of a water treatment plant, electrical upgrades, emergency water supply for PHII, and sanitary system upgrades; and
- On-site infrastructure additions include road reconfiguring, new cafeteria, laboratory expansion, warehouse additions and dock facilities, gatehouse, and rails/sidings and rail fuel unloading and storage system to enable the purchase of bulk fuel by rail.

Scoping / Design Change

Incremental increases in cost due to scope changes and design changes resulted in an increase of CAN$166.3 million that is segregated as seen in Table 18.1.1.5.

Table 18.1.1.5: Scoping / Design Change

Area	CAN$ (Millions)
Development	
Mobile Equipment	
Plant	84.7
Housing	
Expansion	**84.7**
Garage	4.8
Tailings Disposal System	73.9
Mine Infrastructure Upgrades	
Port and Rail	2.9
Common Infrastructure	**81.6**
Total	**CAN$166.3**

The changes on the plant includes electrical work, upgrade to overland conveyor steel, foundations, and the conveyor itself, additional engineering, liner purchase, load-out, services including fire protection and pumping. Additionally, there is a thickener design change. The project capital can be seen in Table 18.1.1.6.

Table 18.1.1.6: Scoping/Design Change Plant

Area	CAN$ (Millions)
Electrical Works	20.2
Overland Foundations	18.2
Services	12.0
Load-out	5.0
Crusher Installation	4.8
Overland Transport	4.5
Engineering	4.4
Overland Conveyor	3.4
Liner Purchasing	2.0
Pumps	1.9
Foreign Exchange	1.6
Fire Protection	1.5
Thickener Change	1.4
Additional Steel	1.3
GPSI Consulting	1.0
Water Treatment Basins	1.0
Miscellaneous	0.5
Plant Total	**CAN$84.7**

Tailings disposal included additional work in dikes/dams, tailings booster station and tailings pipelines, and polishing pond.

Operating Expense (OPEX) to Capital Expense (CAPEX) - CAN$127.4 million

CT originally planned to utilized equipment that was going to be leased and therefore an operating cost. Cliff's chose to direct purchase the equipment and thereby reclassifying the equipment as capital. This included all equipment for mining operations and the additional rail cars for Phase II requirements.

Additional Housing - CAN$7.0 million

A housing facility for company employees is maintained in Fermont. The expansion will require the addition of a second housing unit (hotel) for the additional employees. CAN$7 million is allocated for this expense.

Remove Rail and Port Capital - CAN$(211.3) million

Estimates to this point contain port and rail system costs, in total CAN$211.3 million, CAN$60 million for rail cars and the remaining CAN$151.3 million for rail and port upgrades. The Port work included yardworks, inloading and outloading, port storage, wharf material handling equipment, and wharf and jetty improvements. The rail system work included loops and sidings. Additional funds were designated for growth projects on the rail and port. It should be noted that the Bloom Lake Partnership is not responsible for these capital improvements.

Cliffs BOD Approved Capital Budget for Phase II – January 2012

Table 18.1.1.7: Capital Budget for Phase II – January 2012

Area	CAN$ (Millions)
Development	113.8
Mobile Equipment	111.7
Plant	651.2
Housing	7.0
Expansion	**883.7**
Garage	25.5
Tailings Disposal System	131.1
Mine Infrastructure Upgrades	68.5
Port and Rail	0.0
Common Infrastructure	**225.2**
Total	**CAN$1,108.9**

Cliffs 2012 Modifications

Move to Sustaining or Production CAPEX - CAN$(38.9) million

Cliff's elected to move CAN$38.9 million to productive or sustaining capital. Table 18.1.1.8 shows the breakdown of the items moved out of project capital.

Table 18.1.1.8: Move to Sustaining or Production CAPEX

Category	CAN$ (Millions)
Mine Services & Buildings	(2.5)
Processing Plant Infrastructure	(18.5)
Processing Plant Services & Utilities	(0.3)
On-Site Infrastructure	(17.7)
Total Mine Infrastructure Upgrades	**CAN$(38.9)**

The mine service item is a fueling bay. The processing plant infrastructure includes a stockpile and reclaimer at the site, magnet, filtration table, and dusting issue capital. The processing plant services and utilities are reduced by moving sanitary system upgrades. On-site infrastructure was reduced through moving the cafeteria, warehouses, docks for warehouse, gatehouse, fuel tanks and sidings for rail delivery of fuel.

Move to Operating Cost (OPEX) - CAN$(28.0) million

Overburden removal and haulage in the amount of CAN$28 million was moved to operating cost.

Change in Forecast

A reforecast of the cost provided an updated cost for the project. Table 18.1.1.9 represents the changes.

Table 18.1.1.9: Change in Forecast

Area	CAN$ (Millions)
Development	(19.9)
Mobile Equipment	13.5
Plant	167.4
Housing	-
Expansion	**161.0**
Garage	22.5
Tailings Disposal System	11.1
Mine Infrastructure Upgrades	16.6
Port and Rail	-
Common Infrastructure	**50.2**
Total	**CAN$211.2**

Changes included a reduction in development that was primarily a double accounting for internal development cost that was covered in overburden removal. The increase in mobile equipment was based primarily on an increase in truck cost.

Plant forecast changes were the largest portion of the adjustment and Table 19.1.1.10 shows the breakdown.

Table 18.1.1.10: Plant Change in Forecast

Area	CAN$ (Millions)
Construction	85.2
Purchasing	0.6
Services & Indirects	50.1
Owner's Costs	31.5
Plant Total	**CAN$167.4**

The construction changes in the plant construction by trade can be seen in the following Table 18.1.1.11.

Table 18.1.1.11: Construction Change in Forecast

Area	CAN$ (Millions)
Civil	10.5
Concrete	0.7
Structural Steel	10.9
Architectural	5.1
Mechanical	16.1
Piping	26.8
Electrical	15.0
Instrumentation	0.1
Construction Total	**CAN$85.2**

The Services & Indirect Change in forecast in Table 18.1.1.10 is broken down in Table 18.1.1.12 with detail.

Table 18.1.1.12: Services and Indirects Change in Forecast

Area	CAN$ (Millions)
Services & Indirects	33.0
Engineering & Construction Mgmt. Services & Indirects	17.1
Plant	-
Services and Indirects Total	**CAN$50.1**

The major contributors to increases in Services & Indirects of CAN$33.0 million, is made up of equipment delivery, cranes, concrete purchasing, and camp charges. The engineering and construction management increases were primarily by CIMA, Nardella, and CLM.

Projected cost increases in the Tailings Disposal category was primarily in the pumphouse (tailings booster station and tailings pipeline. The CAN$16.6 million increase in Mine Infrastructure Upgrades was driven by indirects and owner's cost increases.

<u>Scope Change</u>

Scope changes resulted in an additional CAN$59.0 million of increased capital. Table 19.1.1.13 reviews the breakdown of the projected cost difference.

Table 18.1.1.13: Change in Scope

Area	CAN$ (Millions)
Development	2.8
Mobile Equipment	-
Plant	-
Housing	0.5
Expansion	**3.3**
Garage	-
Tailings Disposal System	52.7
Mine Infrastructure Upgrades	3.0
Common Infrastructure	**55.7**
Total	**CAN$59.0**

Dikes and dams contribute the majority of the change in scope. The increase of CAN$46.3 million is chiefly from the West/North Dike partial construction, Dike A raise, dike contractor fixed cost, and project management cost. The remainder of the change includes CAN$13 million for additional roadwork and a deduction of CAN$(6.6) million for the removal of the polishing pond dam from the scope.

<u>Contingency</u>

The contingency for the project totaling CAN$106.7 million was determined by standard industry assessments. Each area was evaluated based on the level of design work complete, the projected construction rates, and overall definition of the scope and cost. Cost items completed but not billed were included in the consideration, as well as potential change orders not processed. The contingency by task ranges from a low of 2% to a high of 35%.

Table 18.1.1.14 is a summary of the project contingency by area.

Table 18.1.1.14: Contingency

Area	Percent (%)	Calculated Contingency - CAN$ (Millions)
Development	2.8	1.9
Mobile Equipment	3.9	4.9
Plant	9.0	73.3
Housing	5.3	0.4
Expansion	**7.9%**	**80.5**
Garage	10.0	4.8
Tailings Disposal System	4.2	8.1
Mine Infrastructure Upgrades	27.0	13.3
Common Infrastructure	**9.0%**	**26.2**
Total	**8.1%**	**CAN$106.7**

The plant contingency of CAN$73.3 million is 69% of the total project contingency with mine infrastructure, garage and tailings following at 27%, 10%, and 4.2%, respectively. The plant contingency is broken down in Table 18.1.1.15.

Table 18.1.1.15: Plant Contingency

Area	Percent (%)	CAN$ (Millions)
Construction	10.0	34.8
Purchasing	6.5	14.8
Services & Indirects	8.0	16.4
Owner's Costs	18.8	7.4
Total Plant Contingency	**9.0%**	**CAN$73.3**

Expansion Capital Estimate as of August 2012

Table 18.1.1.16: Capital Budget for Phase II – August 2012

Area	CAN$ (Millions)
Development	70.6
Mobile Equipment	130.1
Plant	891.9
Housing	7.9
Expansion	**1,100.5**
Garage	52.8
Tailings Disposal System	203.1
Mine Infrastructure Upgrades	62.5
Port and Rail	0.0
Common Infrastructure	**318.5**
Total	**CAN$1,418.9**

18.2 Operating Cost History

Operating costs for the project is based on historical data obtained through on-going operations and estimated future operations.

Tables 18.2.1 through 18.2.3 show historical costs for 2010, 2011 and 2012, respectively.

Table 18.2.1: 2010 Operating Costs (US$000's) – July through September

		Driver			Cost per ton per driver		
Sector	**Cost**	**Total material moved**	**Ore fed to Mill**	**Tonnes produced**	**Total material moved Cost**	**Mill Feed Cost**	**Product Cost**
Mine	46,121,022	13,271,721	6,353,365	2,394,095	3.48	7.26	19.26
Mill	31,252,652	13,271,721	6,353,365	2,394,095	2.35	4.92	13.05
Logistic	60,159,526	13,271,721	6,353,365	2,394,095	4.53	9.47	25.13
Admin	11,639,134	13,271,721	6,353,365	2,394,095	0.88	1.83	4.86
Total	**US$149,172,334**				**US$11.24**	**US$23.48**	**US$62.31**

Table 18.2.2: 2011 Operating Costs (US$000's) – January through December

		Driver			Cost per ton per driver		
Sector	**Cost**	**Total material moved**	**Ore fed to Mill**	**Tonnes produced**	**Total material moved Cost**	**Mill Feed Cost**	**Product Cost**
Mine	142,384,37	30,604,973	15,604,183	5,466,155	4.65	9.12	26.05
Mill	112,042,40	30,604,973	15,604,183	5,466,155	3.66	7.18	20.50
Logistic	163,243,921	30,604,973	15,604,183	5,466,155	5.33	10.46	29.86
Admin	39,415,012	30,604,973	15,604,183	5,466,155	1.29	2.53	7.21
Total	**US$457,085,712**				**US$14.94**	**US$29.29**	**US$83.62**

Table 18.2.3: 2012 Operating Costs (US$000's) – January through December

		Driver			Cost per ton per driver		
Sector	**Cost**	**Total material moved**	**Ore fed to Mill**	**Tonne produced**	**Total material moved Cost**	**Mill Feed Cost**	**Product Cost**
Mine	139,637,456	37,400,000	15,834,000	5,450,000	3.73	8.82	25.62
Mill	96,862,458	37,400,000	15,834,000	5,450,000	2.59	6.12	17.77
Logistic	183,856,813	37,400,000	15,834,000	5,450,000	4.92	11.61	33.74
Admin	58,776,198	37,400,000	15,834,000	5,450,000	1.57	3.71	10.78
Total	**US$479,132,925**				**US$12.81**	**US$30.26**	**US$87.91**

Costs have gone up due to the following reasons:

- Higher strip ratio. To sustain Phase II expansion, the current reserves were not sufficient. By increasing the reserves, mining deeper is necessary which increases the strip ratio;
- The transshipment cost increased to almost US$10/t of product. Once the new updated port is reconfigured and built, this operating cost will no longer be applicable; and
- The need for extra contractors (mining–stripping, plant coarse tailing movement) have increased. When Bloom Lake commenced production, there was not enough mining equipment purchased, large or small. To make up for this deficiency, contractors were brought in. Until all the large and smaller equipment is purchased, continuing higher operating cost will persist.

Future operating costs are based on the following assumptions:

- Mass recovery (MR) = Concentrate Mass = [Mass Crude Ore*[Crude Fe(t) * Fe Recovery]]/Concentrate Fe(t). (Based on 75.3% Fe Recovery and a 66.2% Fe Concentrate);
- Average Product Value = US$112.00/t of concentrate;
- Bulk Mining Cost/t Ore = US$2.70/t; Waste = US$2.70/t (SRK proposed mining CAPEX must be spent to achieve this rate) ;
- Surface Mining Cost/t AMT (All undisturbed <20 m upper surface) = US$4.10/amt;
- Tailings and water management = US$1.00/t RoM;
- Processing (Plant) Cost = US$6.62/t RoM;
- Admin/SG&A Costs = US$4.50/t of concentrate; and
- Logistics: Port and Transportation = US$25.00/t Concentrate (Assuming that port is built and transshipment services are no longer needed)

Based on 2,200 t/h Phase I and Phase II mill throughput, LoM operating costs will be near US$75.00/t of product.

19 Economic Analysis

The financial results of this report are based upon work performed by SRK and have been prepared on an annual basis. All costs are in U.S. constant dollars.

19.1 Principal Assumptions

A financial model was prepared on an pre-tax basis the results of which are presented in this section. Key criteria used in the analysis are discussed in detail throughout this report. Financial assumptions used are shown summarized in Table 19.1.1. The exchange rate used for all economic analysis is CAN$1 to US$1.

Table 19.1.1: Model Parameters

Description	2,200 t/h Case Value	2,400 t/h Case Value
Mine Life	30 years	27 years
Ore Processed	1,044.2 Mt	1,044.2 Mt
Tonnes Mined	2,593.5 Mt	2,593.5 Mt
Payable Tonnes Fe Concentrate	341.4 Mt	341.4 Mt
Life of Mine Operating Costs	$73.14/t Conc.	$67.77/t Conc.
Life of Mine Capital	$2,844.7 million	$2,894.6 million
Life of Mine Fe Recovery	75.35%	75.35%
Market Price (LoM avg.)	$112.00/t Conc.	$112.00/t Conc.

19.2 Cashflow Forecasts and Annual Production Forecasts

The following contain the production and cost information developed for the project. Table 19.2.1 is a summary of the estimated mine production over a 30-year mine life.

Table 19.2.1: Mine Production Summary

Description	Units	2200 t/h Case	2400 t/h Case
Mine Production			
Waste	kt	1,495,332	1,495,332
Low Grade (Waste)	kt	53,986	53,986
RoM Ore	kt	1,044,162	1,044,162
Total Material	**kt**	**2,593,480**	**2,593,480**
Daily Capacity	kt-mat/day	269,000	281,000
RoM Grade			
Fe Grade		28.73%	28.73%
Contained Metal			
Iron	kt	299,950	299,950

A summary of the estimated process plant production for the project is contained in Table 19.2.2 below.

Table 19.2.2: Plant Production Summary

Description	Units	2200 t/h Case	2400 t/h Case
RoM Material Milled	**kt**	**1,044,162**	**1,044,162**
Daily Capacity	t/day	105,000	115,000
Plant Production			
Recovery			
Fe Recovery		75.35%	75.35%
Product-Concentrate			
Fe Concentrate	kt	341,422	341,409
Total Concentrate	**kt**	**341,422**	**341,409**

The economic analysis results, shown in Table 19.2.3, indicate an NPV 9.7% of US$2,545.2 million and IRR of 34% (pre-tax basis) for the 2200 t/h case and an NPV 9.7% of US$3,482.0 million and IRR of 44% (pre-tax basis) for the 2400 t/h case. The NPV and rate of return were calculated for the project as a whole, including revenue from installed and expansion capacities. Capital identified in the economics is for sustaining operations, plant rebuilds as necessary and the Phase II Expansion.

Project economic results and estimated cash costs are summarized in Table 19.2.3 and Table 19.2.4.

Table 19.2.3: Mine & Plant Economic Results

Description	units	2200 t/h Case		2400 t/h Case	
Market Price	/t-product		$112.00		$112.00
Estimate of Cash Flow (all values in $000's)					
Gross Income					
Fe-Concentrate	$112.00		38,239,236		38,239,236
Gross Income			**$38,239,236**		**$38,239,236**
Net Revenue	0.0%		**$38,239,236**		**$38,239,236**
Operating Costs					
Mining (US$/t-mined)		$2.80	7,254,027	$2.83	7,341,227
Concentrating (US$/t-RoM)		$6.62	6,912,351	$4.95	5,168,601
Tailings & Water Manag. (US$/t-RoM)		$1.00	1,044,162	$1.05	1,093,391
Logistics (US$/t-conc)		$21.65	7,390,910	$21.65	7,390,607
G&A (US$/t-conc)		$6.94	2,370,886	$6.27	2,142,144
Total Operating			**$24,972,335**		**$23,135,970**
Operating Margin (EBITDA)			$13,265,905		$15,101,863
Capital			2,844,736		2,894,575
Pre-Tax Cash Flow Available for Debt Service			**$10,422,165**		**$12,207,288**
Pre-Tax NPV 9.7%			**$2,545,153**		**$3,482,001**
Pre-Tax IRR			**34%**		**44%**

Economic results assume that port and rail capital are allocated.
Economics assume that current trans-shipment cost will end in 2014.
2,200 t/h SRK view of the project as it is currently operated.
2,400 t/h Cliffs view of the project applying extra capital to improve cash costs.

Table 19.2.4: Cash Costs Summary

Cash Costs	Units	2200 tpd	2400 tpd
Fe-Concentrate	/t-product	$112.00	$112.00
Milled Material	kt	1,044,162	1,044,162
Fe-Concentrate	kt-product	341,422	341,409
Total Revenue			
Gross Income		38,239,236	38,237,833
Total Revenue		**$38,239,236**	**$38,237,833**
$/t-product		**$112.00**	**$112.00**
Costs			
Operating Costs		24,972,335	23,135,970
Cash Cost		**$24,972,335**	**$23,135,970**
Mining (US$/t-product)		$21.25	$21.50
Concentrating (US$/t- product)		$20.25	$15.14
Tailings & Water Manag. (US$/t- product)		$3.06	$3.20
Logistics (US$/t- product)		$21.65	$21.65
G&A (US$/t- product)		$6.94	$6.27
Total $/t-product		73.14	67.77
Margin			
Operating Margin		**$13,266,901**	**$15,101,863**
$/t-product		**$38.86**	**$44.23**

2,200 t/h SRK view of the project as it is currently operated.
2,400 t/h Cliffs view of the project applying extra capital to improve cash costs.

Figure 19.2.1 and Table 19.2.5 contain the annual estimates of mine production for the 2200 t/h mine production and Figure 19.2.2 and Table 19.2.6 contain the annual estimates of mine production for the 2400 t/h cases.

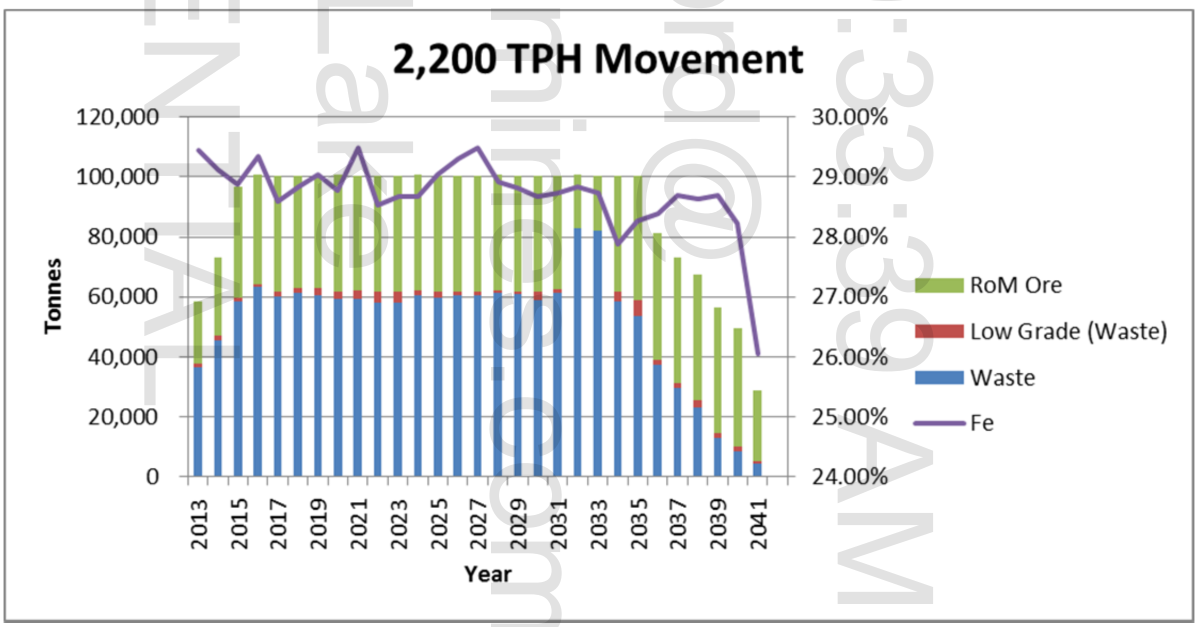


Figure 19.2.1: 2200 t/h Case Annual Production

Table 19.2.5: 2200 t/h Case Annual Concentrate Production and Cashflow

Year	Waste	Low Grade (Waste)	RoM Ore	Total	Ore Grade (%)	Free Cash (US$000's)
2013	36,506	1,019	20,875	58,400	29.4	-418,549
2014	45,389	1,520	26,091	73,000	29.1	-568,145
2015	58,524	1,034	37,167	96,725	28.9	271,177
2016	63,302	1,039	36,309	100,650	29.3	378,567
2017	60,038	1,698	38,639	100,375	28.6	423,403
2018	61,255	1,589	37,532	100,375	28.8	462,397
2019	60,342	2,618	37,415	100,375	29.0	449,643
2020	59,208	2,677	38,765	100,650	28.8	474,711
2021	59,223	2,707	38,445	100,375	29.5	482,160
2022	57,843	3,910	38,622	100,375	28.5	347,612
2023	57,908	3,841	38,627	100,375	28.7	438,892
2024	60,611	1,335	38,704	100,650	28.7	394,799
2025	59,615	2,131	38,629	100,375	29.0	271,445
2026	60,298	1,431	38,646	100,375	29.3	463,295
2027	60,325	1,418	38,632	100,375	29.5	484,685
2028	61,453	461	38,736	100,650	28.9	458,997
2029	60,762	962	38,651	100,375	28.8	459,446
2030	58,714	3,037	38,624	100,375	28.7	367,666
2031	61,179	1,423	37,773	100,375	28.7	403,806
2032	82,701	109	17,840	100,650	28.8	125,244
2033	82,086	135	18,154	100,375	28.7	13,374
2034	58,650	3,111	38,614	100,375	27.9	274,575
2035	53,568	5,410	41,396	100,375	28.3	367,014
2036	37,337	1,654	42,090	81,081	28.4	540,788
2037	29,767	1,553	41,975	73,295	28.7	544,038
2038	23,074	2,305	41,975	67,354	28.6	576,155
2039	13,013	1,448	41,975	56,436	28.7	613,293
2040	8,290	1,671	39,619	49,580	28.2	614,475
2041	4,353	740	23,642	28,735	26.1	623,366
2042	-	-	-	-	0	200,223
2043						(116,387)
Total	**1,495,332**	**53,986**	**1,044,162**	**2,593,480**	**28.73%**	**10,422,165**

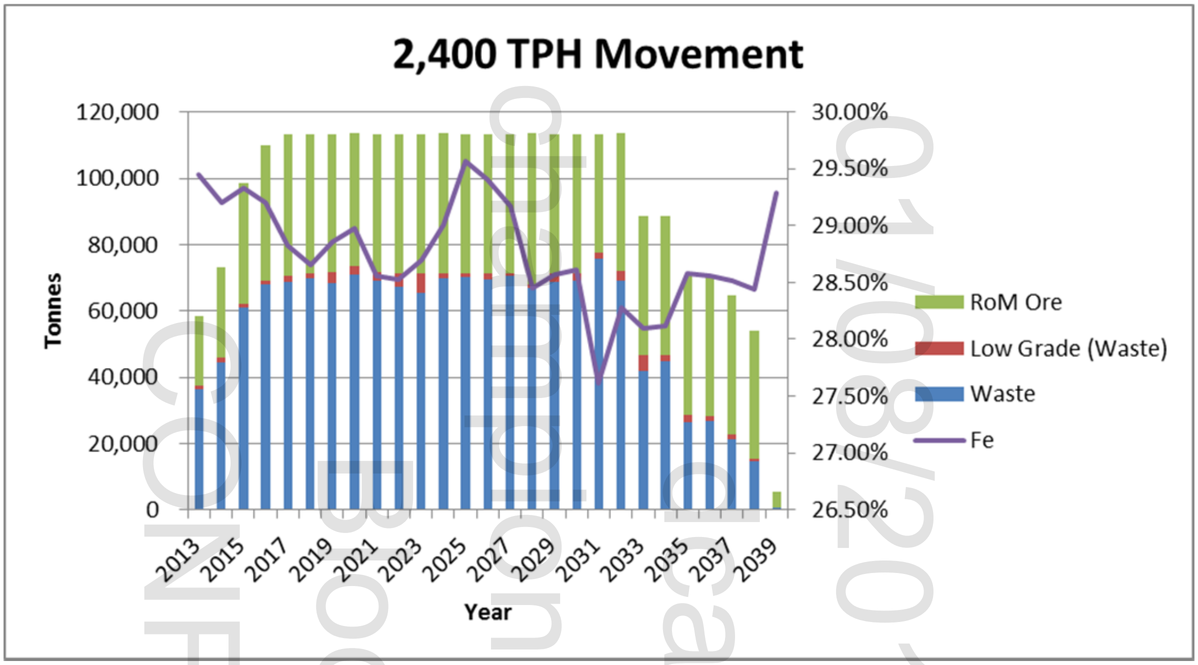


Figure 19.2.2: 2400 t/h Case Annual Production

Table 19.2.6: 2400 t/h Case Annual Production

Year	Waste	Low Grade (Waste)	RoM Ore	Total	Ore Grade (%)	Free Cash (US$000's)
2013	36,506	1,019	20,875	58,400	29.4	-385,869
2014	44,449	1,494	27,057	73,000	29.2	-528,838
2015	60,935	1,046	36,569	98,550	29.3	374,123
2016	67,887	1,135	40,779	109,800	29.2	501,665
2017	68,610	2,106	42,434	113,150	28.8	524,296
2018	69,839	1,460	41,851	113,150	28.7	574,646
2019	68,401	3,150	41,599	113,150	28.9	568,562
2020	70,931	2,652	39,877	113,460	29.0	548,786
2021	69,214	2,385	41,551	113,150	28.6	537,969
2022	67,360	3,811	41,980	113,150	28.5	465,121
2023	65,248	5,895	42,007	113,150	28.7	501,727
2024	69,692	1,674	42,094	113,460	29.0	521,250
2025	70,073	1,101	41,976	113,150	29.6	384,816
2026	69,632	1,535	41,983	113,150	29.4	593,927
2027	70,551	627	41,971	113,150	29.2	566,541
2028	67,020	916	45,524	113,460	28.5	555,484
2029	68,591	2,526	42,033	113,150	28.6	545,668
2030	68,932	2,209	42,009	113,150	28.6	438,307
2031	75,653	2,007	35,490	113,150	27.6	456,498
2032	69,260	2,858	41,342	113,460	28.3	483,349
2033	41,765	4,700	41,975	88,439	28.1	586,224
2034	44,677	1,918	41,975	88,570	28.1	569,893
2035	26,448	2,171	41,975	70,594	28.6	627,297
2036	26,923	1,347	42,090	70,361	28.6	704,806
2037	21,396	1,318	41,975	64,689	28.5	707,705
2038	14,587	928	38,382	53,896	28.4	684,256
2039	751	-	4,789	5,541	29.3	213,237
2040	-	-	-	-	0.0	-114,160
2041	-	-	-	-	0.0	0
2042	-	-	-	-	0	0
Total	**1,495,332**	**53,986**	**1,044,162**	**2,593,480**	**28.7**	**12,207,288**

Table 19.2.7: 2400 t/h Case Annual Concentrate Production and Cashflow

Year	Concentrate	Conc. Grade (%)	Free Cashflow	Discount Cashflow
2013	6,704	66.2	(385,869)	(385,869)
2014	8,368	66.2	(528,838)	(482,076)
2015	12,086	66.2	374,123	310,886
2016	13,600	66.2	501,665	380,009
2017	13,817	66.2	524,296	362,034
2018	13,735	66.2	574,646	361,716
2019	13,818	66.2	568,562	326,241
2020	13,194	66.2	548,786	287,050
2021	13,552	66.2	537,969	256,510
2022	13,676	66.2	465,121	202,165
2023	13,757	66.2	501,727	198,793
2024	13,941	66.2	521,250	188,267
2025	14,175	66.2	384,816	126,699
2026	14,098	66.2	593,927	178,257
2027	13,981	66.2	566,541	155,003
2028	13,676	66.2	555,484	138,539
2029	13,693	66.2	545,668	124,057
2030	13,711	66.2	438,307	90,838
2031	12,358	66.2	456,498	86,242
2032	13,353	66.2	483,349	83,241
2033	13,470	66.2	586,224	92,030
2034	13,479	66.2	569,893	81,556
2035	13,700	66.2	627,297	81,833
2036	13,728	66.2	704,806	83,814
2037	13,670	66.2	707,705	76,717
2038	12,466	66.2	684,256	67,617
2039	1,602	66.2	213,237	19,208
2040	0	0.0	(114,160)	(9,374)
2041	0	0.0	0	0
2042	0	0.0	0	0
Total	**341,409**	**64.1%**	**12,207,288**	**3,482,001**

19.3 Taxes, Royalties and Other Interests

Taxes and other interests have not been calculated for the project.

19.4 Sensitivity Analysis

Sensitivity analysis for key economic parameters is shown in Table 19.4.1 and Table 19.4.2. The Project is nominally most sensitive to market prices (revenues) as shown in Figure 19.4.1 and Figure 19.4.2. The Project's sensitivities to capital and operating costs are similar but slightly more susceptible to operating costs.

Table 19.4.1: 2200 t/h Case Sensitivity Analysis of NPV 9.7% (US$M)

NPV (US$MM)	-20%	-10%	Base	+10%	+20%
Revenue	(140)	1,203	2,545	3,888	5,230
Operating Costs	4,365	3,455	2,545	1,635	726
Capital Costs	2,901	2,723	2,545	2,367	2,189

Table 19.4.2: 2400 t/h Case Sensitivity Analysis of NPV 9.7% (US$M)

NPV (US$MM)	-20%	-10%	Base	+10%	+20%
Revenue	623	2,053	3,482	4,911	6,341
Operating Costs	5,278	4,380	3,482	2,584	1,686
Capital Costs	3,848	3,665	3,482	3,299	3,116

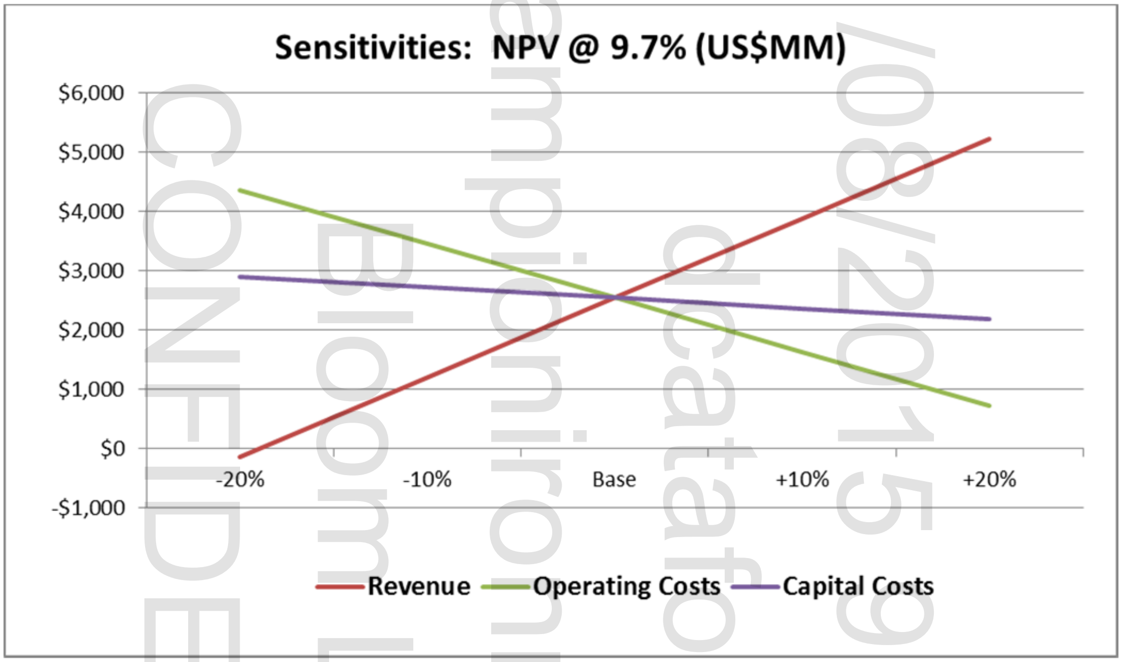


Figure 19.4.1: 2200 t/h Case Sensitivities of NPV 9.7% (US$M)

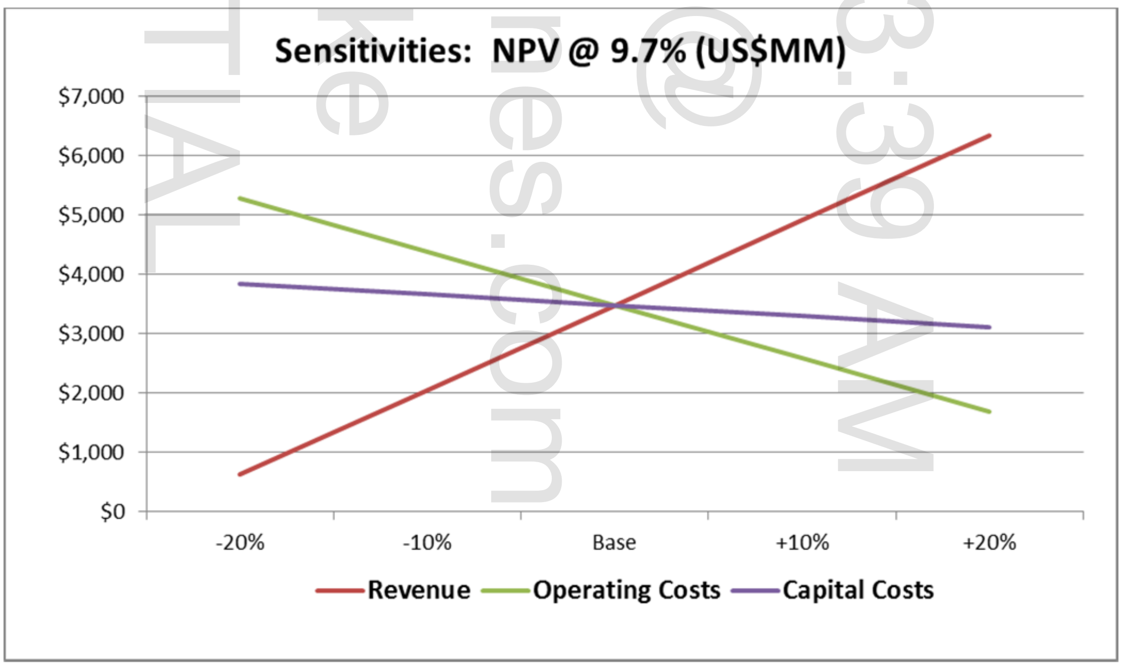


Figure 19.4.2: 2400 t/h Case Sensitivities of NPV 9.7% (US$M)

19.4.1 Results

Based on the data that was supplied and the analysis performed for the project, both cases studied present strong pre-tax economic results, as detailed above.

20 Interpretation and Conclusions

20.1 Results

20.1.1 Environmental and Permitting

The mine has been authorized for operation under the federal environmental authorities and provincial governments. The mine has received authorization for the Phase II expansion (in principle), and been granted permission to construct the associate structures and facilities for that expansion. Authorization to commence operations of Phase II is expected from the Ministry of Sustainable Development, Environment and Parks (Québec) in mid-2013.

The mine conducts routine monitoring of water, wastewater and air as part of their decrees and authorizations. With the exception of some minor releases to air and water, and improper storage of hazardous materials, the mine appears to be in compliance with its permits. The mine has an approved closure plan and should begin in 2013 with amortized annual payments to fund the financial guarantee established for reclamation. The amount of the financial guarantee is set at 70% of the total LoM reclamation costs (excluding the contingency). Both the closure plan and reclamation costs are currently being updated to include the Phase II facilities. The plan and financial guarantee have not get been approved by the Ministry of Natural Resources and Wildlife, but could be up to 150% of the previous estimates.

20.1.2 Exploration

20.1.3 Mineral Resource Estimate

This report states the combined resources for the Main Pit area and Bloom Lake West. Previous NI 43-101 Technical Reports stated resources separately for each of the areas. SRK was responsible for the auditing resource estimation. The estimation was conducted with ordinary kriging and inverse distance squared for each of the domains. Composites coded as SIF could be used in any of the SIF structural domains and composites coded as IF could be used in any of the IF structural domains. The estimation was accomplished in three passes; the first two having a search distance of 200 m x 200 m x 100 m and the third having a search distance of 600 m x 600 m x 200 m. The first pass required a minimum of 2 drillholes for estimation.

The resources were categorized as Measured, Indicated or Inferred using the following criteria:

- Measured: Minimum of 3 drillholes with the closest composite within 80 m of the block centroid and estimated in Pass 1;
- Indicated: Minimum of 2 drillholes with the closest composite within 200 m of the block centroid and estimated in Pass 1; and
- Inferred: Blocks estimated in Pass 2 or 3.

The classification meets CIM guidelines for classification of resources.

20.1.4 Mineral Reserve Estimate

- This report states the combined reserves for the Main Pit area and Bloom Lake West. SRK was responsible for the auditing reserve estimation;

- Due to the increase in strip ratio and mill feed, SRK recalculated the mine fleet to use larger mining equipment. An SRK fleet estimation study suggested that 300-tonne trucks would reduce mining costs and improve production;
- Bloom Lake has the opportunity to adjust to different market conditions based on ore availability. This flexibility allows the site to handle swings in commodity prices; and
- The classification meets CIM guidelines for classification of reserves.

20.1.5 Mineral Processing

- The Phase I concentrator was commissioned during March, 2010;
- The Phase II concentrator plant is currently under construction and scheduled for completion during 2013. The Phase II concentrator will incorporate improvements in equipment selection and sizing based on pilot plant testing and experience gained from the operation of the Phase I concentrator;
- The main difference between the two concentrators is the incorporation of hydrosizers in the Phase II concentrator flowsheet, which will be used, in conjunction with spiral concentrators, to produce an iron gravity concentrate;
- The Phase I concentrator has not yet achieved design production rates or iron recovery projections. During 2012 the plant has averaged 2,033 t/h versus a design production rate of 2,372 t/h. Iron recovery has averaged 72% and concentrate weight percent recovery has averaged 34.7% into final iron concentrates that averaged 66.2% Fe and 4.6% SiO_2;
- It does not appear that the original comminution studies were conducted on samples that represented the extent of the variability that may occur in the deposit. The samples used to for the comminution studies reported by BBA in 2008 were from mini-bulk composites 10 and 11, taken from surface outcrops from the west zone of the ore deposit. Some additional SMC (SAG mill comminution) testing appears to have been done on mini-bulk composites 4 and 8, also from surface outcrop samples. It should be noted that there is the potential of the ore being harder in the eastern zones of the ore deposit due to the higher levels of magnetite. In order to obtain a better understanding of the variability of the ore hardness, SRK would recommend that additional comminution testing be done on selected drill core from throughout the deposit. This will enable a better understanding of throughput capacity achievable with the AG mills installed in both concentrators;
- Iron recovery in the Phase I concentrator has averaged about 72% which is about 12.5% below the original iron recovery target. It is unlikely that iron recovery can be significantly improved without major equipment modifications in the Phase I concentrator. In order to predict iron recovery from the Phase I concentrator, SRK has applied a 12.5% iron recovery discount to the iron recovery equation that was originally used for Phase I;
- The Phase II concentrator has been designed with the same size AG mill that was installed in the Phase I concentrator. However, the Phase II concentrator is projected to process 2,600 t/h. Given that the Phase II concentrator will be processing the same ore as the Phase I concentrator, it is very unlikely that a throughput capacity of 2,600 t/h can be achieved. Phase II concentrator throughput, however, may be incrementally better than the Phase I concentrator due to:
  - o Provision for better blending of the ore prior being fed to the AG mill; and

  - o Installation of hydrosizers in combination with spiral concentrators, which may serve to off-load of portion of the ore at a coarser size split.
- SRK believes that the Phase II concentrator throughput capacity may likely be in the range of 2,200 to 2,400 t/h; and
- Phase I concentrator operating costs for 2012 (January to September) averaged US$18.94 per ton of concentrate, which is equivalent to US$6.20/t of plant feed. It is likely that Phase II concentrator operating costs will be similar.

20.2 Significant Risks and Foreseeable Outcomes

20.2.1 Environmental and Permitting

No Significant Risks. The fact that the Ministry of Sustainable Development, Environment and Parks (Québec) has approved the Phase II expansion project in principle, and simultaneously granted permission to construct the associated facilities, effectively mitigates the risk that the operational authorization will not be granted. The releases to the environment of site water containing elevated suspended solids may become a chronic issue; however, mine has not been in operation long enough to work out some of the environmental management issues that typically plague larger operations in wet climates. At this time, the issue of continued unauthorized discharges does not appear to be a material risk to the project.

20.2.2 Exploration

There are no significant risks associated with the exploration conducted to date. Additional drilling to decrease the drill spacing could increase the confidence in the resource and move Indicated to Measured status and Inferred to Indicated.

20.2.3 Mineral Resource Estimate

There are no significant risks associated with the resource estimation.

20.2.4 Mineral Reserve Estimate

The lack of geotechnical pit slope parameters could increase the strip ratio, increase the mining equipment CAPEX if the pit slope study shows that the overall angles need to be shallower. The major impact of the mineral reserves could be higher strip ratios. Although this is a high risk, the economics of Bloom Lake are very robust.

Waste dump location may need to be moved if more mineralized areas are found under the waste dumps. To remove this risk, the site is planning an area wide condemnation drilling to sterilize areas where major infrastructure may exist.

20.2.5 Metallurgy and Processing

- It is unlikely that current ore tonnage throughput and iron recovery in the Phase I concentrator will significantly improve without appropriate modifications to both the comminution circuit and to the gravity concentration circuit.
- It is unlikely that the Phase II concentrator will achieve the design throughput of 2,600 t/h. A more reasonable throughput capacity may be in the range of 2,200 t/h.

- It is unlikely that the Phase II concentrator will achieve the design iron recovery of 84%. Although iron recovery will be better than currently experienced in the Phase I concentrator, it is likely that Phase II concentrator iron recoveries will be in the range of 78 to 80%.

20.2.6 Projected Economic Outcomes

Based on current iron ore prices, the Bloom Lake economic outlook is considered robust.

21 Recommendations

21.1 Recommended Work Programs

21.1.1 Environmental

The current update of the closure and reclamation plan, *Plan de Restauration – Sec Mine De Fer Du Lac Bloom* (AMEC, 2012), acknowledges that a pit lake will likely form post closure; however, relatively little other information is provided. It is recommended that Cliffs initiate the requisite hydro-geochemistry investigations and modeling efforts to better understand the long-term condition and quality of the future pit lake, and whether it may pose a risk to neighboring water resources.

21.1.2 Exploration

SRK recommends further drilling to refine the structural model and to convert Inferred Resources to Indicated. This would be an optional program if Bloom Lake wants to increase the resource and reserve and SRK has not put a detailed cost estimate on the program.

21.1.3 Mining Reserves

For Mineral Reserves, SRK recommends a pit slope stability study to be performed and concluded by the end of 2012. This is necessary to validate the pit design criteria used in the reserve statement. Once the pit slope stability study is completed, review the mine plan to ensure all assumptions and design support the pit slope stability findings.

21.1.4 Processing

- In order to obtain a better understanding of the variability of the ore hardness, SRK would recommend that additional comminution testing be done on selected drill core from throughout the deposit. This will enable a better understanding of throughput capacity achievable with the existing AG mills.
- Secondary comminution circuits should be evaluated for both the Phase I and Phase II concentrators after commissioning of the Phase II concentrator has been completed and actual concentrator performance has been confirmed. The options that should be investigated include:
  - Pebble crushing;
  - Secondary ball milling; and
  - Conversion of the AG mills to SAG mills.

22 References

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23 Glossary

23.1 Mineral Resources

The mineral resources and mineral reserves have been classified according to the “CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines” (November 27, 2010). Accordingly, the Resources have been classified as Measured, Indicated or Inferred, the Reserves have been classified as Proven, and Probable based on the Measured and Indicated Resources as defined below.

A Mineral Resource is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth’s crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

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 through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes.

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 locations such as outcrops, trenches, pits, workings and drillholes 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 drillholes that are spaced closely enough to confirm both geological and grade continuity.

23.2 Mineral Reserves

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

A ‘Probable Mineral Reserve’ is the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility

Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

A 'Proven Mineral Reserve' is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

23.3 Glossary

The following general mining terms may be used in this report.

Table 26.3.1: Glossary

Term	Definition
Assay:	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure:	All other expenditures not classified as operating costs.
Composite:	Combining more than one sample result to give an average result over a larger distance.
Concentrate:	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.
Crushing:	Initial process of reducing ore particle size to render it more amenable for further processing.
Cut-off Grade (CoG):	The grade of mineralized rock, which determines as to whether or not it is economic to recover its gold content by further concentration.
Dilution:	Waste, which is unavoidably mined with ore.
Dip:	Angle of inclination of a geological feature/rock from the horizontal.
Fault:	The surface of a fracture along which movement has occurred.
Ferric	Refers to iron-containing materials or compounds. In chemistry the term is reserved for iron with an oxidation number of +3, also denoted iron(III) or Fe^{3+}
Ferrous	Refers to iron with oxidation number of +2, denoted iron(II) or Fe^{2+}.
Footwall:	The underlying side of an orebody or stope.
Gangue:	Non-valuable components of the ore.
Grade:	The measure of concentration of gold within mineralized rock.
Hangingwall:	The overlying side of an orebody or slope.
Haulage:	A horizontal underground excavation which is used to transport mined ore.
Hydrocyclone:	A process whereby material is graded according to size by exploiting centrifugal forces of particulate materials.
Igneous:	Primary crystalline rock formed by the solidification of magma.
Kriging:	An interpolation method of assigning values from samples to blocks that minimizes the estimation error.
Level:	Horizontal tunnel the primary purpose is the transportation of personnel and materials.
Lithological:	Geological description pertaining to different rock types.
LoM Plans:	Life-of-Mine plans.
LRP:	Long Range Plan.
Material Properties:	Mine properties.
Milling:	A general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral/Mining Lease:	A lease area for which mineral rights are held.
Mining Assets:	The Material Properties and Significant Exploration Properties.
Ongoing Capital:	Capital estimates of a routine nature, which is necessary for sustaining operations.
Ore Reserve:	See Mineral Reserve.
Pillar:	Rock left behind to help support the excavations in an underground mine.
RoM:	Run-of-Mine.

Term	Definition
Sedimentary:	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Shaft:	An opening cut downwards from the surface for transporting personnel, equipment, supplies, ore and waste.
Sill:	A thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness.
Smelting:	A high temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from the gangue components that accumulate in a less dense molten slag phase.
Stope:	Underground void created by mining.
Stratigraphy:	The study of stratified rocks in terms of time and space.
Strike:	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulfide:	A sulfur bearing mineral.
Tailings:	Finely ground waste rock from which valuable minerals or metals have been extracted.
Thickening:	The process of concentrating solid particles in suspension.
Total Expenditure:	All expenditures including those of an operating and capital nature.
Variogram:	A statistical representation of the characteristics (usually grade).

23.4 Abbreviations

The following abbreviations may be used in this report.

Table 26.4.1: Abbreviations

Abbreviation	Unit or Term
%	percent
°	degree (degrees)
°C	degrees Centigrade
°F	Degrees Fahrenheit
°F	Degrees Fahrenheit
µm	micron
Al_2O_3	alumina
AMP	amphibolite
ANFO	ammonium nitrate fuel oil
APSI	Administration Portuaire de Sept-Îles (Sept-Îles Port Authority)
BTU	British thermal unit
CAN$	Canadian dollars
CaO	calcium oxide
Capex	capital expenditure
CEAA	Canadian Environmental Assessment Agency
cfm	cubic feet per minute
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CLM	Consolidated Thompson Iron Ore Mines Limited
cm	centimeter
cm^2	square centimeter
cm^3	cubic centimeter
CoG	cut-off grade
Cr_2O_3	chromium oxide
dia.	diameter
Dt/h	dry tonnes per hour
EIA	Environmental Impact Assessment
Fe	iron
$Fe^{2+}Fe_2^{3+}O_4$	magnetite
$Fe_2^{3+}O_3$	hematite

FOB	Freight on Board
g	gram
g/L	gram per liter
g/t	grams per tonne
GHG	greenhouse gas
gpm	gallons per minute
ha	hectares
hp	horsepower
ID2	inverse-distance squared
ID3	inverse-distance cubed
IF	iron formation
in	inches
IRR	internal rate of return
K_2O	potassium oxide
kA	kiloamperes
kg	kilograms
km	kilometer
km/h	kilometers per hour
km^2	square kilometer
kt	thousand tonnes
kt/d	thousand tonnes per day
kt/y	thousand tonnes per year
kV	kilovolt
kW	kilowatt
kWh	kilowatt-hour
kWh/t	kilowatt-hour per metric tonne
L	liter
L/sec	liters per second
L/sec/m	liters per second per meter
lb	pound
LOI	loss on ignition
LoM	Life-of-Mine
m	meter
M	million
m.y.	million years
m^2	square meter
m^3	cubic meter
m^3/h	cubic meters per hour
Mag	magnetite
MagFe	percent iron in magnetite
masl	meters above sea level
MDDEP	Ministère du Développement Durable de l'Environnement et des Ressources
MgO	magnesium oxide
min	minute
mm	millimeter
mm^2	square millimeter
mm^3	cubic millimeter
MnO	manganese oxide
MRN	Ministère des Ressources Naturelles
MS	Mica schist
Mt	million tonnes
Mt/y	million tonnes per year
MW	million watts
NAD	North American Datum
NaO	sodium oxide
NI 43-101	Canadian National Instrument 43-101
NPV	net present value

NTS	National Topographic System
OIF	Oxide iron formation
Opex	Operating cost
P_2O_5	phosphorous oxide
PLC	Programmable Logic Controller
PMF	probable maximum flood
ppb	parts per billion
ppm	parts per million
QA/QC	Quality Assurance/Quality Control
QCM	Quebec Cartier Mining Company
QNS&L	Quebec North Shore & Labrador Railway
QR	Quartz Rock
QRMS	Quartz rock mica schist
RoM	Run-of-Mine
RQD	Rock Quality Designation
SEC	U.S. Securities & Exchange Commission
sec	second
SG	specific gravity
SIF	silicate iron formation
SiO_2	silica
SR	stripping ratio
t	tonne (metric ton) (2,204.6 pounds)
t/d	tonnes per day
t/h	tonnes per hour
t/m^3	tonnes per cubic meter
t/y	tonnes per year
TFe	Total iron
TiO_2	titanium oxide
TSF	tailings storage facility
US$	United States Dollar
UTM	Universal Transverse Mercator
V	volts
V_2O_5	vanadium oxide
VFD	variable frequency drive
W	watt
WGM	Watts, Griffis & McOuat Limited
WLIMS	Wet, low-intensity magnetic separator
WRA	Whole rock analysis
WREC	weight recovery percentage
WtR	Weight recovery
XRD	x-ray diffraction
XRF	x-ray fluorescence
y	year

Appendices

Appendix A: Mining Claims

Bloom Lake General Partner Limited
Listing of Bloom Lake Property Claims and Mining Lease
As of January 8, 2013

Property	Type	Claim #	Area (ha)	Amount Work Paid	Amount Rent Paid	Expiry Date	Record Name	
SNRC 23B14	BM	877	6,857.6			April 13, 2029	Bloom Lake General Partner Limited (84111) 100 %	mining lease
SNRC 23B14	CDC	98977	9.8	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	98978	0.9	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	98986	10.7	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	98994	11.0	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	98995	27.4	$800	$98	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99884	0.2	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99885	7.0	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99886	9.9	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99887	19.2	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99888	47.8	$800	$109	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99889	35.9	$800	$98	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99890	21.2	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99891	14.6	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99892	0.2	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99894	24.6	$800	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99895	21.3	$800	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99896	18.0	$800	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99897	26.5	$320	$98	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99898	12.5	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99902	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99903	51.6	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99904	4.8	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99905	3.0	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99910	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99911	51.5	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99918	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99919	51.5	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99935	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99936	45.2	$900	$109	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99937	31.6	$800	$98	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99938	18.0	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99939	4.6	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99951	24.7	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99952	39.6	$320	$98	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99953	26.8	$320	$98	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99954	0.6	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99956	3.2	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	
SNRC 23B14	CDC	99957	13.7	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %	

SNRC 23B14	CDC	99965	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	99966	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	99967	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	99968	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	99969	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	99970	51.8	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	99971	42.9	$800	$98	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	99972	26.4	$800	$98	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	1133844	52.0	$1,350	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	1133845	52.0	$1,350	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	1133846	52.0	$1,350	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	1133847	52.0	$1,350	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082920	5.1	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082921	13.4	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082922	50.5	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082923	52.1	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082925	1.7	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082926	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082927	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082928	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082929	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082930	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082931	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082932	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082933	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082934	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082935	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082936	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082937	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082938	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082939	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082940	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082941	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082942	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082943	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082944	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082945	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082946	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082947	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082948	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082949	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082950	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082951	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %

SNRC 23B14	CDC	2082952	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082953	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082954	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082955	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082956	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082957	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082958	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082959	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082960	49.0	$900	$109	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082961	14.6	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082962	49.4	$900	$109	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082963	52.1	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082964	50.5	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082965	1.7	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082966	39.5	$800	$98	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082967	12.6	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082968	23.3	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082969	24.8	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082970	0.0	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082971	36.5	$800	$98	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082972	4.9	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082973	37.2	$800	$98	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082974	18.9	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082975	24.5	$320	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082976	31.3	$800	$98	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082977	28.0	$800	$98	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082978	35.1	$800	$98	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082979	35.2	$800	$98	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082980	36.9	$800	$98	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2082981	52.0	$900	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2177003	13.8	$160	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2183070	0.1	$160	$27	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
SNRC 23B14	CDC	2188096	52.0	$135	$123	January 9, 2015	Bloom Lake General Partner Limited (84111) 100 %
Totals	**Claims**	**114**	**4,101.3**	**$80,155**	**$10,014**		
	Permit	**1**	**6,857.6**				
			10,958.9				