Feasibility Study Technical Report - Update

Scully Mine Re-start Project Wabush, Newfoundland & Labrador, Canada

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Feasibility Study Technical Report - Update – Scully Mine Re-start Project

Revision #

Wabush, Newfoundland & Labrador, Canada

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1 SUMMARY

1.1 Summary of Average Life-of-Mine Metrics

Table 1.1: Summary of Average Life-of-Mine Metrics

Parameter	Unit	Value
Economic Assumptions		
Benchmark 62% Fe – CFR China-Long Term	USD/dmt con.	70
Benchmark 65% Fe – CFR China-Long Term	USD/dmt con.	85
Concentrate Adjusted Price-Premium - Marketing	USD/dmt con.	82
Exchange Rate	CAD/USD	1.25
Mining & Processing		
Total Tonnage Mined	Mt	868.8
Ore Tonnage Mined	Mt	477.9
Strip Ratio	w:o	0.82
Mining Dilution	%	5.2
Mining Recovery	%	97
Maximum Annual Mining Rate	Mtpy	36.0
Average Annual Processing Rate	Mtpy	18.0
Initial Ore Stockpile Tonnage	Mt	1.0
Weight Recovery	%	33.22
Average Fe Grade in feed	%	34.88
Concentrate Tonnage Production	Mt	159.1
Average Annual Concentrate Tonnage Production	Mtpy	6.0
Concentrate Fe Grade	%	65.6%
Cost Parameters - LOM		
Mining	CAD/dmt con.	14.74
Processing plant	CAD/dmt con.	14.31
General and administration	CAD/dmt con.	2.61
Cash Cost FOB Mine	**CAD/dmt con.**	**31.66**
Concentrate transportation & handling	CAD/dmt con.	20.27
Cash Cost FOB Pointe Noire	**CAD/dmt con.**	**51.93**

Parameter	Unit	Value
Royalties	CAD/dmt con.	6.94
Cash Cost FOB Pointe Noire including Royalties	**CAD/dmt con.**	**58.87**
Economic Results		
Upgrade Capital	M CAD	95.03
Sustaining Capital	M CAD	618.23
After-Tax NPV @ 0%	M CAD	2,612
After-Tax NPV @ 8%	M CAD	1,345
After-Tax NPV @ 10%	M CAD	1,205

1.2 Introduction

The Scully Mine was operated by Picklands Mather & Company (PM), from 1965 to 1986 when PM was acquired by Cleveland-Cliffs Inc. ("Cliffs") who operated it from 1986 until being shut down for economic reasons in February 2014. For most of its life, the mine was a joint venture owned by Stelco (37.9%), Dofasco (24.3%), Inland Steel (15.1%), Acme Steel (15.1%) and Cliffs (7.7%) but after various mergers and acquisitions in the North American steel industry, the ownership was consolidated between Cliffs, ArcelorMittal and U.S. Steel Canada whereby each company respectively owned a joint venture percent ownership of 26.8%, 28.6% and 44.6%. Cliffs exercised their right of first refusal in February 2010 to acquire 100% ownership of the property for approximately USD 88 million.

The Scully Mine and associated pellet plant located at Pointe Noire under Cliffs had the capacity of producing 6 million tonnes of iron ore pellets per year. An integrated rail system was used to move the iron ore concentrate product to the pelletizer, located at Pointe-Noire near Sept-Iles, Quebec utilizing a bottom dump unloading system. From there, the product could be shipped via boat to clients in America or elsewhere on the seaborne market. The product produced from the Scully Mine contained higher than normal levels of manganese due to the geology of the deposit. The Scully Mine's integrated mine and pellet plant facilities produced two (2) types of iron ore pellets with varying manganese levels as controlled only by the ore blends as the concentrating process was unable to reduce the manganese content in the ore.

Cliffs shutdown the pellet plant in May 2013 followed by the mine and concentrator in February 2014. The closure was due to increased costs, reduced production rates and a dramatic decrease in seaborne iron ore prices combined with a decrease on pellet premium pricing. On November 19, 2014 Cliffs announced the pursuit of exit options from its Eastern Canadian iron ore operations, and on May 20, 2015, Cliffs commenced restructuring proceedings under Companies' Creditors Arrangement Act (Canada) ("CCAA"), ultimately selling the Scully Mine and other Eastern Canadian assets. The Scully Mine was sold to Tacora in July of 2017. The asset purchase agreement between Tacora and Wabush Iron Co. Limited, Wabush Resources Inc. and Wabush Lake Railway Company Limited was executed through a court supervised process under the CCAA.

Tacora Resources Inc. ("Tacora") restarted production of the Scully Mine in June of 2019. This report contains the information pertaining to Scully ongoing ramp up and continuing operations adhering to the standards and reporting regulations of the National Instrument 43-101 - Standards of Disclosure for Mineral Projects ("NI 43-101").

Tacora has undertaken certain steps to restart the mine and significantly improve the cost structure. Steps by Tacora include:

- The negotiation and signing of a new Collective Bargaining Agreement ("CBA") with the United Steelworkers ("USW") union, incorporating high performance work systems including a significant amount of variable compensation.
- Execution of a 15-year (13 years remaining) iron ore concentrate purchase contract off-take agreement with a subsidiary of Cargill.
- The negotiation of a comprehensive amendment to the mineral lease for the Scully Mine ore body.
- Direct sale of concentrate instead of pellets to remove any constraints that historically came from pellet production.
- The execution of a new life of mine rail transportation agreement with the Quebec North Shore and Labrador Railway ("QNS&L") railway, owned by Iron Ore Company of Canada ("IOC"), for the transportation of product from Wabush to Sept-Ile.

The remaining resources at the Scully Mine consist of high-manganese iron ore located above, below and around previous operations. Four additional manganese reduction lines using modern rare earth magnetic separators have been installed to supplement the two existing manganese reduction lines to reduce the manganese content to less than 2% and silica to less than 3% in the finished product. The concentrator annual production rate is planned to ramp-up to about 6 Mt per year of iron ore concentrate at 65.6% iron content.

1.3 Access, Local Resources and Infrastructure

The mine site lies 3 km west of the town of Wabush and 3.5 km south of Labrador City. A 4-km road connects the site to Highway 500 and Highway 389 which is accessible by road from Baie-Comeau on the north shore of the St. Lawrence River. The Wabush airport is 3 km from the mine site within the city limits of Wabush. The Scully Mine is 950 km North East of Montreal by plane or 1,250 km by highway.

The rail access to the port consists of two separate segments. The first segment uses QNS&L railway from Wabush to Arnaud Junction in Sept-Îles and from there, the second section is from Arnaud Junction to Pointe-Noire (Sept-Îles), property of "Les Chemins de Fer Arnaud", where the concentrate is unloaded, stockpiled, and loaded on vessels. The second segment is owned by the Government of Quebec through the Sociéte du Plan Nord, which acquired these assets from Cliff's bankruptcy of Canadian assets, originally Wabush Railway Company Limited.

The towns of Wabush and Labrador City are well established with populations of 1,905 (2016) and 7,220 (2016), respectively. The unemployment rate of Wabush and Labrador City are of 11.1% and 8.5% respectively with an unemployment rate of 13.7% across the province of Newfoundland and Labrador; these statistics are taken from the 2016 Census from Statistics Canada. The two (2) towns are located 5 km apart from one another and they contain the infrastructures and necessities to house the employees and their families who live there including, indoor shopping centers, hotels, lower, middle and high school, a community center and a hospital.

The mine site is connected to the Newfoundland & Labrador Hydro electrical network. Power is generated at Churchill Falls, 200 km to the East. The Churchill power station has the second largest hydroelectric generating capacity in North America at 5,428 MW installed. A 46-kV electrical grid on site electrifies the mine area powering the process plant, mine equipment and pumping stations.

The site includes: mine electrical infrastructure; a maintenance facility with 5 bays and cranes; warehouses; wash bay; explosive storage; machine shop; dewatering equipment; fuel storage; administration buildings; a concentrator plant; and required rail load-out and track infrastructure. The buildings are in good condition to support operations. The concentrator underwent some upgrade prior to restart in 2019 with the installation of additional processing equipment.

1.4 Geology, Drilling and Sampling Methodologies

The Scully Mine lies within the Labrador Trough in Western Labrador. The Sokoman Formation is an iron formation that consists of three iron bearing formations, named the Upper, Middle and Lower Iron Formations. The Sokoman Formation is more than 300 m thick near the Scully Mine and has been subjected to two episodes of folding and metamorphism during the Hudsonian and Greenville orogenies, resulting in a complex structural pattern in the Wabush Area. The younger Menihek and Shabagamo Formations and the older Denault, Attikamagen, and Wishart Formations all outcrop in the vicinity of the mine site. The mineral deposit that defines the Scully Mine consists of folded and faulted stratigraphic beds of iron-bearing units within the regional Sokoman Iron Formation.

The ore minerals are hematite (specularite), magnetite, and martite. The waste minerals are quartz and hydrated iron oxides such as limonite and goethite. Manganese oxides also occur in bands or are disseminated throughout the iron-bearing units.

The geological interpretation for the Scully deposit is based primarily on diamond drilling data and 2D sectional interpretations by representatives of Cliffs in 2014. The geology of the deposit is well understood.

The drilling database contains historical data with records beginning in 1969 and continuing up to 2013. Data was recorded in the local coordinate system in feet and was subsequently converted to meters by GMS. Limited metadata exists pertaining to the drilling database however extensive drill logs are available in pdf format.

Sampling and logging procedures relating to exploration drilling and blasthole drilling were available for GMS to review. Details on analytical methods were detailed in procedures and shaking table test methods were reviewed. Due to the disparate nature of documents and data on site, GMS was unable to find details on QAQC protocols, and we believe that no QAQC database was maintained. GMS embarked on an independent re-assaying program to confirm the validity of the assay database, for which the results were satisfactory.

Mr. Réjean Sirois, P.Eng from G Mining Services Inc. conducted a site visit between August 17th and 18th, 2017 to undertake data verification and independent resampling of drill core.

1.5 Mineral Resource Estimate

G Mining Services Inc. ("GMS") was mandated by Tacora to produce the Mineral Resource estimate for the Tacora Mine Project. The Mineral Resource estimate was prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Standards for Mineral Resource and Mineral Reserves (2014) as incorporated in NI 43- 101. The Scully Mine Mineral Resource presented herein was prepared by Mr. Réjean Sirois, P.Eng., and James Purchase, P.Geo from GMS and has an effective date of February 15th, 2021. Messrs. Sirois and Purchase are independent "Qualified Persons" as defined in NI 43-101.

The 2021 Mineral Resource Estimate ("MRE") of the Scully Deposit uses the same exploration drilling database and generally follows the same methodology as applied in the previous Mineral Resource published in 2018. The key additions have been a reconciliation exercise to review the performance of the 2018 block model against production since mid-2019. In addition, the following changes were made:

- Incorporation of 2019 and 2020 blastholes into the blasthole database.
- Modifications to the weighted average grade scripts to apply tonnage weightings rather than volume weightings. In addition, overburden blocks were removed from the weighted average script to avoid diluting surface blocks with overburden (which is stripped pre-mining).
- Mining depletion was accounted for in the block model, by adjusting block percentages to account for the end of year 2020 pit survey.
- Pit optimisations were run with updated assumptions, and were used to constrain the Mineral Resource.

The geological interpretation remains identical to the 2018 block model, and grade estimation parameters are unchanged.

Drilling and blasthole data were initially converted from feet into meters, and validated for erroneous surveys and collars, missing intervals and out-of-range values. Geological wireframes for each geological unit were constructed using the drill hole intercepts, and the 2D sectional interpretations by site geologists. Fault blocks were also modelled, and structural domains were interpreted according to the dip and dip direction of each fold limb. Drill hole assay data was composited to 6 m run-lengths, using the geological units as boundaries. Various statistical analyses were undertaken on the assay attributes of the composites, in addition to variography for each Member (Lower, Middle and Upper). Block modelling was undertaken using a percentage-style model, with a block size of 20 m x 20 m x 12 m. Crude assay attributes for Fe, Mn and Satmagan (magnetite content %) were estimated using ordinary kriging, with weight recovery and concentrate attributes (Fe conc., Si Conc., Mn Conc., Satmagan Conc.) estimated using Inverse Distance Squared. Kriging was undertaken using four estimation passes, with Inverse Distance interpolation using a single-pass. The resulting block model was visually validated against the composites on a section-by-section basis, and by using statistical validations such as domain-wise descriptive statistics, swath plots and quantile plots (Q:Q plots). The estimation was also compared to blasthole data to ensure reproducibility of the blasthole grades. Bulk density was determined using a regression curve based on crude Fe % and confirmed by laboratory density determinations undertaken as part of the resampling program in 2017.

The Mineral Resource estimate was classified into Measured, Indicated and Inferred categories based on estimation pass, and the classification is in accordance with the CIM Definition Standards on Mineral Resources and Mineral Reserves.

The Measured and Indicated Mineral Resource for the Scully Deposit is estimated at 721.9 Mt at an average grade of 34.7% Fe, and the Inferred Mineral Resource at 263.4 Mt at an average grade of 34.1% Fe. Table 1.2 presents the resource estimation tabulation by category. The Mineral resources are reported inclusive of the Mineral Reserves.

The Mineral Resources are reported within the conceptual open pit shell at a cut-off value of 10% WTREC (Wilfley Table Wet Weight Recovery). The conceptual pit shell represents potentially extractable Mineral Resources in the Measured, Indicated and Inferred categories using an iron ore concentrate price of USD 93.5/dmt CFR Client (14% higher than the reserve price assumption). Key infrastructure (rail loop, processing plant) and lakes were considered by incorporating a mining exclusion zone into the pit optimization process. The end-of-year 2020 mining topography was used as the mining depletion surface. Lastly, only rock codes 22, 31, 32, 33, 34, 51, 52 and 53 were included in the Mineral Resource.

Table 1.2: 2021 Mineral Resource Estimate for the Scully Deposit at a 10% WTREC Lower Cut-Off

Classification	Tonnage (dry)	Fe	Mn	SAT	WTREC	Fe Conc.	SiO_2 Conc.	Mn Conc.	Sat Conc.
	Mt	%	%	%	%	%	%	%	%
Measured	195.6	35.1	2.3	5.9	36.6	64.6	3.7	2.0	7.1
Indicated	526.3	34.5	2.4	6.1	35.5	63.9	3.9	2.5	8.2
Total M&I	**721.9**	**34.7**	**2.4**	**6.0**	**35.8**	**64.1**	**3.8**	**2.4**	**7.9**
Inferred	263.4	34.1	2.1	6.2	34.0	64.2	3.9	2.1	9.1

Notes on Mineral Resources:

1. *The Mineral Resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards for Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council May 10th, 2014.*
2. *The independent and qualified persons for the 2021 Scully resource estimate, as defined by NI 43-101, are Mr. Réjean Sirois, P. Eng., and Mr. James Purchase, P.Geo from G Mining Services Inc. The effective date of the estimate is February 15th, 2021.*
3. *The Mineral Resources are reported at a lower cut-off grade of 10% WTREC, within rock codes 22, 31, 32, 33, 34, 51, 52 and 53 only.*
4. *The Mineral Resources have been depleted using a mining surface representing the end-of-year 2020.*
5. *The Mineral Resources are reported within an optimized Whittle shell using a long-term iron price of USD 93.5/dmt conc. with an exchange rate of 1.25 USD/CAD. Measured, Indicated, and Inferred categories are considered. Manual manipulation was undertaken to remove isolated partial blocks that do not meet REEE (Reasonable Prospects of Eventual Economic Extraction).*
6. *The Mineral Resources are reported inclusive of the Mineral Reserves.*
7. *"SAT" stands for Satmagan or Saturation Magnetization Analyzer, an instrument which estimates magnetite content in samples.*
8. *"Conc." stands for concentrate grades obtained from shaker table test work.*
9. *"WTREC" stands for Wilfley shaking table recoveries (wet process) under laboratory conditions.*
10. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Mineral Reserves.*
11. *The number of metric tonnes was rounded to the nearest hundred thousand. Any discrepancies in the totals are due to rounding effects.*
12. *The weight recoveries and concentrate grades stated in the table above are derived from the Wilfley shaking table process. These values are only indicative of potential concentrate recovery and quality under laboratory conditions, and no modifying factors have been applied to compensate for the scaling up of the recovery process to industrial levels.*

1.6 Mineral Reserves

The Mineral Reserve for the Tacora Mine is estimated at 478.9 Mt at an average grade of 34.9% Fe and 2.62% Mn as summarized in Table 1.3. The Mineral Reserve estimate was prepared by GMS. The resource block model was also generated by GMS.

The mine design and Mineral Reserve estimate have been completed to a level appropriate for feasibility studies. The Mineral Reserve estimate stated herein is consistent with the CIM definitions and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated ("M&I") Mineral Resources, and do not include any Inferred Mineral Resources. The Inferred Mineral Resources contained within the mine design are treated as waste.

Table 1.3: Tacora Mine Mineral Reserve Estimate (Effective Date of January 1st, 2021)

Mineral Reserves by Category		Proven	Probable	Stockpile	Proven & Probable
Crude Ore Tonnage	k dmt	341,439	136,508	997	478,943
Crude Iron Grade	% Fe	34.85	34.97	38.41	34.89
Crude Manganese Grade	% Mn	2.72	2.35	5.31	2.62
Concentrate Tonnage	k dmt	113,577	45,478	369	159,425
Concentrate Iron Grade	% Fe	65.60	65.60	65.30	65.60
Concentrate Manganese Grade	% Mn	1.63	1.53	5.92	1.61
Concentrate Silica Grade	% SiO_2	3.06	3.22	3.00	3.11
Total Weight Recovery	%	33.26	33.32	37.02	33.29
Total Fe Recovery	%	62.62	62.49	62.94	62.59

Notes on Mineral Reserves:

1. *The Mineral Reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards for Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council May 10th, 2014.*
2. *Mineral Reserves based on December 2020 depletion surface merged with an updated Lidar dated September 2017.*
3. *Mineral Reserves are estimated at a minimum of 20% Lab weight recovery for all sub-units except sub-unit 52 which is 30%. In addition, sub-unit 34 must have a ratio of weight recovery to iron of at least 1.*
4. *Mineral Reserves are estimated using a long-term iron price reference price (Platt's 62%) of USD 70/dmt and an exchange rate of 1.25 CAD/USD. An Fe concentrate price adjustment of USD 12/dmt was added as an iron grade premium net of a USD 5/dmt marketing charge.*
5. *Bulk density of ore is variable but averages 3.20 t/m^3.*
6. *The average strip ratio is 0.82:1.*
7. *The Mineral Reserve includes a 5.2% mining dilution and a 97% ore recovery.*

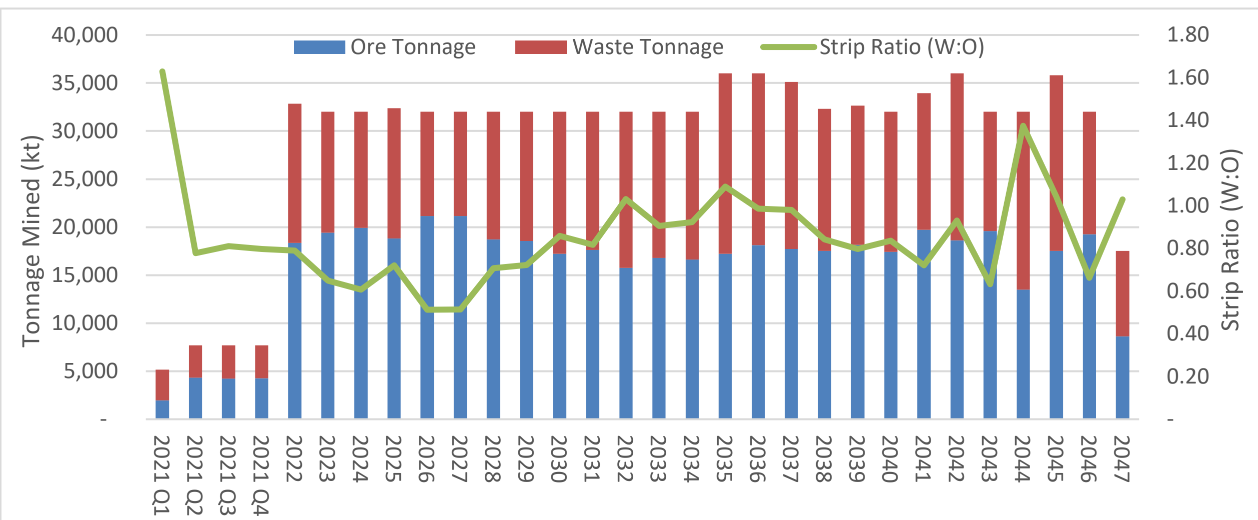
8. *The number of metric tonnes was rounded to the nearest thousand. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101.*

Previously, Golder Associates Ltd. ("Golder") was mandated by Tacora to produce a pit slope design study to support the mine design. The recommendations from this study have been used as an input to the pit optimization and design process. The Golder scope included geotechnical and hydrogeological evaluations to support the slope design criteria for the open pit. No major pit slope failures have occurred or are present, but all catch benches are filled with material.

1.7 Mining

The operation consists of a conventional surface mining method using an owner mining approach with electric and diesel hydraulic shovels and mine trucks. Some major mine equipment required for the restart of the project, such as drills and hydraulic shovels, are present on site as this equipment was acquired early on in 2017. The study consists of resizing the open pit based on parameters outlined in this study and producing a life-of-mine ("LOM") plan to fill the mill to capacity subject to constraints with a mining rate of 35 Mtpy (Figure 1.1).

Figure 1.1: Mine Production

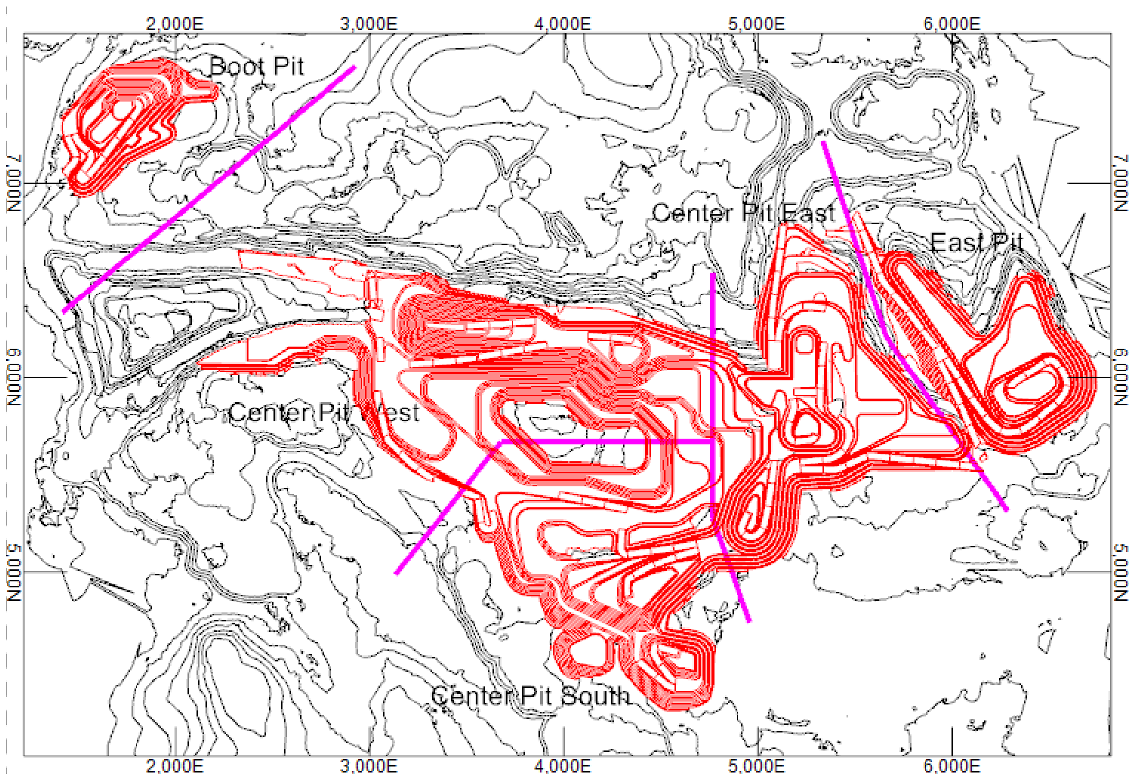


Drill and blast specifications are established to effectively single pass drill and blast a 12 m bench. For this bench height a 349 mm (13¾ in.) blast hole size is used with a 9.25 m burden by 9.25 m spacing with 1.8 m of sub-drill in ore. The blast pattern in waste is identical with adjusted explosive column height. These drill parameters, combined with a high energy bulk emulsion with a density of 1.2 kg/m^3, result in a targeted powder factor of 1.0 kg/m^3 in ore. Blast holes are initiated with electronic detonators and primed with 450 g boosters. The bulk emulsion product is a gas-sensitized pumped emulsion blend specifically designed for use in wet blasting applications.

The majority of the loading in the pit will be done by three hydraulic shovels equipped with a 24 m^3 bucket. The shovels are matched with a fleet of 211 metric tonnes payload mine trucks. The project already owns two 24 m^3 hydraulic front shovels (one electric and one diesel) and one 17m^3 diesel front shovel. The hydraulic shovels will be complemented by one production front-end wheel loader with a 21 m^3 bucket. A 13.5 m^3 wheel loader is planned, being matched with two 100t class trucks in order to cover the rehandling and complement the main fleet as required. The truck fleet reaches a maximum of 15 units excluding rehandling equipment.

Mining of the Scully Mine is planned with five pit sectors referred to as Boot Pit, Center West Pit, Center East Pit, Center South Pit and East Pit. Mining has taken place in all of these pit sectors.

Figure 1.2: Pit Layout



Mining Services	Tacora Mine Final Pit Design	Author: Peter Harquail Date: April, 2021

The pit area measures approximately 5.4 km in an east-west direction and is approximately 2.1 km north-south in relation to the South Pit. The final pit contains 478.9 Mt of ore at an average grade of 34.89% Fe, 2.62% Mn and 5.53% SAT. This Mineral Reserve is sufficient for a 27-year mine life with possibilities for expansion at higher iron ore prices and the conversion of Inferred Mineral Resources to Measured and Indicated Mineral Resources. A total of 868.8 Mt is to be mined for an overall strip ratio of 0.82:1.

The plant ramp-up period is planned in 2021 during which the processing rate will reach its targeted production rate of 6.0 Mt of concentrate per year throughout the mine life.

For this study, three distinct dewatering activities were planned. The first consists in the dewatering of the flooded pits to restart mining and to allow for in-pit dumping. The first pits to dewater are the West and South pits. The second dewatering activity to take place is the long hole dewatering. All the infrastructure is present on the western end of the pit along the Long Lake boundary. Finally, sump pumps will be installed at pit bottoms or on working benches when water is present. Water pumped in the pit, will first be sent to a sedimentation pond before being released in the surrounding lakes.

1.8 Mineral Processing and Metallurgical Testing

The Scully Mine mineral processing facility is a mature plant that has been producing from 1965 to 2014 and then from 2019 onward. No major modifications were made before restarting production other than the addition of four (4) manganese removal lines. Since the plant restarted in June 2019, some initiatives were identified to increase throughput, recovery and availability:

- Optimize the efficiency of the mill planned maintenance Improve parts availability and reduce crude end feed outages.
- Replace the mill starters and relays, improve the mill lube systems, and replace the mill gear on two (2) mills.
- Replace the mill recirculation pumps.
- Replace the crusher discharge conveyor belts.
- Add new scavenger spirals to process the hydrosizer overflow.
- Modernize mill instrumentation.
- Replace cyclones to increase mill availability when the filters are down.
- Restart the high-tension scavenger circuit that is part of the manganese removal plant to recover additional concentrate.

Based on these modifications, improved operation and maintenance practices, the targeted mill availability is 89.5%.

It is important to limit the quantity of ore from South Pit and the lower member ore in any given year, due to its hardness explained by its high magnetite content, to maintain adequate throughput. To meet the grinding throughput requirement, parameters were identified to limit the proportion of ore originating from unit 52 to a maximum of 15% and not to be blended with any other similarly hard ore from other pits.

Considering the mill throughput and mill availability observed since the plant restart, improvement opportunities were identified in order to reach the targeted concentrate production. The AG mills should be operated in such way to maximize the motor power utilization so the throughput will be maximized. This should be done by implementing a control loop to automatically adjust the feed throughput as a function of the motor power. The operations data show that increasing the power utilization to 85% and increasing the maximum allowable throughput for all mills to 450 tph would increase the average throughput to 385 tph.

1.9 Recovery Methods

For the remainder of 2021, the availability, the mill throughput, and the recovery will be gradually increased as modifications of the equipment and process are implemented. The modifications should be completed by the end of 2021 so that in 2022 onward, the concentrate production target could be met.

Table 1.4 presents the high-level Scully Mine design criteria once the modifications are completed.

Table 1.4: Scully Mine High Level Design Criteria

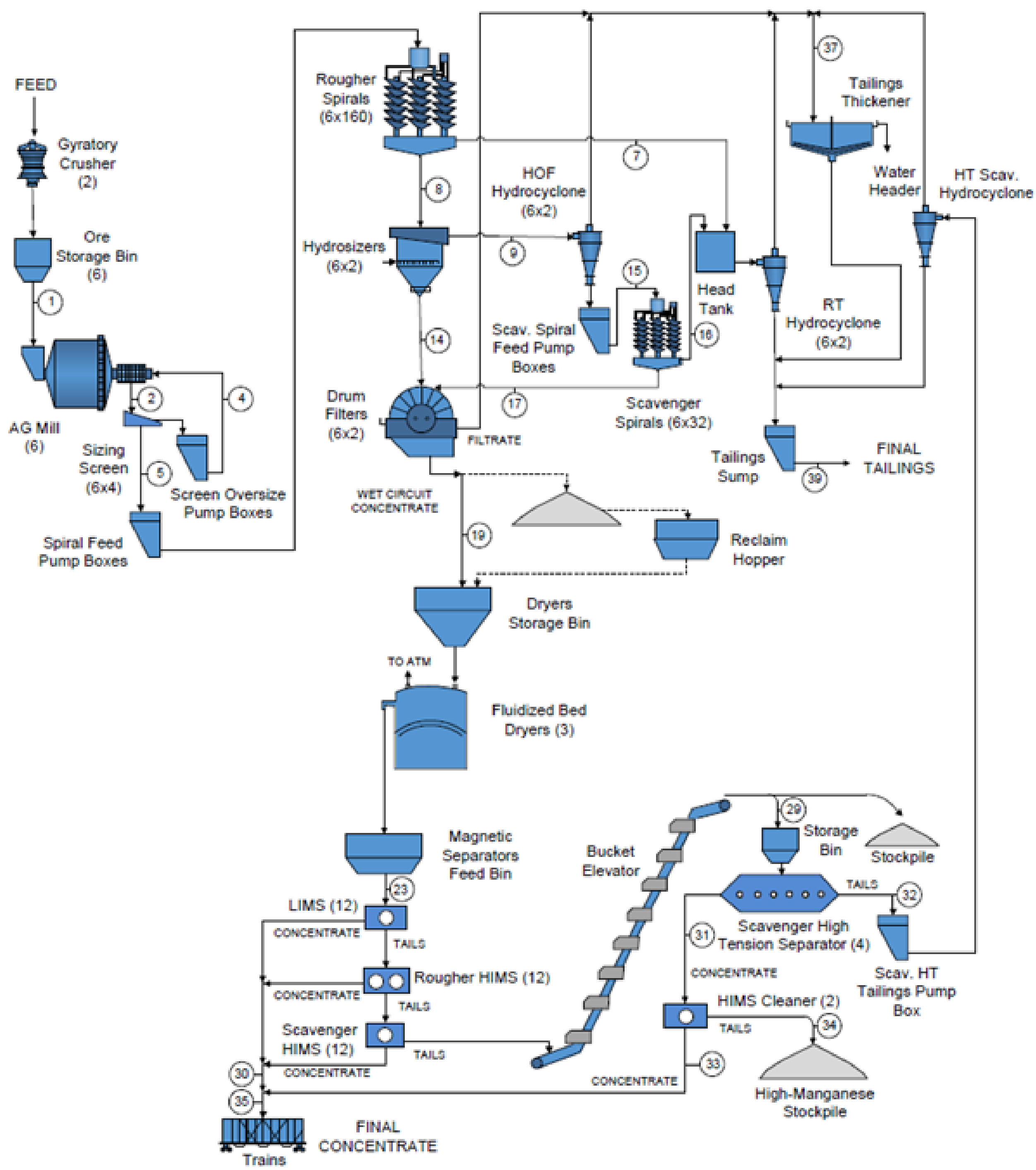
Parameter	Unit	Value	Sources
Mill Availability	%	89.5	A
Production Hours	hr/y	7,840	C
Feed Annual Tonnage	t/y	18,110,862	C
Concentrate Annual Tonnage	t/y	6,045,649	C
Fresh Feed Iron Grade	%	35.0	A
Fresh Feed Silica Grade	%	40.6	A
Fresh Feed Manganese Grade	%	2.6	A
Solids Feed Rate - Nominal per Mill	t/hr	385	B
Solids Feed Rate - Nominal Total	t/hr	2,310	C
Wet Circuit Concentrate Iron Grade	%	61.0	A
Wet Circuit Concentrate Silica Grade	%	7.2	A
Wet Circuit Concentrate Manganese Grade	%	2.9	C
Final Tails Tonnage	t/y	11,752,808	C
Final Tails Iron Grade	%	19.6	C
Final Concentrate Iron Grade	%	65.2	C
Final Concentrate Silica Grade (maximum)	%	2.8	A
Final Concentrate Manganese Grade (maximum)	%	2.0	A
Final Concentrate Weight Recovery	%	33.4	A
Final Concentrate Iron Recovery	%	62.2	C
Final Concentrate Tonnage	t/hr	771	C

Sources:

A: Tacora
B : Soutex Recommandations
C: Calculated Data
D: Historical Data

The process description that follows was written in association with a Simplified Flow Diagram shown in Figure 1.3.

Figure 1.3: Scully Mill Flow Sheet



The ore is hauled by 211 metric tonne trucks and discharged into one of the two primary gyratory crushers. The ore is crushed to minus 8'' (203 mm) and transferred by two belt conveyors to 22,000 metric ton capacity crude ore storage bins ahead of the grinding circuit.

The ore is fed by vibratory feeders from the ore storage bins onto the mill feed conveyors. The ore is fed to fully autogenous (AG) mills that are 24 feet in diameter and 8 feet long. Each mill discharges through a trommel screen where the undersize is fed onto four (4) wet vibrating screens. The screens undersize is pumped to the gravity concentration circuit and the screen oversize is pumped back into the mills through the discharge end.

The first stage of the wet concentration circuit is made of 160 rougher spirals per line. The spiral concentrate is processed in hydrosizers where the lighter waste material is removed from the denser iron-ore concentrate with the use of a counter-current upflow of water through a bed of concentrate. The iron ore concentrate recovered at the underflow is sent to the drum filters to remove excess water prior to the drying step. The hydrosizer tails are sent to a cyclone and its underflow is pumped to new scavenger spirals to recover additional iron and increase the overall plant weight recovery.

The concentrate from the drum filter is fed to the dryers to obtain a dry concentrate which can then be processed in the dry magnetic drums of the manganese removal circuit (MRC). The MRC is made of low intensity magnetic separators (LIMS) followed by rougher and scavenger high intensity magnetic separators (HIMS). Each stage is used to separate hematite and magnetite from silica and pyrolusite to produce a low manganese and low silica concentrate.

The upgraded concentrate from the MRC's is transferred to the load-out silos. From the load-out silos, the concentrate is loaded into a freight train with closed rails cars for transportation to the port facilities.

The rougher spiral tails and the scavenger spiral tails are sent to a cyclone, where the overflow is directed to a 280-ft thickener to increase the density. The thickener overflow is directed to the process water tank. The cyclones underflow and the thickener underflow are pumped through three (3) lines each having two (2) pumps connected in series. In addition, on each line, there are up to 8 booster pumps located along the pipeline between the plant and the tailings impoundment area.

1.10 Mine Infrastructure

Service buildings and ancillary facilities exist and are in good condition to support operations. This infrastructure includes:

- Mine fleet maintenance facility with 5 bays and 50 t overhead crane;
- Wash bay within the maintenance facility;
- Auxiliary mine maintenance facility with 7 bays and 10 t overhead crane;
- Warehouse;

- Machine shop, electrical shop, paint shop and welding shop;
- Explosives storage magazines;
- Mine dewatering equipment and sedimentation ponds;
- Fuel storage tanks;
- Administration building.

1.11 Port Infrastructure

The concentrate is unloaded from railcars at Pointe Noire and stockpiled onto the former Wabush Yard, which is owned by SFP Pointe-Noire ("SFPPN"). The yard has a capacity of approximately 2,000,000 t of concentrate and once investment into a connection conveyor is completed, concentrate and can be either loaded directly onto a vessel at Dock 30 or multi-user Dock 35 or stockpiled to be reclaimed and loaded at a later date.

A new multi-user dock, Dock 35, owned by the Port of Sept-Iles, was built at Pointe-Noire in 2013. The dock has a capacity of 50 Mtpa via two 10,000 tph travelling ship loaders. The dock was designed to receive up to 400,000 DWT Chinamax vessels.

1.12 Tailings and Surface Water Management

The mill will produce approximately 320 million tonnes of tailings over a period of 27 years (2021 to end of 2047). The tailings output will vary between 10.5 to 13.3 million tonnes per year, with an average of 12 million tonnes per year. The tailings will be pumped in slurry form at a density of about 35% up to 48% solids by weight and transported via pipelines to the existing Tailings Impoundment Area ("TIA") for storage. The TIA will be expanded southward beyond its current permitted MDMER Schedule 2 limits following the natural valley confines of the Flora Lake watershed. The expansion of the existing TIA will require amendment of the existing MDMER Schedule 2 to continue storage of the life of mine tailings production. The proposed TIA expansion is designed to accommodate 320 million tonnes of tailings with opportunity to expand further if required but this could require expansion of footprint of the TIA and associated land rights, environmental permits and compensation considerations.

The design and operation of the existing and expanded TIA are in accordance with the CDA Dam Safety Guidelines (CDA, 2013) and the associated Application of Dam Safety Guidelines to Mining Dams Technical Bulletin (CDA, 2019). Appropriate instrumentation with dam safety monitoring and surveillance procedures are established for the existing dikes and will continue the safety management for future TIA expansion.

The tailings are considered low risk tailings from an acid leach and chemical standpoint; therefore, liners will not be required for tailings storage. As no flotation stream is involved in the process of the tailings, the material pumped to the tailings impoundment area is relatively coarse, allowing for good drainage and use as construction material for future embankments. The existing storage capacity, with the current dikes, is sufficient for at least 3 years of storage. Capital cost, in the form of new equipment or dike raises, is not required for this period and it is assumed that the current pumping system design is sufficient to discharge until 2024 with the exception of installing the planned third pump in PH#8 for Line 1 in 2022. Two additional pumphouses PH#9 and PH#10 will be required in Phase 5 (2024) and Phase 7 (2030) of the deposition plan, respectively. Future costs for tailings infrastructures have been estimated and included in the economic modelling of the project.

New construction is not planned for at least three years, additional studies, geotechnical field investigations and detailed engineering may be carried out during that time to finalize deposition beyond the southern dike. Given the low cost of constructing the dike raises in the upstream direction using free-draining, consolidated coarse sand tailings, adjustments resulting from detailed geotechnical stability studies are not expected to result in cost increases/decreases beyond the accuracy of the current feasibility study.

The decant water will report indirectly to North Flora Lake ("NFL") via South Flora Lake ("SFL"). Tailings are not permitted to be deposited directly into NFL, which will act primarily as a polishing system for the decant water and help prevent suspended sediments from continuing downstream towards Wabush Lake. Considering the side valley discharge approach of the TIA, there are effectively no cross-valley impounding dam structures, no decant pond formed and no spillway structure constructed for the operation of the TIA. Hence, the water management strategy requires the tailings deposition beach development be controlled and does not impede the drainage pathway (i.e., Diversion Channel) for the upper Flora Lake watershed. There is no water reclaim pumping system at the TIA given there is no requirement for tailings water re-use.

The closure strategy of the TIA incorporates progressive reclamation of inactive areas where tailings deposition has been rendered complete. The reclamation works generally include rehabilitation with vegetation on exposed tailings and disturbed surfaces, regrading of overstepped dike bench slopes, construction of an engineered Diversion Channel, and surficial drainage ditches on the tailings beach where concentration of runoff is anticipated.

1.13 Environmental Studies, Permitting and Social or Community Impact

Tacora prepared and submitted an Environmental Assessment Registration ("EA Registration") to the Government of Newfoundland and Labrador on September 28, 2017 in accordance with the Newfoundland

and Labrador *Newfoundland and Labrador Environmental Protection Act* (NL EPA, Part 10). The Government placed the document on a public notice period, responded to public comments, and released the reactivation project from further environmental assessment on November 21, 2017.

The EA Registration included discussions regarding the physical features of the project, natural habitat, potential resource conflicts, and socioeconomic influences of this site. These various factors were considered from reactivation, continued operation and eventual closure and rehabilitation of the mine site. The 2017 EA Registration document is the most recent overall environmental study completed at this site. The release outlined several conditions that were required prior to restart of the concentrator and mining. Another EA Registration document was also prepared and submitted by Cliffs Natural Resources ("Cliffs", the previous owners) in November 2015, however, that document was written with a focus on decommissioning and rehabilitation of the mine. This decommissioning project was released on February 29, 2016.

Ahead of reactivation, Tacora prepared four (4) major submissions for approval. This included three (3) plans and one (1) application to the Government of Newfoundland and Labrador in support of the facility which relate to environmental and other operational impacts of resuming operations at the Scully Mine. These were:

- Reactivation Plan: a document that described Tacora's plan to restore the site from its current condition at the time to operational readiness.
- Development Plan ("DP"): a document that describes the resumed operations of the mine and associated facilities for the remaining mine life.
- Rehabilitation and Closure Plan ("RCP"): a document that outlines mine closure and site rehabilitation for future land use. This plan includes closure cost estimates that were used to document the necessary financial assurance to the Government of Newfoundland and Labrador.
- Operating Certificate of Approval ("CofA" AA18-015646) application: this document describes the environmental control, monitoring and reporting measures that Tacora follows to assure compliance with federal and provincial environmental regulations.

The previous owners conducted air emissions testing (2008 through 2011) to determine pollutant emission rates for various aspects of the facility operations. These rates were then used as some of the input values into an air dispersion computer model that simulates potential ambient air quality impacts to the local area from facility operations over many years. Other model inputs include fugitive emissions from other mining related activity such as storage piles and vehicular traffic. The modeling report from 2014 indicated potential exceedances of ambient air quality standards for total suspended particulate matter ("TSP") and fine particulate matter ($PM_{2.5}$). While the computer simulation predicts possible ambient air quality standard exceedances, two actual ambient air monitoring stations located downwind of the site in the Town of Wabush have not recorded an actual exceedance attributable to facility operations in the multiple years that the monitors have been in service.

The area around the Scully mine has been heavily used and impacted by industrial mining operations for over fifty years. Most elements of the natural environment have therefore been temporarily or permanently altered by vegetation clearing, overburden/waste rock excavation, and ore mining.

The 2015 Decommissioning and Rehabilitation (Cliffs, Registration 1825) and 2017 Wabush Scully Mine Reactivation (Tacora Resources Inc., Registration 1931) environmental assessment registration documents contain descriptions of the upland environment around Scully Mine, but comprehensive terrestrial studies of this area have never been conducted. The major habitat loss for flora and fauna occurred decades ago when mining first began in this area. There is minimal habitat remaining for large mammals, forest specialist birds and mammals, amphibians, and forest-reliant vegetation. However, the area around Scully Mine still provides some suitable habitat for open habitat specialist songbirds, some species of small mammals, and generalist species like Red Fox.

There are no known Species at Risk using this area, and no rare plants have been documented from areas surrounding Scully Mine. However, surveys would be required to confirm these assertions.

Minimal clearing and grubbing are required as part of on-going operations, however, to avoid adverse effects on potential migratory birds and bird species of special conservation concern, all clearing activities will be conducted in accordance with accepted protocols related to avoidance of disturbing nesting sites. Tacora's no hunting, fishing, or trapping policy will be implemented throughout the construction and operation of the Project, therefore no other wildlife conflicts are anticipated.

The Project site is situated in a region with abundant aquatic resources including many small and large lakes, rivers, and associated streams and fish communities. The area affected by Tacora's operations has been subjected to industrial activity for over 50 years, and fish resources have been able to conduct some, or all their life cycle stages within suitable areas of the mine site.

Tacora is currently fully compliant with all fisheries compensation and offsetting required under the *Fisheries Act*. The Loon Lake Extension and Flora Riverine Compensation Channel projects, required as compensation for tailings deposition in Tacora's Tailings Impoundment Area, have been completed and monitoring concluded in 2014 and 2018, respectively. The Hay Riverine Compensation Channel Project, as required for the removal of the Hay Lake outlet for mining purposes, has also been completed and monitoring concluded in 2018. Tacora has been released from Letters of Credit associated with these projects. The Loon Lake Enhancement Project, as required for the removal of the Hay Lake for mining purposes, will undergo the last year of monitoring in 2021 and it is anticipated Tacora will be released from the Letter of Credit "(LOC") associated with this Project.

Tacora is also continuing all required monitoring of effluent discharges and water quality as required under the federal Metal and Diamond Mining Effluent Regulations ("MDMER"), and provincial (Certificate of Approval) criteria including acute and sub-lethal biological testing.

The reactivation of the mine and associated deposition of mine tailings to the end life of mine of 2047 will result in an expansion of the area affected by tailings and this will require an amendment to Schedule 2 of the MDMER under the *Fisheries Act*. This amendment will generate a requirement for an environmental assessment ("EA"). Continued tailings deposition to ELOM 2047 will impact several lakes and streams in the Flora Basins watershed and Tacora will be required to offset the fish habitat losses associated with these waterbodies. The expanded tailings deposition will affect approximately 360.6 units (100 m^2) of stream habitat and 50 hectares of lake habitat. Tacora is currently exploring the technical and financial feasibility options to offset the habitat losses. Tacora will be required to complete a fish offset plan, acceptable to Fisheries and Oceans Canada ("DFO"), and issue Letters of Credit to cover the plan's implementation and monitoring costs as required under the *Fisheries Act*. Tacora will need to complete the fish habitat offsetting prior to any habitat losses associated with continued tailings deposition.

Tacora will implement a no fishing policy throughout the operation of the mine; therefore, no conflicts with recreational fisheries are anticipated. There are no known historic and heritage resources within the Project area, however, there are no existing environmental studies that focus on cultural or historical resources. The site has already been heavily impacted and is located within an area that has been subject to on-going mining activity for the past five decades. It is therefore unlikely that the area contains, or that the Project will result in the disturbance or destruction of historic and heritage resources.

Water treatment associated with tailings management consists of natural (unaided) settling of solids in Flora Lake, approved under MDMER for tailings management. Flora South acts as a settling basin before water is passed onto Flora North. Water quality in Flora Lake, as measured at Flora Lake Discharge (Final Discharge Point under MDMER), which then flows to the exposure site Flora Lake Outlet Arm, is consistently in compliance with the metal criteria in the federal MDMER and provincial Effluent discharge Criteria in Table 5 of Certificate of Approval AA18-015646 and the acute lethality criteria in the MDMER and the CofA. However, TSS measurements typically exceed criteria during springtime monitoring events.

TSS exceedances in the Spring are attributable to on-going spring thaw and run-off resulting in higher-than-normal water levels increasing the TSS levels in all areas. The large amounts of snow melt during this period attributes to the amount of runoff. In addition, the forest fire of 2013, which surrounded Flora Lake, destroyed vegetation that would have previously aided in reducing the flow rate and natural filtration of any run-off. TSS levels are expected to diminish during the spring runoff period as the tailing's revegetation program proceeds and the vegetation matures and inhibits the erosion due to high runoff.

The cost to decommission and reclaim the open pits, waste rock dumps, ore handling and processing facilities, tailings management area and all of the related infrastructure associated with the Scully Mine site is estimated to be CAD 90,283,206. The cost estimate was determined by reviewing and updating the approved 2017 cost estimate, with some changes to reflect updated site quantities, additional research, recent contractor estimates, and updated cost estimates based on RS Means Costworks. Where possible costs based on current knowledge for activities were used. This estimate includes costs for an ongoing environmental monitoring program which will extend beyond closure. Additional long-term monitoring costs are included for maintenance and geotechnical inspections.

1.14 Market Studies

AME Consulting Ltd., a firm specialized in the study of metals markets, was retained by Tacora to provide information on the strongly related steel and iron ore markets. Most of the information in this section was provided by AME, but some complements were also provided by other specialists and analysts.

Iron ore is primarily used in the steel industry and is one of the key raw materials in the iron making process along with coke and limestone in a blast furnace ("BF"), and natural gas or coal in a direct reduction furnace.

Iron ore is generally produced from two types of ores: haematite and magnetite. The type of iron ore deposit will often determine the final iron ore product that can be produced. Magnetite ores are mostly lower in iron content than haematite and as such, must be beneficiated to produce finer grained concentrate products.

Haematite ore (Fe2O3) is a high-grade ore mainly found in large deposits of haematite rock formed by the in-situ enrichment of a protore already enriched in iron, most commonly a banded iron formation ("BIF") which consists of thin layers of iron oxides. Generally, the range of ore head grades, or contained iron content, for haematite deposits is greater than 55% Fe and may reach levels up to 70% Fe. Haematite ore has commonly been found in large-scale deposits in Brazil, Australia and India. When haematite ore is of sufficient contained iron content, it may be mined and processed using crushing and screening procedures before being exported for use in steel mills as direct ship ore (DSO).

Magnetite (Fe3O4) deposits have relatively lower contained iron content than haematite deposits, typically grading between 25-40% Fe. It is mainly found in BIFs located in several countries, including China, Russia, Ukraine, and the Americas. Due to its lower iron content compared to haematite, magnetite ore requires beneficiation to be converted into a higher-grade concentrate product to be viable for commercial use. Magnetite is processed into an iron ore concentrate via the beneficiation process.

The most important quality of an iron ore product is the Fe grade, or contained iron content, which is required to be within a particular range for commercial use in different steelmaking processes. While iron ore in-situ grades vary widely within a range of 25% up to around 70%, iron ore products generally contain iron content levels above 50% depending on the steelmaking process.

Pellet feed/concentrates are fines and ultra-fines which have undergone a beneficiation process. Pellet feed/concentrates are generally beneficiated from magnetite ore and exhibits higher iron grades and lower levels of impurities compared to DSO products. While magnetite deposits account for the vast majority of beneficiated orebodies, there are several notable beneficiated haematite deposits, particularly in North America. These are the Tilden, Carol Lake, Wabush and Mont Wright deposits.

Steel is one of the most widely used materials in the world given its applications in the construction, automobile and consumer durables sectors. Therefore, the underlying trend of iron ore demand is primarily driven by that of steel production. Over the past decade, industrialization and urbanization mainly occurred in developing countries such as China. During the course of this development, China became the world's largest steel producer and also the largest importer of iron ore. Increased urbanization rates in China, along with large investments in construction, power and infrastructure over the past fifteen years, have driven Chinese steel demand and therefore its demand for iron ore.

The steel industry largely operates on a regional basis, as freight costs are typically high in relation to the value of steel product, reducing its competitiveness if the product is shipped for a long distance. In addition, potential trade routes for steel can also be challenged by import quotas, anti-dumping duties and countervailing duty orders.

In 2020, global crude steel production fell 0.8% year-on-year to around 1,858 Mt, as steel output was disturbed by the ongoing impact of Covid-19. Despite this, China's crude steel production rose over 6% year on year to a record 1,056 Mt, as demand remained relatively strong despite the pandemic. Indian crude steel output was down around 10% at 100 Mt for the year, but still remain as the world's second largest steel producer.

AME explains the current status of the iron ore market as the result of highly unusual events. Being by far the largest steel producer and importer of iron ore, China's economic expansion and demand for steel in 2020 compensated for the demand shortfall in the rest of the world. China experienced an iron demand boom in 2020 that drastically contrasts with a slump in the rest of the world. As the COVID-19 pandemic became demonstrably contained within the March quarter, the Chinese government stepped in to stimulate the economy. Significant investment in infrastructure projects substantially renewed strength in manufacturing activity and steel production, increasing demand for iron ore. The country experienced an accelerated run-rate in construction to make up for the loss at the outset of the year.

In addition, the coronavirus pandemic was a significant roadblock for Brazil's iron ore operations and resulted in global supply disruption. The rapid spread of the virus in Itabira saw infection rates of around 12% of Vale's workers. In response, the company was required to send sick employees and those who had been in contact with them home – running on minimum staff numbers and reducing production rates. The iron ore volumes were also affected by mine shutdowns and heavier than usual rains that resulted in flooding. The combination of supply disruption and unprecedented demand from China resulted in a price spike in December to USD 143.27/t.

The impact of COVID-19 on the rest of the world was prolonged in the first quarter of 2021 until control measures were expanded and vaccines become available. Most of the world registered negative growth for 2020, with the exception of China. A major economic recovery is expected in 2021-2022 with economic growth exceeding average levels due to pent up demand and major investments in infrastructures by governments to fuel the economic recoveries.

Policy responses from governments to the pandemic are likely to favor spending on fixed asset and infrastructure projects which will lead to recoveries in demand for steel. Deficit spending is likely to fund these stimulus programs given the historically low interest rates seen worldwide, a palatable outcome politically given the low cost of debt funding. Concentrated stimulus programs targeting infrastructure have been initiated in both China with many economies in the world likely to follow, this poses a potential upside to medium term iron ore demand.

Key drivers for iron ore demand in the long term include the emerging economies of India, Brazil and Vietnam. Indian iron ore demand is driven by government policy to lift Indian crude steel production from 101 Mt in 2017 to 255 Mtpa by 2030. Steel development in India will focus on BF-BOF steelmaking and DRI, which are the major sources of iron ore demand. Vietnam's iron ore demand will be supported by ramp up of BF/BOF dominated steel expansions, while Brazil iron ore demand growth will benefit from its relative proximity to cheap iron ore sources. This will be offset by a decline in Chinese iron ore demand and an increase in scrap substitution.

A large proportion of global iron ore supply is not situated in proximity to major steelmaking regions and consequently there is a large internationally traded seaborne iron ore market. Over the medium to long term, exporters from Australia and Brazil will continue to dominate the global supply. From a 2020 base, Australia will increase production to 941 Mt by 2030 and Brazil will increase production to 501 Mt. Low-cost resource bases and sophisticated infrastructure see the combined countries control 57% of the global export market share. The key driver for iron ore supply is the major producers. In the event of the iron ore price falling, the major producers have demonstrated the ability to cut the cost of production and sustain margins, they may also increase or sustain production volume so as to grab market share from the higher-cost, marginal producers. Alternatively, they may slow down to decrease supply in response to low prices.

Undeveloped, high-quality, large-scale deposits are now more commonly located in remote areas, away from existing rail and port facilities, or near infrastructure where there is limited capacity available. New iron ore projects for delivery in the longer term will have higher capital intensity, given the need to develop associated rail or road networks and port infrastructure. Examples are the 40 Mtpa Mbalam-Nabeba project in Cameroon and the 100 Mtpa Simandou project in Guinea.

In recent years, spot pricing for iron ore has been widely adopted among market players with reference to the Platts 62% Fe Fines IODEX (Iron Ore Index). Value-in-use is a term used to describe the adjustments made against a benchmark price to account for differences in ore quality. Prices for iron ore products are generally set against the 62% Fe Fines Spot Price CFR North China benchmark prices and adjusted for value-in-use and freight differentials. The Benchmark 62% Fe Fines Spot Price is typically considered to have the following quality parameters: 4.5% silica (SiO_2), 2% alumina (Al_2O_3), 0.075% phosphorous (P), 8% moisture and 0.02% sulphur (S). The costs incurred at a steel mill are influenced, to an extent, by differing ore chemistries. The premium and discount applied to the Benchmark price for a specific ore is calculated based on the difference in iron content to benchmark and the impurity levels relative to trigger grades. Key impurities considered are silica, alumina, phosphorus and sulphur

Several sources were consulted to propose a price forecast for iron ore Benchmarks of 62% Fe and 65% Fe Northern China CFR. They can be included in three different groups:

- Specialized firms in metals markets.

It includes AME and two similar firms that focus on the demand and supply of steel and iron ores globally.

- Mining Analysist

These are the analysists retained by the research departments of various financial institutions. Our survey identified nine analysts with their individual forecasts.

- Forward Quoted Prices

The Bloomberg SGX (Singapore Exchange) presents daily the forward prices for both iron ore Benchmarks for the next three years. It must be noted that the contract volumes are relatively small beyond the next twelve months.

Our review of the different sources of information indicated that the forecasts from specialized firms and mining analysists underestimate the global economic recovery initiated in China in 2020 and expanding to the rest of the world in 2021 and 2022. Consequently, we used the average forward prices for 2021 and 2022, since it reflects better the global economic growth that will very likely result in a “super cycle” for metals and minerals similar to the period of 2010-2012. Our forecast for 2023 is the average of the forward prices and the estimates form mining analysts. Finally, the forecast for 2024 and beyond is based on a return to normal economic growth and demand for steel and iron ore in a relatively balanced market for steel and iron ore. There is a strong convergence of all forecasts beyond 2023. The various price forecast for iron ore are presented in Table 1.5.

Similarly, historical and forecast for the iron ore Benchmark 62% Fe are plotted on Figure 1.4. It shows that Benchmark prices are well supported in the range of $60-70 per tonne of concentrate and can average $100-110 per tonne over a complete price cycle.

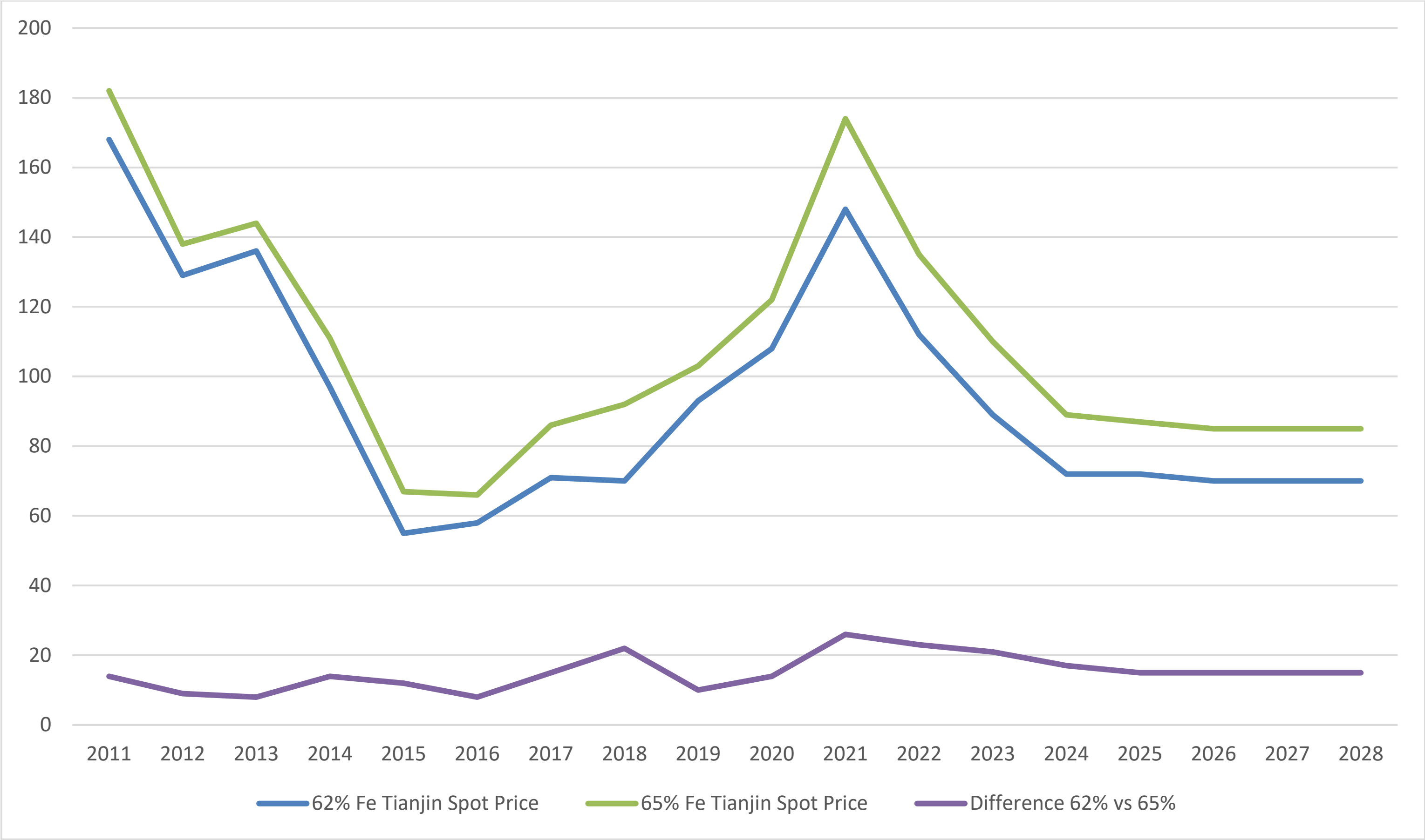
A general widening of the 62% Fe fines and 65% Fe fines pricing spread has been observed since mid-2016; it appears to be driven by higher demand for ferrous products with high-value-in-use, such as high grade and pellets to reduce coking coal consumption. AME expects this pricing spread to ease marginally in the short term, but remain elevated above long-term levels, as steel mills focus on productivity with an average of around 24%. The long term shift to higher grade raw material is expected to be driven by ongoing drive to reduce the pollution through increased productivity and efficiency in integrated steel works.

Table 1.5: Iron Ore Price Forecast 62% Fe China / 65% Fe China (USD/t)

Iron Ore Price Forecast 62% Fe China / 65% Fe China (USD/t)						
	2021	**2022**	**2023**	**2024**	**2025**	**LT**
AME (Feb 2021)	93.8/108.8	74/91	70/87	72/89	NA	72/89
FIRM 1 (June 2020)	68/75	64/72	69/79	71/80	73/84	70/83
FIRM 2 (Feb 2021)	113/128	80/94	70/84	74/89	66/79	65/78
Mining Analysts (Jan 2021)	128.89/NA	96.38/NA	83.83/NA	75.75/NA	77.67/NA	69.99/NA
Forward Market Prices (March 15, 2021)	147.99/173.96	112.12/135.29	94.19/116.01	NA	NA	NA
Updated Feasibility Study	147.99/173.96	112.12/135.29	89/110	72/89	72/87	70/85

Figure 1.4: Historical and Forecast Iron Ore Price 2011-2030 (USD/t, Real 2020)

	Historical										Forecast							
	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028
62% Fe Tianjin Spot Price	168	129	136	97	55	58	71	70	93	108	148	112	89	72	72	70	70	70
65% Fe Tianjin Spot Price	182	138	144	111	67	66	86	92	103	122	174	135	110	89	87	85	85	85
Difference 62% vs 65%	14	9	8	14	12	8	15	22	10	14	26	23	21	17	15	15	15	15



Based on the typical specifications for the Scully Sinter product, as supplied by Tacora, AME was able to compare the key parameters to other high-grade fines products, as well as the standard Benchmarks (Table 1.6)

Table 1.6: Comparison of Scully Sinter Specification with Peers and Benchmarks

Dry Weight %	Fortescue Blend	62% Fines CFR North China (Benchmark)	Pilbara Blend	65% Fines CFR North China (Benchmark)	Carajas Sinter Feed	Tacora Fines	Bloom Lake Sinter
Iron Content	58	62	61-62	65	65	65.9	66.2
Silica	5.5	4.5	3.5-4.5	3.5	2.1	2.6	4.4
Alumina	2.5	2	2.0-2.5	1	1.7	0.2	0.27
Phophorous	0.08	0.08	0.07-0.09	0.08	0.06	0.01	0.014
Sulphur	0.04	0.02					0.01

Source: AME, Tacora Resources, Company Reports

The iron ore content of the Scully Sinter product is above the high grade Benchmark iron content of 65%, and as such would be expected to attract a premium. This is also higher than the largest brand of high grade sinter in the seaborne market, the Carajas Sinter Feed blend. However, this is less than other Labrador Trough producers, such as Bloom Lake, with 66.2% iron.

Scully’s silica content is low at 2.6%, which is low against other Labrador Trough producers and high-grade Benchmarks of 4.4% and 3.5%, respectively. However, this is higher than the Carajas blend. With Chinese domestic ores typically having higher silica content, low silica iron ores are sought for blending.

Scully’s alumina levels are very low at 0.2%, which is typical of Labrador Trough ores, and well below benchmark specifications and other high-grade ores. Additionally, with very low phosphorous levels, again below benchmark and other high-grade iron ores, make this advantageous for blending with lower grade ores which typically have higher levels of these deleterious elements.

The LOI of the Scully product is expected to be very low at 0.5%. This is advantageous, as it can increase the efficiency of the sintering process. This compares to a 2-5% LOI typically seen in Brazilian ores and up to 10% LOI seen from some lower grade ores from the Pilbara.

1.15 Capital Costs

It was decided to classify capital expenditures (“CAPEX”) in two sections:

- Upgrade CAPEX: expenditures required to purchase equipment, repair, and upgrade the existing facilities and execute pre-production to reach a stable production of 6 Mt of concentrate per year.
- Sustaining CAPEX: expenditures required to maintain the production level of 6 Mt of concentrate per year through equipment replacement and major repairs or purchase of additional equipment and expansion of the Tailings facilities.

1.15.1 Upgrade Capital Expenditures

In 2018, the budget to re-start the Scully Mine of CAD 210.1 m inclusive of contingency but exclusive of applicable taxes, escalation, risk and management reserves. The current updated CAPEX adds up to CAD 251 m for an overrun of CAD 41 m or 19.5% as shown in Table 1.5. The accuracy of the estimate-to-complete (ETC) of CAD 95.03 m is at ± 15% given the methodology and level of confidence with respect to the project definition.

Table 1.7: Capex Estimate Summary per EWP

EWP # and Description	FEAS 2018 Total Costs (k CAD)	Closed Capital Cost (k CAD)	ETC (k CAD)
01 - Dryer Re-Built, #2	3,437	3,278	0
02 - Mill Gear Replacement, #6	1,021	781	0
03 - 1B Belt Magnet	313	229	0
04 - Fresh Water Header	218	35	0
05 - Mill Feed Chute	879	863	0
06 - Mill screen oversize and Spiral Feed Pump	389	-	0
07 - Manganese Reduction	8,360	4,529	0
08 - Primary Crusher Rock Hammer	1,995	1,499	0
10 - Added Loadout Bin Capacity	14,707	15,963	981
11 - Plant Structural Repairs	405	2	0
12 - Dryer PLC Update	1,212	3,292	0
13 - Electrical Refurbishment	4,757	4,778	0
14 - New Instrumentation	298	511	0
15 - Rod Deck Conversion	630	181	0
16 - Plant PLC Upgrade	2,020	3,181	0
17 - Railcars and Tripack	4,030	2,487	0
18 - Rail Track	287	53	0
19 - Mining	105,732	80,366	0
2021 Updated Feasibility			
21 - MCC Upgrade			2,595
22 - Mill Lubrication Project			1,262
23 - Mill Gear Replacement on L1 &L4			2,538
24 - Mill Recirc Line Project			1,137
25 -1B/2B Belt Replacement			429
26 - Scavenger Spiral Expansion			1,865
27 - Modernize Instrumentation and Control			851
28 - Tails Line Improvement			3,872
29 – Tailings Earthwork			385
30 - Mining Equipment			17,570
31 – Commissioning Bunker C			670
33 – 2022 Mill VFD, Gearing, other Upgrades			15,000
34 - 2022 SFPPN Port Upgrades			15,880
Subtotal: Direct Costs	**150,690**	**122,029**	**65,035**
Indirects (excluding escalation and risk)	**59,382**	**33,972**	**19,126**

EWP # and Description	FEAS 2018 Total Costs (k CAD)	Closed Capital Cost (k CAD)	ETC (k CAD)
Contingency			**10,867**
Total	**210,072**	**156,001**	**95,028**

Note: Numbers may not add due to rounding, Currency for CLOSED CAPEX is 1.25 USD TO CAD

1.15.2 Sustaining Capital Expenditures

The Sustaining CAPEX is grouped in three large sectors: Mine, Plant, Tailings. The CAPEX estimates per sector are shown in Table 1.8 for a total of CAD 618,231,844.

Table 1.8: Sustaining Capex Estimates

	CAD (000)
Mine	
Addition + Replacement	214,883
Major Repairs	224,810
Plant (Conc. + Infrastructures)	72,000
Tailings	
Earthwork	32,748
Pump & Pipelines	31,385
Contingency	42,405
Total	**618,232**

1.16 Scully Operating Costs

The operating costs at the Scully Mine were included in three areas:

- Mining
- Processing
- General Services and Administration.

The operating costs (Table 1.9) for the Project have been estimated at a Feasibility Study level with an accuracy of ±15%. Operating costs are based on year-end 2020 prices and wages and include procurement and logistics costs. No contingency was added to operating costs.

Table 1.9: OPEX

	$/t of concentrate		$/t milled		$/t mined	
	CAD	**USD**	**CAD**	**USD**	**CAD**	**USD**
Mine	14.74	11.79	4.90	3.92	2.70	2.16
Process	14.31	11.45	4.75	3.80	-	-
G&A	2.61	2.09	0.87	0.69	-	-
Total	**31.66**	**25.33**	**10.52**	**8.41**	**2.70**	**2.16**

1.16.1 Mining Operating Costs

Mining expenses were estimated by GMS from its data bank and specific consumables costs at Scully. All explosives and diesel costs are charged to mining operations. Equipment hours required to meet the LOM plan are based on productivity factors or equipment simulations. The delivered fuel price to site used in estimating mining costs is CAD 0.88/l and the consumption is calculated at 0.63 l/t mined. The mine wage scale established for the Project has operators earning between CAD 48/h and CAD 57/h, including benefits and bonus allowances.

The average mining cost during operations is estimated at CAD 2.70/t mined, including re-handling costs. This operating cost estimate excludes capital repairs, which are treated as sustaining capital. If considering the major components repairs, the mining cost would be CAD 2.96/t mined.

1.16.2 Processing Operating Costs

Processing cost were based on the following parameters proposed by Tacora and reviewed by GMS for reasonableness:

- Management and supervision costs.
- Operating and maintenance labor rates.
- Electrical power consumption and costs.

- Bunker C consumption and costs for boilers and dryers.
- Contract maintenance during shutdowns.
- Laboratory operations.
- External consultants.

The costliest consumable is the Bunker C for the boilers and dryers which require 4.442 l/t of concentrate. At current price of CAD 0.64/l, it represents an expenditure of CAD 17 M/y. All site electrical power is included in the processing costs. At full capacity, demand is 50 MW and the demand monthly charge is CAD 1.68/kw adding up to CAD 1M/y. Overall power consumption is 25.44 kw-h/t of ore processed at CAD 0.026/kw-h for CAD 0.66/t of ore.

Processing costs for an average year excluding the ramp-up and closure years, are estimated as follows:

Table 1.10: Processing Costs per Year

Fixed Costs	CAD (000)	CAD/t Processed	CAD/t Concentrate
Tacora Labour	24,353	1.39	4.06
Contractors	7,200	0.41	1.20
Vehicles	1.523	0.09	0.25

Variable Costs	CAD (000)	CAD/t Processed	CAD/t Concentrate
Power		0.66	1.93
Bunker C		0.97	3.02
Parts & Equipment		1.19	3.47

The total unit costs will vary from year-to-year depending on the tonnages processed to deliver the 6 Mt of concentrate per year.

1.16.3 General Services & Administration Operating Costs

General Services and Administration (G&A) costs include the following activities:

- Senior Management
- Insurance
- Accounting & Management Systems
- Security
- Health & Safety
- Environment
- Human Resources & Training
- IT and Communications
- Housing and Camps
- Offices and Building Maintenance

In Tacora's situation, some management time is required to plan and supervise activities performed by others such as rail transportation and port storage and ship loading, marketing and ocean freight.

The G&A cost was estimated at CAD 15,500,000/y or 9.0% of direct costs. Corporate costs were not included in the G&A costs.

1.17 Economic Analyses

The economic assessment of the Scully Mine has a start time of January 1st, 2021 and is based on price projections in the U.S. currency and cost estimates in the Canadian currency. An exchange rate of CAD 1.25 per USD was assumed to convert particular components of the cost estimates and USD market price projections into CAD. No provision was made for the effects of inflation: all costs are based on year-end 2020 (constant dollars).

The evaluation was carried out on an all-equity investment basis and assumed no initial debt and financial leverage. It must be noted that commercial production was declared in 2019 and most of the planned investment to repair and upgrade the facilities, mainly the process plant, were completed by year-end 2020. The remaining expenditures in 2021-2022 are for items added subsequently to the initial 2018 Feasibility Study. The current reserves are sufficient for the next 27 years (2021-2047).

Table 1.11 presents all revenues, transportation and port charges, royalties, OPEX and CAPEX, working capital requirements and closure cost on an annual basis. These are all input data required to determine annual cash flow streams, as discussed in previous sections.

Unit revenues are estimated during the period of the price super cycle currently experienced followed by an eventual return to long-term price forecast. These unit revenues are calculated on the bases of CFR Client, FOB Pointe Noire and FOB Scully Mine after taking account costs for ocean freight, rail transportation and port and royalties to third parties (Table 1.12). Some of these costs vary with time, concentrate tonnage handled or/and concentrate prices and are further discussed in Section 22.

Table 1.11: Annual Cash Flow

Annual Cash Flow

Tacora Scully Mine

					2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Period number																		
Operations		01-Jan-21	30-Sep-47		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
Closure		01-Oct-47	30-Sep-48		-	-	-	-	-	-	-	-	-	-	-	-	-	-
Production Highlights																		
Tonnage Mined	kt			868,807	28,228	32,849	32,000	32,000	32,381	32,000	32,000	32,000	32,000	32,000	32,000	32,000	32,000	32,000
Tonnage Processed	kt			478,943	15,588	18,084	17,573	18,328	18,530	18,239	18,715	19,292	18,236	18,043	18,334	18,802	18,475	18,232
Iron concentrate - dry	k dmt			159,100	5,124	6,000	6,009	6,000	6,002	6,000	5,999	5,987	6,000	5,999	5,995	5,946	5,980	5,948
Cash flow waterfall (k CAD)																		
Income			CAD/t of conc.															
Revenue (FOB Pointe Noire) (after 1 % deduction for losses)			87.10	13,856,858	928,541	840,807	642,621	506,533	495,599	482,105	482,007	481,056	482,105	482,024	481,714	477,742	480,527	477,924
Royalties MFC			-5.75	(915,517)	(63,405)	(57,039)	(43,024)	(33,403)	(32,630)	(31,676)	(31,670)	(31,604)	(31,676)	(31,671)	(31,650)	(31,378)	(31,568)	(31,390)
Other Royalties			-1.18	(188,422)	(6,068)	(7,106)	(7,116)	(7,106)	(7,109)	(7,106)	(7,104)	(7,090)	(7,106)	(7,105)	(7,100)	(7,041)	(7,083)	(7,044)
Total			80.16	12,752,920	859,068	776,662	592,480	466,024	455,860	443,323	443,233	442,361	443,323	443,249	442,965	439,322	441,877	439,490
Operational Expenditure																		
Mining		100%	-14.74	(2,345,076)	(88,745)	(89,882)	(86,750)	(87,869)	(86,481)	(86,226)	(87,509)	(94,078)	(96,421)	(98,435)	(88,892)	(87,821)	(84,040)	(84,415)
Processing		100%	-14.31	(2,276,398)	(103,567)	(93,215)	(82,979)	(86,331)	(86,468)	(85,113)	(87,333)	(90,027)	(85,097)	(84,199)	(85,555)	(87,739)	(86,210)	(85,076)
General & Administration		100%	-2.61	(415,493)	(16,368)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)
Concentrate Rail Transport and Port			-20.27	(3,224,736)	(145,947)	(156,214)	(131,772)	(119,823)	(118,885)	(117,689)	(117,672)	(117,503)	(117,689)	(117,675)	(117,620)	(116,914)	(117,409)	(116,946)
Total Opex		100%	-51.93	(8,261,704)	(354,627)	(354,811)	(317,001)	(309,522)	(307,334)	(304,529)	(308,014)	(317,107)	(314,707)	(315,809)	(307,566)	(307,974)	(303,159)	(301,938)
Operating Cost (CAD/t ore)				17.25	22.75	19.62	18.04	16.89	16.59	16.70	16.46	16.44	17.26	17.50	16.78	16.38	16.41	16.56
Working capital adjustments (during ops.)			-0.25	(40,289)	- 41,461	- 16,716	- 3,231	- 688	1,343	175	218	- 925	283	738	-	- 1,883	- 103	1,001
Operating cash flow			27.98	4,450,927	462,980	405,135	272,248	155,815	149,868	138,969	135,437	124,329	128,899	128,178	135,398	129,466	138,615	138,553
Upgrade Capital Costs																		
Upgrade Capital Costs				(84,161)	(31,780)	(52,381)	-	-	-	-	-	-	-	-	-	-	-	-
Upgrade Capital Costs-Contingency				(10,867)	(3,916)	(6,951)	-	-	-	-	-	-	-	-	-	-	-	-
Pre-production revenue				-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sub-total Upgrade Capital Costs		100%		(95,028)	(35,696)	(59,332)	-	-	-	-	-	-	-	-	-	-	-	-
Total upgrade capital cost				(95,028)	(35,696)	(59,332)	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital Costs																		
Mining Equipment and Major Repairs				(439,694)	-	(11,772)	(5,981)	(16,472)	(19,510)	(11,346)	(7,648)	(18,302)	(15,862)	(13,286)	(18,920)	(17,387)	(13,776)	(18,441)
Concentrator and Infrastructure				(72,000)	-	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)
Tailings Earthworks				(32,748)	-	(228)	(731)	(6,321)	-	-	(1,183)	(1,843)	(9,267)	-	-	(1,115)	(1,225)	(2,925)
Tails Pipeline				(31,385)	-	(1,107)	-	(6,726)	(1,081)	(1,081)	(1,081)	(1,081)	(1,081)	(6,726)	(648)	(648)	(648)	(648)
Grand Total Contingency				(42,405)	-	(1,239)	(859)	(3,231)	(1,588)	(1,179)	(1,172)	(1,804)	(2,795)	(2,123)	(1,493)	(1,584)	(1,420)	(1,908)
Total sustaining capital				(618,232)	-	(17,345)	(10,570)	(35,749)	(25,178)	(16,606)	(14,083)	(26,029)	(32,005)	(25,135)	(24,062)	(23,735)	(20,070)	(26,923)
Closure & Rehabilitation Costs																		
Closure & reclamation expenditures				(90,283)	(432)	(648)	(2,913)	(347)	(5,597)	-	(8,864)	-	(1,021)	(2,045)	-	-	-	(1,776)
Closure & reclamation cash flows				(90,283)	(432)	(648)	(2,913)	(347)	(5,597)	-	(8,864)	-	(1,021)	(2,045)	-	-	-	(1,776)
Cash flow Before Tax				3,647,384	426,852	327,810	258,765	119,719	119,094	122,364	112,489	98,300	95,873	100,997	111,337	105,731	118,545	109,855
Tax																		
Provincial income tax				(517,682)	(46,976)	(57,440)	(35,255)	(18,244)	(16,328)	(16,122)	(14,778)	(14,832)	(15,015)	(14,579)	(16,152)	(15,628)	(16,874)	(16,481)
Federal income tax				(517,682)	(46,976)	(57,440)	(35,255)	(18,244)	(16,328)	(16,122)	(14,778)	(14,832)	(15,015)	(14,579)	(16,152)	(15,628)	(16,874)	(16,481)
Total				(1,035,363)	(93,952)	(114,880)	(70,511)	(36,488)	(32,656)	(32,244)	(29,557)	(29,664)	(30,030)	(29,157)	(32,303)	(31,256)	(33,748)	(32,963)
Net Cash Flow				2,612,021	332,899	212,930	188,254	83,231	86,437	90,120	82,932	68,636	65,844	71,840	79,033	74,475	84,797	76,892

Annual Cash Flow

Tacora Scully Mine

Period number	Unit	%	CAD/t of conc.	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048
Operations		01-Jan-21	30-Sep-47														-
Closure		01-Oct-47	30-Sep-48	-	-	-	-	-	-	-	-	-	-	-	-		
Production Highlights																	
Tonnage Mined	kt			36,000	36,000	35,129	32,308	32,631	32,000	33,944	36,000	32,000	32,000	35,801	32,015	17,520	-
Tonnage Processed	kt			18,657	18,121	17,733	17,539	17,511	17,133	18,064	19,080	18,785	16,441	17,528	16,810	11,070	-
Iron concentrate - dry	k dmt			5,942	6,000	6,000	6,000	6,000	6,000	6,000	5,966	6,000	5,915	6,000	6,000	4,287	-
Cash flow waterfall (k CAD)			CAD/t of conc.														
Income																	
Revenue (FOB Pointe Noire) (after 1 % deduction for losses)			87.10	477,479	482,105	482,105	482,105	482,105	482,105	482,105	479,412	482,105	475,261	482,105	482,105	344,454	-
Royalties MFC			-5.75	(31,360)	(31,676)	(31,676)	(31,676)	(31,676)	(31,676)	(31,676)	(31,492)	(31,676)	(31,208)	(31,676)	(31,676)	(22,586)	-
Other Royalties			-1.18	(7,038)	(7,106)	(7,106)	(7,106)	(7,106)	(7,106)	(7,106)	(7,066)	(7,106)	(7,005)	(7,106)	(7,106)	(5,077)	-
Total			80.16	439,081	443,323	443,323	443,323	443,323	443,323	443,323	440,854	443,323	437,048	443,323	443,323	316,791	-
Operational Expenditure																	
Mining		100%	-14.74	(90,068)	(89,984)	(89,747)	(87,870)	(89,150)	(85,734)	(82,908)	(85,277)	(81,785)	(98,610)	(91,733)	(78,109)	(46,538)	-
Processing		100%	-14.31	(87,060)	(84,561)	(82,751)	(81,846)	(81,714)	(79,949)	(84,296)	(89,038)	(87,657)	(76,722)	(81,792)	(78,444)	(51,658)	-
General & Administration		100%	-2.61	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(11,625)	-
Concentrate Rail Transport and Port			-20.27	(116,867)	(117,689)	(117,689)	(117,689)	(117,689)	(117,689)	(117,689)	(117,211)	(117,689)	(116,473)	(117,689)	(117,689)	(85,224)	-
Total Opex		100%	-51.93	(309,496)	(307,734)	(305,687)	(302,905)	(304,053)	(298,872)	(300,394)	(307,026)	(302,632)	(307,305)	(306,714)	(289,743)	(195,045)	-
Operating Cost (CAD/t ore)				16.59	16.98	17.24	17.27	17.36	17.44	16.63	16.09	16.11	18.69	17.50	17.24	17.62	-
Working capital adjustments (during ops.)			-0.25	3,299	-3,521	1,540	-2,066	1,139	-1,724	2,376	930	389			6,100	12,500	
Operating cash flow			27.98	132,884	132,068	139,176	138,352	140,409	142,727	145,305	134,758	141,080	129,743	136,609	159,680	134,246	-
Upgrade Capital Costs																	
Upgrade Capital Costs				-	-	-	-	-	-	-	-	-	-	-	-	-	-
Upgrade Capital Costs-Contingency				-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pre-production revenue				-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sub-total Upgrade Capital Costs		100%		-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total upgrade capital cost				-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital Costs																	
Mining Equipment and Major Repairs				(10,491)	(80,269)	(27,218)	(28,478)	(7,564)	(18,417)	(7,989)	(8,781)	(28,798)	(14,151)	(13,839)	(4,997)	-	-
Concentrator and Infrastructure				(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	-	-	-
Tailings Earthworks				-	-	(1,115)	-	(1,115)	-	(1,225)	-	(1,089)	-	(2,046)	(1,319)	-	-
Tails Pipeline				(648)	(648)	(648)	(648)	(648)	(648)	-	-	-	-	(4,940)	-	-	-
Grand Total Contingency				(1,072)	(4,561)	(2,075)	(1,971)	(1,093)	(1,468)	(1,033)	(889)	(2,053)	(1,158)	(2,190)	(448)	-	-
Total sustaining capital				(15,211)	(88,478)	(34,057)	(34,097)	(13,420)	(23,534)	(13,248)	(12,670)	(34,941)	(18,309)	(26,015)	(6,763)	-	-
Closure & Rehabilitation Costs																	
Closure & reclamation expenditures				(1,984)	-	-	-	-	-	-	-	-	-	-	-	(20,933)	(43,725)
Closure & reclamation cash flows				(1,984)	-	-	-	-	-	-	-	-	-	-	-	(20,933)	(43,725)
Cash flow Before Tax				115,690	43,590	105,119	104,255	126,989	119,194	132,058	122,088	106,140	111,434	110,593	152,918	113,313	(43,725)
Tax																	
Provincial income tax				(15,385)	(15,529)	(14,695)	(15,278)	(15,619)	(16,982)	(17,198)	(16,371)	(17,400)	(15,653)	(16,773)	(19,605)	(6,489)	-
Federal income tax				(15,385)	(15,529)	(14,695)	(15,278)	(15,619)	(16,982)	(17,198)	(16,371)	(17,400)	(15,653)	(16,773)	(19,605)	(6,489)	-
Total				(30,771)	(31,059)	(29,390)	(30,556)	(31,239)	(33,964)	(34,395)	(32,743)	(34,800)	(31,306)	(33,546)	(39,209)	(12,978)	-
Net Cash Flow				84,919	12,532	75,729	73,698	95,750	85,230	97,662	89,345	71,340	80,128	77,047	113,708	100,335	(43,725)

Table 1.12: Revenue Estimates (USD/t of Concentrate)

		2021				2022	2023	2024	2025	LT
Benchmark		**1Q**	**2Q**	**3Q**	**4Q**					
62% Fe China		164.83	154.20	142.35	130.57	112.12	89.00	72	72	70
65% Fe Premium		26.00	26.30	26.41	25.19	23.17	21.00	17	15	15
Other Premium		0.98	0.82	0.76	0.71	2.00	2.00	2	2	2
Cargill Marketing		-7.50	-7.53	-7.54	-7.23	-7.04	-6.50	-5.50	-5.00	-5.00
Revenue (CFR Client)		184.31	173.79	161.98	149.24	130.25	105.50	85.50	84.00	82.00
Freight		-19.09	-17.46	-19.43	-19.62	-17.01	-19.08	-17.28	-17.28	-17.07
Net Revenue (FOB Pointe Noire)		165.22	156.33	142.55	129.62	113.24	86.42	68.22	66.72	64.93
Rail Transport - QNSL		-18.27	-15.87	-14.92	-13.96	-13.25	-11.38	-10.01	-10.01	-9.85
Railcar Maintenance		-0.53	-0.37	-0.35	-0.32	-0.27	-0.27	-0.27	-0.27	-0.27
Port SFPPN		-9.93	-7.29	-5.85	-5.64	-6.51	-5.56	-5.56	-5.56	-5.56
Royalties	**MFC**	-11.24	-10.62	-9.64	-8.76	-7.61	-5.73	-4.45	-4.35	-4.22
	Others	-0.34	-0.34	-0.34	-0.34	-0.84	-0.84	-0.84	-0.84	-0.84
Net Revenue (FOB Scully)		**124.91**	**121.84**	**111.45**	**100.60**	**84.76**	**62.64**	**47.09**	**45.69**	**44.19**

Notes: Exchange Rate: 1.25 CAD/USD
Mass Recovery: 34.65%
Concentrate Losses at Port: 1%
Net Revenue (FOB) Pointe Noire will be reduced to 99%

In order to evaluate the investment required in the Working Capital, the basic information was obtained from the Tacora Corporate Financial model which provide an estimated statement of short-term assets and liabilities.

The investments initially required in the Working Capital are as follows:

Table 1.13: Working Capital Requirement – Early years

	CAD (000)
2021	41,461
2022	16,716
2023	3,231

Fluctuations in the working capital afterwards remain in the ±CAD 2M until 2046-2047 when the working capital is liquidated.

For rehabilitation and closure costs, progressive grading and revegetation of the tailings pond and waste dumps will be scheduled during operations with expenditures of CAD 31,984,000 including contingency. However, the greater amount of expenditures for closure will occur in 2047-2048 at the end of operations and are budgeted as follows:

Table 1.14: Rehabilitation and Closure Costs

	CAD (000)
Progressive	31,984
Closure 2047	14,575
Closure 2048	43,724
Total	**90,283**

Tacora is exempt from the current regime of mining taxation in Newfoundland & Labrador. On the other hand, it is required to pay the Nalco-Javelin royalty in lieu of mining taxes. However, Tacora is subject to federal and provincial income tax at a rate of 15% of taxable income at each level of government (total 30%). Taxable income is calculated as the remaining amount after deducting from revenues all OPEX and capital cost allowances based on schedules specified by the tax authorities. Taxes are payables in instalments monthly or quarterly and tax losses can be recovered from taxes in future years or from taxes paid in the prior three years.

Tacora has extensive tax pools of CAD 310M at the end of 2020.

1.17.1 Valuation – Base Case

The valuation of the Scully Mine is based on the cashflows estimated over the 27 years of Life-of-Mine (LOM). The start of the valuation is based on the situation prevailing on January 1st, 2021. Prior expenditures to bring the Mine out of receivership and to upgrade facilities are excluded from the analysis; however, tax pools available at the end of 2020 were used in the tax calculations for following years.

The net cashflows before and after taxes over the LOM are presented in Table 1.15. Over the period 2021-2023, the Scully Mine will benefit from the price super-cycle being experienced in the iron ore industry; net cashflow is heavily leveraged to Benchmark prices. It is also the period when income taxes will be at their highest levels between CAD 71 m to 115 m per year.

Once Benchmark prices return to their long-term level, yearly cashflows net of taxes will vary from CAD 65 m to CAD 97 m and income taxes will fluctuate between CAD 29 m to 36 M.

Table 1.15: NPV Results – Base Case

Discount	NPV – Before Taxes		NPV – After Taxes	
	(CAD M)	(USD M)	(CAD M)	(USD M)
0%	3,647	2,918	2,612	2,090
8%	1,872	1,498	1,345	1,076
10%	1,675	1,340	1,205	964

1.17.2 Sensitivity Analyses

There are many scenarios that can be tested to estimate the impact of variable parameters on the value of the Scully Mine. The following parameters were selected:

- Net Revenues – Pointe Noire
- CAD/USD exchange rate

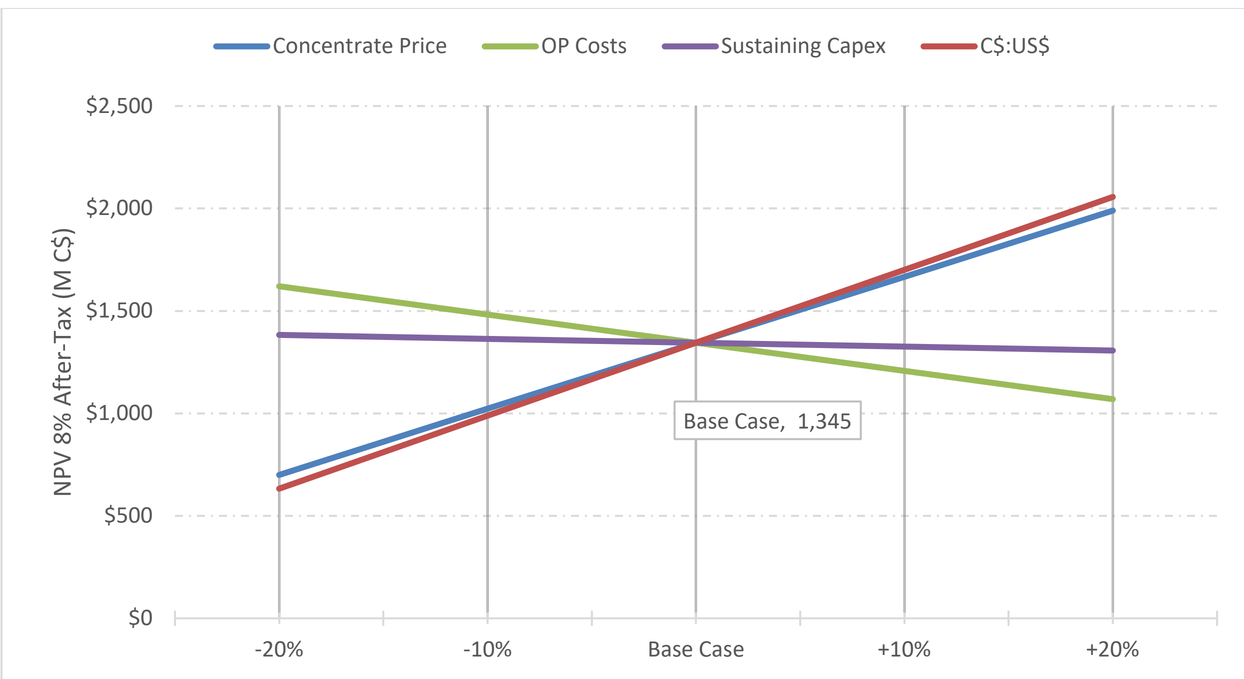
- Operating Costs at Scully Mine
- Sustaining Capital

The ± 10% and ± 20% variations of these parameters were computed to study the resulting fluctuations in NPV values. In all cases, the data for 2021 were fixed and not affected by sensitivities. Net Revenues – Pointe Noire were selected to study the impact of fluctuations in iron ore price; these fluctuations can be due to variations in Benchmark prices, quality premiums, location of clients and related ocean freight. Costs for rail transportation and the MFC royalties are partially indexed to the Net Revenues - Pointe Noire.

Variations in the After-Tax NPV's for the four sensitivities are graphed in Figure 1.5.

When looking at results in Canadian dollars, the after-tax NPV's are particularly sensitive to fluctuations in revenues that can be caused by Benchmark prices, premiums, client location, ocean freight and exchange rate; a 20% variation in Net Revenues at Pointe Noire and exchange rate will result in an NPV (8%) differential of CAD 645 m and CAD 711 M, respectively. The next parameter of importance is the operating cost at the Scully mine with an NPV (8%) differential of CAD 276 m for a 20% variation. Finally, a 20% variation in sustaining capital cause only an NPV (8%) reaction of CAD 32 M.

Figure 1.5: Sensitivity – 8% NPV – After Tax



1.18 Interpretation and Conclusions

1.18.1 Geology and Mineral Resources

The geological interpretation for the Scully deposit is based primarily on diamond drilling data and 2D sectional interpretations by representatives of Cliffs in 2014. The geology of the deposit is well understood.

The mineralization is found in three main members (Upper, Middle and Lower) of the Sokoman Formation. Each member is subdivided into sub-units, which present different ore mineralogy, recoveries and contaminant levels. The mineralization controls of the deposit are also well understood.

The protocols followed to collect sample data post-2002 are considered sufficient for NI 43-101 purposes. No evidence of established protocols was found for drillings prior to 2002; therefore, confidence in this information is lower. Sampling has been undertaken based on geological logging and is adequate for the mineralization style and size of the deposit.

QAQC samples submitted as part of the reanalysis program returned values within expectations. No QAQC data was available for historical drilling (all drilling pre-2014), and all historical analyses were undertaken by the on-site laboratory. The laboratory was not certified. GMS considers this insufficient, however Scully was a long-term past producer of iron ore concentrate and has demonstrated the ability to reconcile grade with the geological model. GMS has outlined considerable recommendations to return the laboratory and sampling practice on site to NI 43-101 compliance.

The geological model includes a total of 13 lithology units which were modelled using the provided 2D sectional interpretations and the coded drill hole database. 3D wireframe solids were produced in Leapfrog™ and are representative of the folded lithologies present in the Scully deposit. Each bedding orientation is appropriately defined within the 14 structural domains dividing the geological model.

Mineral Resources were estimated in the mineralization and structural domains, using Geovia GEMS™ from 6.0 m long composites using four ordinary kriging interpolation passes. Blastholes were included in a small initial pass, and the remaining three passes only used exploration drill holes. Each search ellipse was incrementally larger than the previous and was based on variogram ranges and the drill holes spacing.

The performance of the block model to predict resource estimates was evaluated through global and local validation methods, including visual comparisons, descriptive statistics, swath plots, Q:Q plots and blasthole data comparisons. No production data was available to validate the accuracy of the model to true known grade. Block grades were found to reproduce composite grades sufficiently in the block model.

The Mineral Resources are reported within a Lerchs-Grossman open pit shell and are effective as of January 1st, 2021, using a minimum Lab Weight Recovery (Wilfley Table results) of 10% and a long-term iron concentrate price of USD 93.50/t CFR Client.

Open Pit Measured and Indicated Mineral Resources total 721.9 Mt at an average grade of 34.65% Fe. Open Pit Inferred Mineral Resources total 263.4 Mt at an average grade of 34.1% Fe.

Mineral Resources were classified into Measured, Indicated, and Inferred categories according to the CIM Definition Standards on Mineral Resources and Mineral Reserves as adopted in NI 43-101.

A compilation of the mineralized blocks generated a graph of cumulative tonnage for various iron grades. It confirmed that the iron mineralization is relatively uniform and entirely contained in the mineralized members. The graph confirms that there is very limited ore tonnage below 28% Fe.

1.18.2 Mining and Mineral Reserves

Open pit optimization was conducted using Whittle software to determine the optimal economic shape of the open pit to guide the final pit design process. Pit optimizations use a long-term Benchmark Iron Ore Price 62% Fe-CRF China of USD 70/t of concentrate; after taking into account premiums, marketing fees and ocean freight, the Benchmark Price translates to USD 64.93/t of concentrate FOB Pointe Noire for a 65.6% Fe iron ore concentrate.

The mine design and Mineral Reserve Estimate have been completed to a level appropriate for feasibility studies. Definitions for Mineral Reserve categories used in this Report are consistent with the CIM definitions as adopted by NI 43-101. Historically, the Mineral Reserve Estimate was based on a minimum Lab Weight Recovery of 27% for all sub-units except sub-unit 52 at 30%. %. In addition, sub-unit 34 has a ratio of weight recovery to iron of at least 1. Another compilation of the mineralized blocks generated a graph of cumulative tonnage for various Lab Weight Recovery. It can be observed that minimal tonnage is added with a Wet Recovery below 18%. However, almost 100 Mt of reserves could be added if the minimum Wet Recovery is reduced from 27% to 20%. It was subsequently calculated that ore with a minimum 20% Wet Recovery still generated a positive cash flow and should be included in mineral reserves.

Proven and Probable Mineral Reserves are estimated to be 478.9 Mt at an average grade of 34.89% iron and 2.62% manganese for 159.4 Mt of iron concentrate at 65.6% Fe. The open pit generates 390.9 Mt of overburden and waste rock for a strip ratio of 0.82:1 over the LOM with 18.8 Mt of ore transiting through stockpiles for blending purposes and to balance mining and milling constraints. A total of 868.8 Mt is mined from the pits. An owner mining approach with conventional open pit mining techniques is planned.

The LOM plan details 27 years of production (2021 to 2047) which assumes one year of continuing ramp-up before achieving the target production rate of 6 Mt of concentrate per year. The peak mining rate will be approximately 36 Mt with a peak of 37.4 Mt moved. The average mining rate is 32.2 Mtpy over the LOM.

Waste rock will be stored in several waste dumps around the pit limits including the North Dump, NW Dump, SW Dump and South Dump. In addition, an in-pit dump is planned in the West Extension to reduce haulage costs and footprint impact.

1.18.3 Processing

The process plant upgrades have for objective to produce 6 Mt of concentrate per year from a maximum of 18 Mt of ore feed to the plant.

Mill upgrades are planned to increase plant reliability and availability. These improvements include replacement of the gears on two mills, replacement of the mill lube systems, replacement of the mill starters and relays and improvement of the mill recirculation line and pump type.

The operation philosophy for the autogenous (“AG”) mills will be modified to utilize the motor power to its full capacity in order to reach the targeted throughput.

Scavenger spirals were added to increase the plant wet recovery and the high-tension scavenger circuit is to be restarted to improve dry recovery.

Efficient preventive maintenance and the use of contractors to assist in major planned shutdowns are considered key factors to improve plant availability.

1.18.4 Environmental and Permitting

The reactivation project was released from further environmental assessment in November 2017. Following the environmental assessment release, numerous approvals, permits and authorizations required from municipal, provincial, and federal regulators, were obtained or amended prior to reactivation.

The reactivation of the mine and associated deposition of mine tailings to the end life of mine of 2047 will result in an expansion of the area affected by tailings and this will require an amendment to Schedule 2 of the MDMER under the *Fisheries Act*. This amendment will generate a requirement for an environmental assessment (EA). Continued tailings deposition to ELOM 2047 will impact several lakes and streams in the Flora Basins watershed and Tacora will be required to offset the fish habitat losses associated with these waterbodies. The expanded tailings deposition will affect approximately 360.6 units (100 m^2) of stream habitat and 50 hectares of lake habitat. Tacora is currently exploring the technical and financial feasibility options to offset the habitat losses. Tacora will be required to complete a fish offset plan, acceptable to Fisheries and Oceans Canada (DFO), and issue Letters of Credit to cover the plan's implementation and monitoring costs as required under the *Fisheries Act*. Tacora will need to complete the fish habitat offsetting prior to any habitat losses associated with continued tailings deposition.

Revised Development and Rehabilitation and Closure Plans (DP & RCP) were submitted to IET in 2017/2018 and approved in 2018. A proponent is required to submit an updated DP and RCP every five years, or if there is a significant change to the project, therefore, updates will be required to these documents in 2023. Tacora is also required to provide a financial assurance covering the estimated closure costs outlined in the RCP.

Since re-activation in 2019, Tacora continues to comply with numerous conditions and commitments of approval, and various standards contained in federal and provincial legislation, regulations and guidelines. These conditions and commitments will need to be considered during any remaining design, construction, operations and closure phases of the mine.

1.18.5 Tailings Storage Facility and Dams

The existing storage facility will be used with an estimated capacity sufficient for three years. As this capacity is depleted, planning and construction of new dikes and additional pumping capacity will be required.

Current and future expansion capacity is available without needing to directly discharge tailings into NFL. To meet life of mine targets for tailings impoundment, the storage capacity of SFL and part of its surrounding watershed is required.

After almost 50 years of deposition, there exists a substantial tonnage of tailings in the tailings basin at Flora Lake. Due to gravity segregation from slurry discharge of the tailings a large high grade delta containing significant iron units has been created over the 50 years of prior operations. This iron resource presents a potentially attractive opportunity for mining and processing of iron bearing tailings to recover iron. Such recycling operations present an excellent improvement in sustainability of iron mining operations and should be investigated in due course. The new mineral lease covering the Scully Mine comprehends such a possibility and provides an incentive for such sustainable recycling of tailings by providing a 40% reduction in the royalty rate applicable to concentrate produced from tailings.

1.18.6 Project Economics

When compared to the global suppliers of iron ore, the Scully Mine will always present a cash operating cost above average for the following fundamental reasons:

- Nature of the ore and its grade require concentration with inherent Fe losses and additional costs
- Relatively higher stripping ratio
- Costs of rail transportation and port operations owned and supplied by third parties
- Higher ocean freight due to a larger distance to the main Asian customers

On the positive side, the Scully Mine benefits from the following:

- Large resources and reserves
- High concentrate quality that results in significant premium from customers
- A stable political and social environment.

The cash operating revenues and costs based on long-term estimates for the Scully operations are as follows:

Table 1.16: Cash Operating Revenues & Costs on Long-term Estimates

	CAD/t of Concentrate	USD/t of Concentrate
Benchmark 62% Fe China	87.50	70
Premium-Marketing	15.00	12
Ocean Freight	(21.34)	(17.07)
Concentrate Losses – Pointe Noire	(0.81)	(0.65)
Rail Transportation & Port	(19.60)	(15.68)
Royalties	(6.33)	(5.06)
Scully Mine Operating Cost	(31.66)	(25.33)
Operating Cash Flow	**22.76**	**18.21**

Total Cash Operating Costs are USD 63.79 /t of concentrate (CAD 79.74/t) or 77% of revenues; only 37% of the Total Cash Operating Costs are under Tacora's direct control. Obviously, Tacora must realize excellence in its operations and demonstrate discipline in its sustaining capital to remain in business. Tacora's capability to maintain its assets and meet and exceed its targeted performance will be critical to its financial future.

1.19 Recommendations

Multiple recommendations are listed in Section 26 and Tacora should address them once costs and priorities are determined.

The single most important recommendation from a business and financial point of view would be to determine the ultimate reserves and its related LOM mining and processing plan.

The resources are compared to mineral reserves in Table 1.17. Obviously, there is a large potential to increase reserves beyond the current level. In addition, pit modeling for resources and reserves estimates were limited in space to provide buffers to rail loop, primary crushers, and lakes. Sensitivity analyses should be conducted to determine if the value of additional resources/reserves could justify the removal and / or relocation of some of the obstacles. In any case, the location of the resources outside of the reserves envelopes should be identified to guide additional exploration and drilling programs and aim to generate the ultimate total reserves for the Scully Deposit.

Table 1.17: Comparison Resources and Reserves

	Resources M&I	Resources Inferred	Reserves
Tonnage (Mt)	721.9	263.4	478.9
Fe Grade (%)	34.7	34.1	34.89
Mn Grade (%)	2.4	2.1	2.62
WTREC (%)	35.8	34.0	33.29

The additional and final exploration program should be combined with additional hydrology, hydrogeology and geotechnical studies to allow for a final pit design. These final reserves and related pit designs would allow to properly plan the long-term locations of waste dumps (both outside and inside depleted pits) and tailings disposal facilities.

More specifically, additional hydrogeological investigations are recommended for the East, South and Boot Pits to investigate groundwater infiltration. It should include the following:

- Start a groundwater level monitoring program.
- Drill and conduct hydraulic tests at Boot Pit, South Pit, East-Pit-East and East-Pit-West.
- Validate the perched condition hypothesis for Knoll and Vern Lakes.
- Carry out additional geophysical surveys in the Duley and Little Wabush Lakes areas.
- Update the groundwater model with the data obtained during the field operations.

Additional waste rock storage options should be investigated. In the event of expanded larger open pit limits optimized for higher iron ore prices, additional waste dump storage capacity will be required and may limit or defer the possibilities of in-pit waste storage.

Recommendations were made for further work to gain confidence in pit slope design. Additional work would include:

- Drilling geotechnical holes and the installation of wire piezometers to verify pore pressure buildup.
- Carrying out uniaxial compressive strength tests on samples from geotechnical holes for all geological units.
- Carrying out four direct shear tests along bedding planes.
- Review variation in unit quality related to degree of weathering.
- Gather structural data and review potential kinematic failures on the hanging walls and end walls.
- Install an inclinometer in the east wall of the East Pit East.

2 INTRODUCTION

2.1 Background

Tacora Resources Inc. (¨Tacora¨) acquired the assets associated with the Scully Mine located in Wabush, Newfoundland and Labrador, Canada, as announced in the July 19, 2017 press release. The asset purchase agreement with Wabush Iron Co. Limited, Wabush Resources Inc. and Wabush Lake Railway Company Limited was executed through a court-supervised process under the Companies' Creditors Arrangement Act ("CCAA").

The Scully Mine, also referred to as Scully Mines, was operated by Cliffs Mining Company, a subsidiary of Cliffs from 1986 when Cliffs acquired PM to its closure in February 2014. PM designed and built the facilities and began operations in 1965. The Scully Mine had the capacity to produce up to 6 Mt of iron ore pellets per year. The former operation included a pellet plant located at Pointe-Noire near Sept-Iles.

Cliffs shut the pellet plant in May 2013, and then the mine and concentrator closed in February 2014. On November 19, 2014, Cliffs announced the pursuit of exit options from its Eastern Canadian iron ore operations.

Several efforts were undertaken by Tacora to restart the Scully Mine, including:

- The negotiation and signing of a new collective bargaining agreement with the United Steelworkers union.
- Establishment of a 15-year iron sales agreement with Cargill for the purchase of 100% high grade iron ore concentrate produced through 2032.
- Negotiation of a new mineral lease valid to May 2055 with improved commercial terms.
- Execution of a new life-of-mine rail transportation agreement with the Quebec North Shore & Labrador ("QNS&L") Railway, for the transportation of product from Wabush to Sept Iles.
- Purchase of several pieces of mobile mining equipment, including shovels and rotary drills to allow rapid start-up of operations.
- A decision was made to sell concentrate instead of pellets, thereby the mine has been de-bottlenecked from pellet production.

The bulk of the remaining Scully Mine resources consist of high-manganese iron ore, primarily at depth below the existing operations and near surface as pushbacks to the existing open pit. To achieve a concentrate with a manganese content below 2% subsequently. Cliffs piloted a manganese removal circuit and commissioned two (2) lines in the plant which demonstrated its efficiency in reducing manganese with an incrementally modest cost. Four (4) additional manganese reduction lines were installed in the concentrator (magnetic separation) to treat the entirety of the mill feed.

2.2 Scope

Tacora engaged G Mining Services ("GMS") to update the previous Canadian National Instrument 43-101 ("NI 43-101") Technical Report dated May 10th, 2018.

Since the issuance of the 2018 NI 43-101 Technical Report, Tacora has added manganese reduction lines in the concentrator to reduce the manganese content in the final concentrate to below 2%. It has also undertaken various plant refurbishments required for the concentrator building and equipment to operate reliably. A steady state annual production rate of 6 Mtpy of iron ore concentrate is expected, with an iron grade in excess of 65% iron. Several pieces of mine equipment were purchased for the project. Furthermore, Tacora has initiated several projects described in the current report, which aim at allowing Tacora to reach the target production rate by the end of 2021

A number of scope elements were prepared by additional consultants including Soutex, Hatch and Sikumiut Environmental Management Limited ("SEM"). Table 2.1 provides a summary of the areas of responsibility and associated firms.

Table 2.1: Areas of Responsibility

Scope	Company
Geological model; Mineral Resource Estimate; data verifications	GMS
Mineral Reserve Estimate; mine plan; mining methods and equipment requirements; explosives supply; mine operating and capital cost estimation	GMS
Pit slope geotechnical assessment; mine hydrogeological model	GMS
Mineral processing; recovery methods; mass balance; concentrator and G&A operating cost estimation	Soutex
Manganese removal line integration; plant and equipment refurbishments capital cost estimation;	Soutex
Tailings storage facility design; tailings management; tailings filling plan	Hatch
Concentrate rail transportation; port facilities	Tacora
Environmental impact assessments; permitting	SEM
Concentrate marketing	Tacora
Economic analysis	GMS

2.3 Basis of the Report

Information presented in this Technical Report is based on the following:

- Information provided by Tacora. It should be noted that the update to the Technical Report relied heavily on data provided by Tacora on its current and forecasted operational data in terms of throughputs, capacity and availability of equipment and Plant.
- Manganese removal line commissioning reports.
- Various engineering reports and assessments of the former Scully Mine.
- Previous operations data.
- COREM – core sample characterization test work.

2.4 Project Description

The restart of the Scully Mine Project two years ago included the following elements:

- A new mining plan and additional mobile equipment including:
    - Purchase of mine trucks.
    - Purchase of support equipment such as track-dozers, graders, etc.
    - Purchase of mine dispatch system and radio network.

- Reuse of the mine maintenance facility, workshops and washbay.
- Reuse of the explosives magazines with an upgrading of the emulsion storage site in accordance with supplier specifications.
- A new waste rock management plan including dump expansions, raises and in-pit dumping.
- Open pit dewatering.
- Installation of rock breakers at the primary crushers.
- Concentrator upgrade with the conversion of HT separation lines to magnetic separation for better manganese removal.
- Revised tailings management plan and storage facilities in accordance with new mine plan.
- Additional rail loadout storage capacity to accommodate larger trains.
- Upgrade of the dryer burner system.
- Upgrade of the dryer scrubbers for better environmental performance.
- Completion of several equipment modifications or upgrades to improve plant availability.

2.5 Qualified Persons ("QP") and Site Visits

The list of QPs who participated in the update of the current Technical Report and their areas of responsibility are summarized in Table 2.2.

Table 2.2: Qualified Persons Summary

Name of QP	Company	QP Sections
Joel Lacelle, P. Eng. Elie Accad, P. Eng.	GMS	Sections 1, 2, 4, 5, 6, 15, 18, 19, 22, 25.2, 26.2 and 27
Réjean Sirois, P Eng. and James Purchase, P.Geo	GMS	Sections 1, 6, 7, 8, 9, 10, 11, 12, 14, 25, 26
Antoine Champagne, P Eng.	GMS	Sections 1, 16, 21, 23, 24, 27
Antoine Berton, P Eng.	Soutex	Sections 1, 13, 17, 21, 25, 26, 27
Karlis Jansons	GeoMin Initiatives	Sections 1.12, 1.18.5, 4.6, 5.4.5, 17.3.7, 18.4, 20.2.7.1, 20.4.1, 21.2.3, 25.6, 26.4
Adam Salomon de Friedberg	Tacora	Section 1, 4, 23, 24
Amy Copeland	SEM	Section 1, 20, 25, 26

3 RELIANCE ON OTHER EXPERTS

The authors have updated this Technical Report using existing information gathered from previous studies and engineering work, past operations, historical operational data from the plant, historical data from the mining operation of the Scully Mine, technical field surveys and a metallurgical test work campaign.

The authors have relied heavily on technical, operational and financial information supplied by Tacora Resources Inc. as it pertains to their current and forecasted operations, their current and future improvement projects as well as the actual and projected performance of the Scully Mine. The authors of this Technical Report did not conduct a thorough review of each consultant's work. The sections provided for this Technical Report were supplied by reputable consultants, and there is no reason to doubt the validity of the information.

The marketing Section 19 in the initial Technical Report was provided by AME. The environmental Section 20 of the Technical Report was updated by SEM. SEM is the largest wholly Newfoundland and Labrador owned and operated environmental consulting firm in the province. It is registered as an Inuit Business with the Nunatsiavut Government and is considered as an Aboriginal Business under Federal Government criteria.

Golder Associates issued memorandums in January 2021 pertaining to the geotechnical and hydrogeological parameters of the Scully Mine. The reports were reviewed by G Mining Services who updated the relevant sections accordingly.

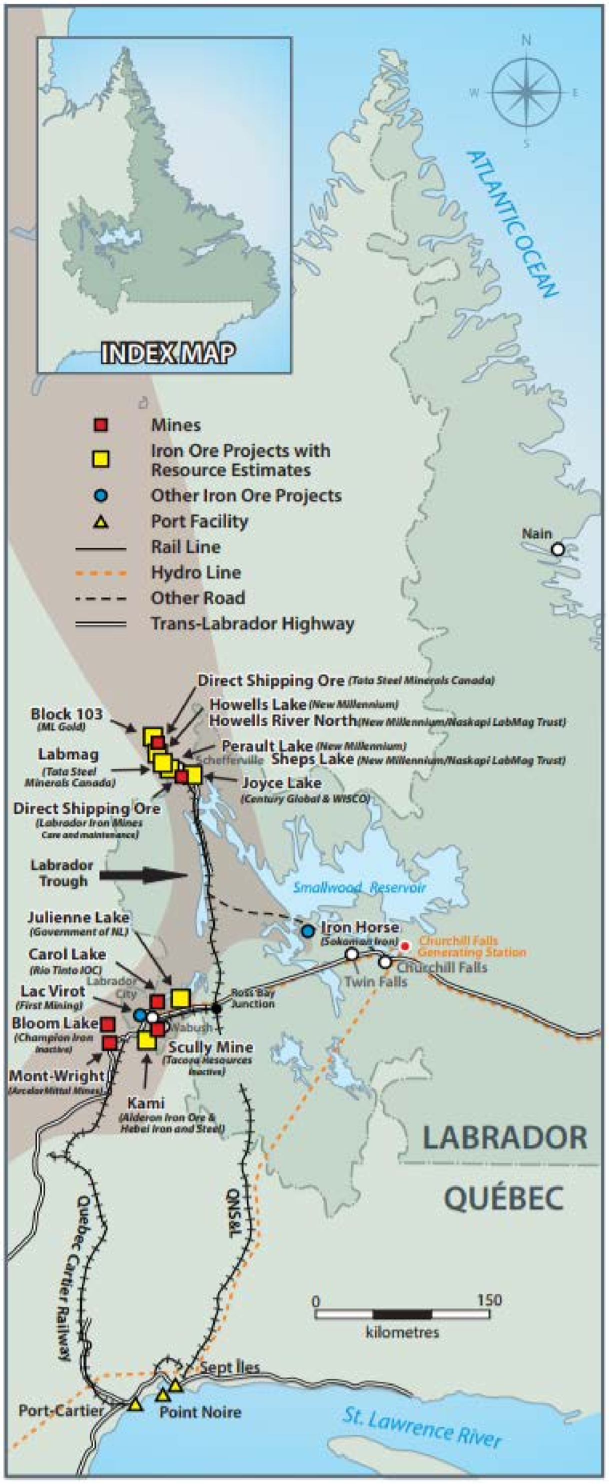
4 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Description & Location

The Scully Mine is located in the south-west corner of Labrador, three (3) km from the town of Wabush. Geographically, the mine is located approximately between 52°53.5' and 52°55.5' latitude and 66°54' and 66°58' longitude. The open pits are located west of the town of Wabush and are reachable via the plant access road off Highway 503 while the tailings disposal area (Flora Lake) is situated east of the town. The deposit is part of the Labrador Trough and covers an area of approximately 23 km^2.

Many other iron ore mines are located in proximity of the Scully Mine: to the west, in the Quebec Province, are the Mount-Wright Mine and the Bloom Lake Project; to the north of the Scully Mine, in Labrador, is the IOC Mine (Figure 4.1).

Figure 4.1: Project Location



4.2 Mineral Titles

Surface control is a combination of owned real property and leased surface rights. There are no federal lands, including national parks or Canadian Forces bases proximate to the Project area and the Project is wholly located within the province of Newfoundland and Labrador. The mining lease locations (Figure 4.2) and their land control are summarized in Table 4.1.

Figure 4.2: Mining Lease Locations

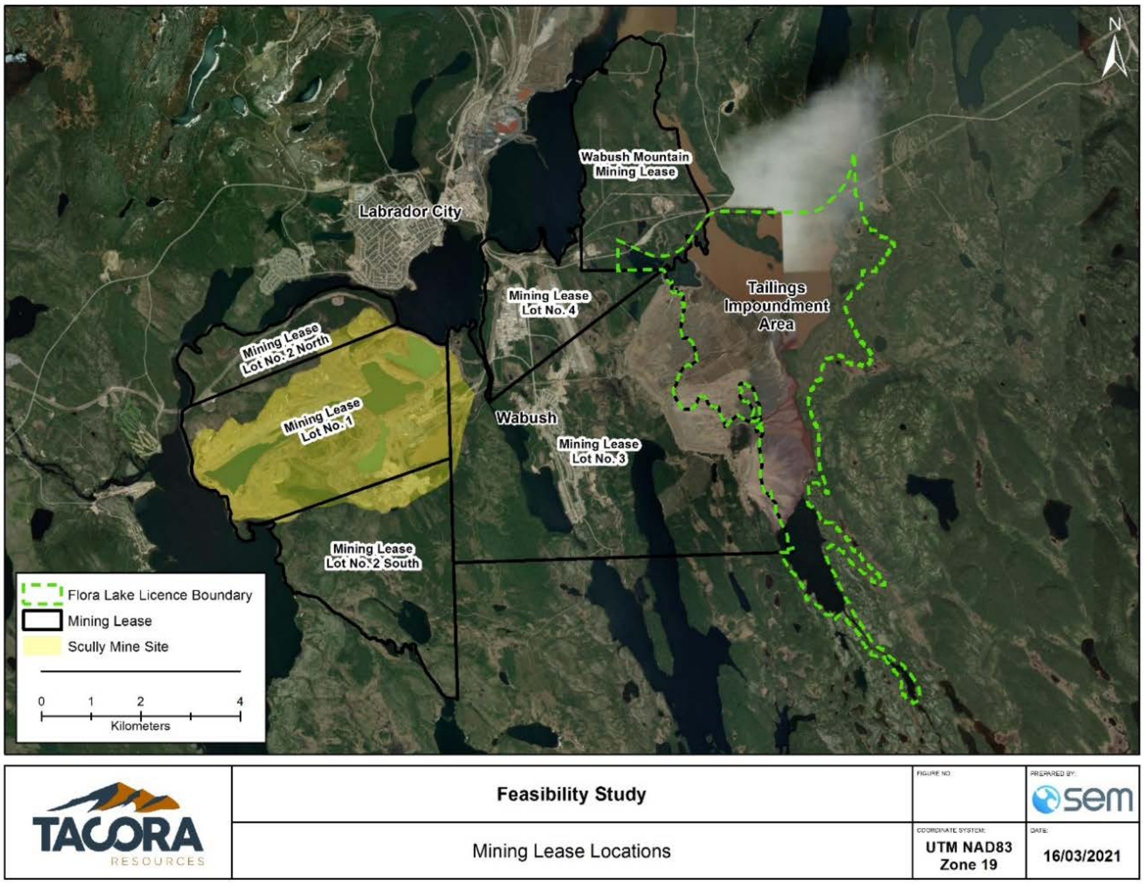


Table 4.1: Land Control Summary

Description	Original Size (ha)	Surface	Minerals
Lot #1	1,450.4	Leased	Included
Lot #2	1,517.8	Owned	Leased
Lot #3	2,688.5	Owned	Leased
Lot #4	595.7	Owned	Leased
Wabush Mountain Area	911.7	None	Leased

4.2.1 Lot #1

The industrial site and open pits are located within the area shown on Figure 4.2 as "Mining Lease Lot # 1". The surface and mineral rights on this parcel are leased from the Government of Newfoundland and Labrador. This 99-year lease will expire in 2055. The lease history has two aspects: the leases themselves and the various mergers and acquisitions of lease parties.

The original Lot #1 lease was issued from the Province of Newfoundland (the "Crown") to the Newfoundland and Labrador Corporation Limited ("NALCO") in May 1956. NALCO immediately signed a lease with Canadian Javelin Limited ("Javelin") for the same lands and minerals, effectively passing the Crown's Lot #1 lease to Javelin. In the fall of 1959, Javelin further passed the Crown's Lot #1 lease to Wabush Iron Co., Limited ("Wabush Iron") and thereby consolidated some other leases into a single, overarching document for the surface use and mineral rights to Lot #1. These various agreements have been amended from time to time, but the overall lease path remains as described.

Over the ensuing years, mergers, acquisitions and divestitures have changed the names of these legal entities, but the lease path remains the same. The NALCO interest is now controlled by Knoll Lake Minerals ("Knoll Lake"). Javelin's position is owned by an affiliate of MFC Bancorp, Ltd. ("MFC"). Wabush Iron's interest now belongs to Tacora Resources Inc. ("Tacora").

4.2.2 Lots #2, #3 and #4

As part of the Assets Purchase Agreement ("APA"), Tacora obtained the surface rights to all of Lots #2, #3, and #4 "excepting all portions of that real property that have been sold, conveyed, or assigned…to any third parties… registered in the Registry of Deeds for Newfoundland and Labrador". The previous owners had been selling real property to third parties for several years, and any unsold residential or commercial property within the municipal boundary of Wabush were explicitly excluded from the APA.

The majority of the land area that has been removed from Lots #3 or #4 over time have been residential and commercial properties within the Wabush municipal limits. A portion of Lots #2, #3 and #4 had been sold to Quebec Iron Ore Co. when they purchased the Bloom Lake facility and associated rail spur which crosses the northern portion of these lots.

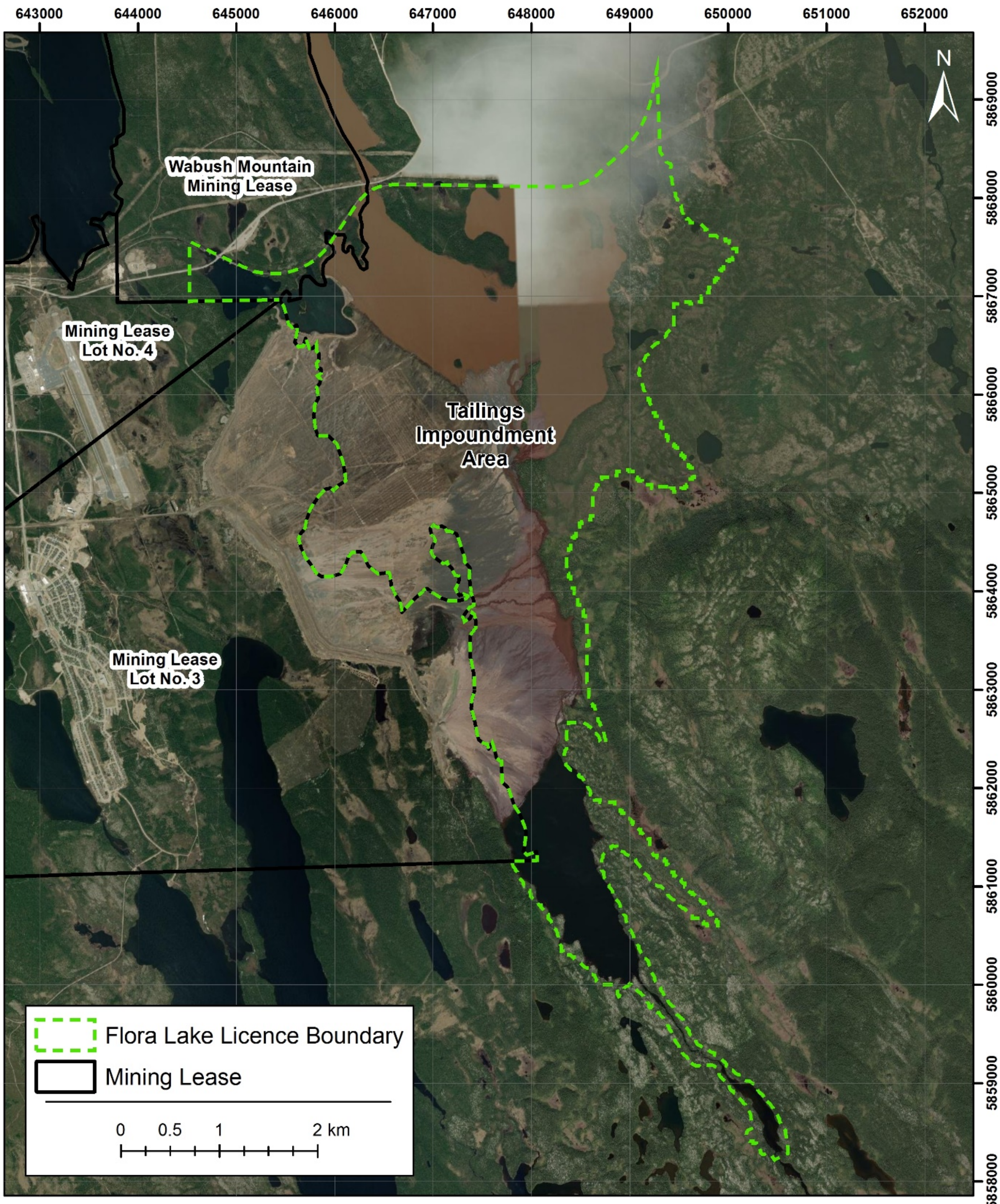
4.3 Wabush Mountain Area

Tacora did not acquire the surface rights to what is referred to in Figure 4.2 as "Mining Lease Wabush Mountain". The previous owner gave up those surface rights in December 2016. Mineral rights are described in Table 4.1.

4.3.1 Flora Lake Tailings Impoundment Area

The tailings storage facility shown on Figure 4.2 and further depicted in Figure 4.3 with 2017 aerial imagery is formally called the Flora Lake Tailings Impoundment Area ("TIA") within various regulatory approval documents. The boundaries of this TIA are set out in the Flora Lake Tailings Licence dated May 15, 1962. The Licence was granted to NALCO for a term of 99 years coincident with the original Lot #1 lease and expires in 2055. The Licence was assigned from NALCO to Javelin and then to Wabush Iron, each in 1962.

Figure 4.3: Tailings Impoundment Area



TACORA RESOURCES	Feasibility Study	PAGE NO: n/a	PREPARED BY: sem
	Flora Lake Licence Boundary	COORDINATE SYSTEM: NAD 83 UTM Zone 19	DATE: 26/03/2021

4.4 Mining Lease #1

Mineral rights underlying the surface of Government Lot #1 are included in the original surface lease which has been described above in the section relating to surface rights. These mineral rights are retained by Tacora until 2055.

4.5 Wabush Mountain Area

The Crown leased mining rights to NALCO on May 15, 1962. As was done with other leases, NALCO immediately assigned this lease to Javelin. Javelin subsequently assigned the lease to Wabush Iron, and then it was included in the APA.

4.6 Tailings

The Flora Lake Tailings Licence grants the licensee the right to deposit into Flora Lake tailings and other waste materials resulting from the mining operation conducted by the licensee and to recover and remove iron ore from any tailings and other waste materials deposited by the licensee.

4.7 Permits

The reactivation project has been released from the environmental assessment process and necessary approvals, permits and authorizations were obtained or amended prior to reactivation. In addition, throughout the operation and future expansions, compliance with various conditions, commitments and standards contained in federal and provincial legislation, regulations and guidelines will be required. Section 20 of this report outlines the permits, approvals and authorizations that are in place or will be required for continued operation and additional tailings expansion.

4.8 Royalties, Agreements, Encumbrances

4.8.1 Mineral Royalties

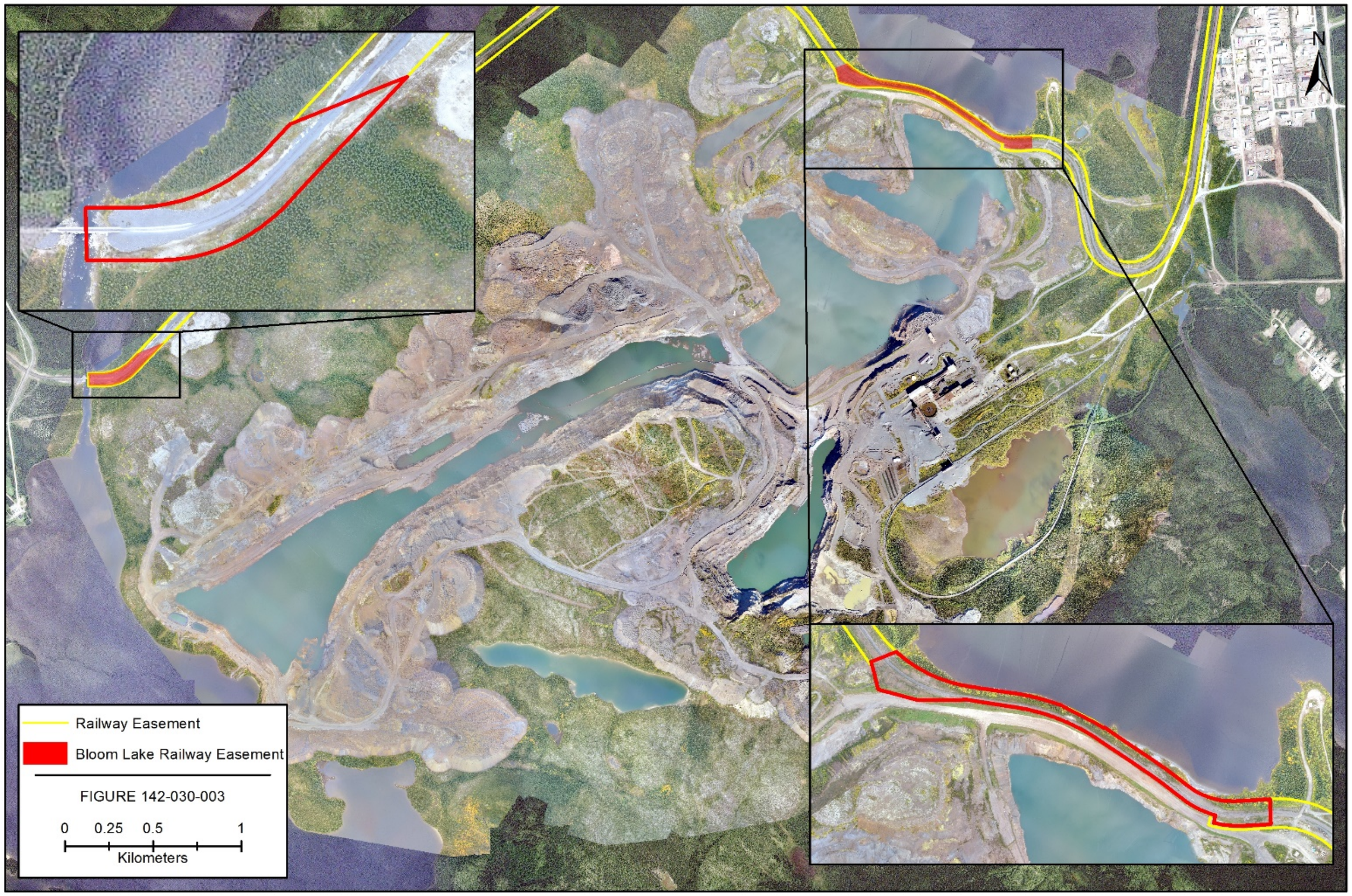
Mineral royalties on ore and any reprocessed tailings follow existing lease agreements as identified for Lot #1 surface rights. Tacora pays a royalty to MFC Industrial, who pays an agreed amount to Knoll Lake, who in turn meets the requirements of the original Crown lease (as amended).

4.8.2 Encumbrances

When the Bloom Lake mining site was developed, a railway spur was constructed along the northern portion of the Scully Mine site. Portions of this railway are located on Lot #1, and thus there is an existing easement for two (2) portions of the Bloom Lake Railway. These easements do not affect the mining plans for the Scully Mine. Figure 4.4 shows these easements overlaid on a 2017 aerial photograph.

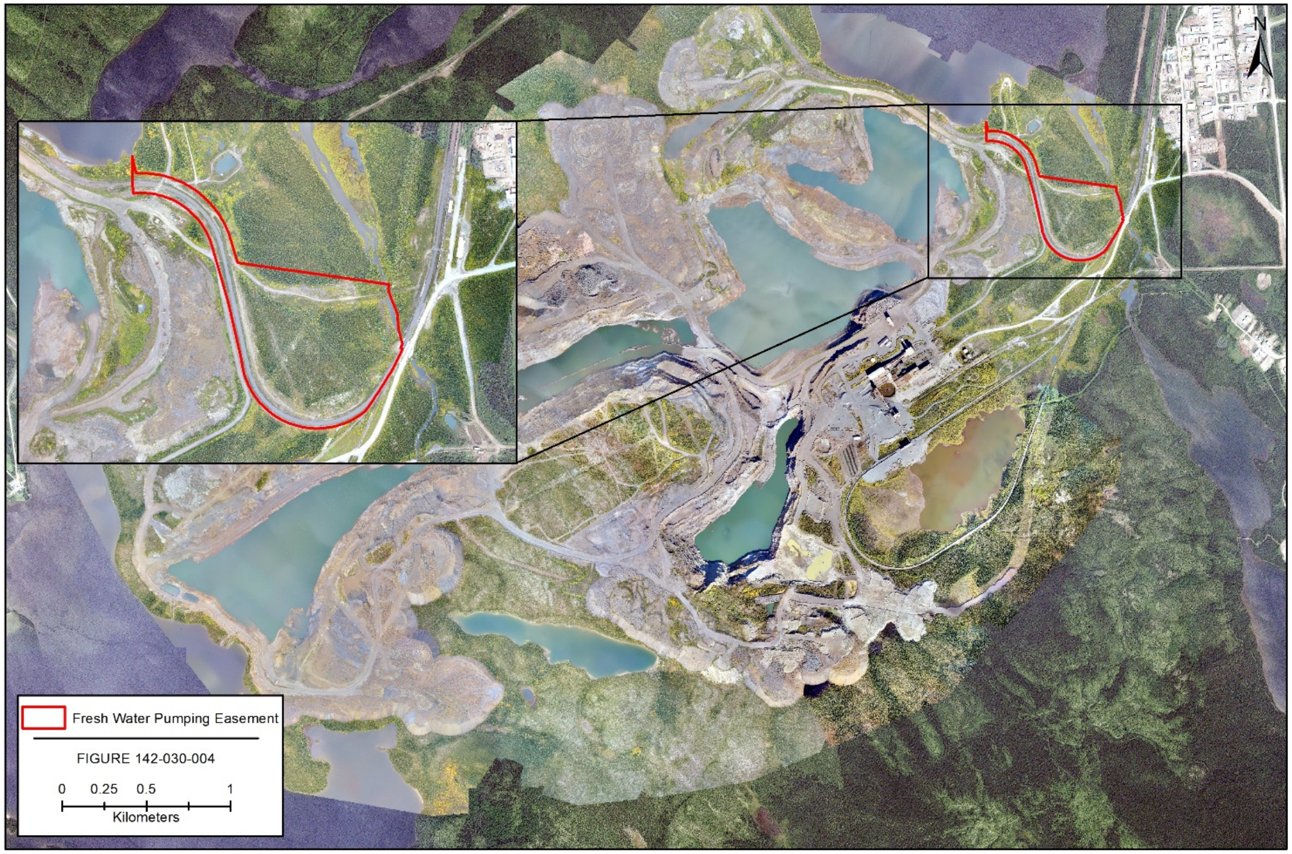
Similarly, Tacora has an easement across a land parcel that was sold in 2016 for part of the Bloom Lake railway. The easement is required for the access and operation of the fresh water pumping system that existed prior to the construction of the Bloom Lake railway. Figure 4.5 shows this existing infrastructure on a 2017 aerial photograph overlaid by the parcel boundary.

Figure 4.4: Bloom Lake Rail Easement



TACORA RESOURCES	**Feasibility Study**	FIGURE NO:	PREPARED BY: sem
	Bloom Lake Rail Easement	COORDINATE SYSTEM: **UTM NAD83 Zone 19**	DATE: **16/03/2021**

Figure 4.5: Fresh Water Pumping System Easement



TACORA RESOURCES	**Feasibility Study**	FIGURE NO:	PREPARED BY: sem
	Fresh Water Pumping System Easement	COORDINATE SYSTEM: **UTM NAD83 Zone 19**	DATE: **16/03/2021**

4.9 Environmental Liabilities

4.9.1 Potential Environmental Liabilities

Potential Environmental risks to the Project are noted below.

4.9.1.1 Stack Emission Testing

Under the Certificate of Approval ("CoA"), Tacora is currently required to complete stack emission testing and dispersion modelling every two (2) years. Upon demonstrating that the Mine meets the guidelines, testing and modelling can be completed every four (4) years. The Tacora Environmental Assessment Registration document committed to completing stack testing within 12 months of resuming operations which was June 2019. Stack testing has yet to be completed. A request was submitted to NL Department of Environment, Climate Change and Municipalities ("DECCM") in December 2020 to postpone the first round of stack testing until Spring 2021. This is to allow improvement to the APC systems prior to stack testing. The last round of stack testing was completed at the Mine between September 2011 and January 2012. Results from this test program were used in a dispersion model, with results reported in January 2014. The modelling report identified that the Mine was in exceedance for ground-level dust concentrations.

To meet the Approval requirements, source emission testing will be completed in 2021 and every subsequent two (2) years, however if in compliance, this frequency is reduced to every four (4) years. These results will be used for air dispersion modelling of the site for submission to the NL government. This submission will support the ongoing requirements to demonstrate compliance with provincial air emissions regulations.

4.9.1.2 Schedule 2 Amendment

To continue mining activities to 2044, an expansion of the tailings facility is required. To expand the tailings outside the existing tailings basin, an amendment is required to the existing Schedule 2 regulation. The Schedule 2 amendment process involves the regulatory approval of provincial and federal agencies and the engagement with Indigenous groups. Due to the scale of the project and large number of sequential approval steps involved, there is a risk that the Project will be delayed, or alternative solutions will be explored.

To better position for approval, Tacora has brought together an experienced, multifaceted team that is in dialogue with regulators. The team will prepare an environmental assessment registration and complete robust field studies in 2021 that will provide key information on how to best structure the project moving forward. The team will also review past 'alternatives assessments' to determine whether alternative tailings disposal options are available and whether new assessments are required. The process for achieving approval may take more than three (3) years.

4.9.2 Potential Environmental Opportunities

Potential environmental opportunities to the Project are listed below.

4.9.2.1 Reducing Frequency of Testing for a Parameter in the Effluent Discharge Criteria ("EDC")

To ensure compliance with Provincial and Federal Regulations, Tacora has an Effluent Monitoring Program. The Effluent Monitoring Program includes effluent sampling, analysis and reporting of results to both the NL DECCM as well as Environment and Climate Change Canada ("ECCC").

Both the CoA and the Metal and Diamond Mining Effluent Regulations ("MDMER") present the opportunity to reduce the frequency of testing for certain parameters. Tacora will implement reduced testing when possible for deleterious substances that meet the reduction criteria. An effort will be made to reduce testing for Arsenic, Copper, Lead, Nickel, Zinc, Radium 226, and Acute Lethality Testing ("ALT") at all final discharge points.

4.9.2.2 Reduce Water Withdrawal from Little Wabush

Tacora currently has two (2) Water Use Licences ("WUL") that permit water withdrawal from Little Wabush Lake and dewatering from the mine pits. Both licences require annual reporting of water withdrawal and usage amounts, which then presents a cost to the business as Tacora is charged under the Water Use Charge Regulations NLR 60/16.

Tacora plans to implement a water recycling project which will allow water from the mine pits to be used as process water. The project will reduce both the amount of water withdrawn from Little Wabush Lake and the amount of water discharged into various receiving water bodies. Tacora will be able to recycle water while reducing WUL fees.

4.9.2.3 Habitat Banking

Tacora will be required to offset fish habitat losses resulting from tailings expansion to support continued mining. Fisheries and Oceans Canada ("DFO") administers the Fisheries Act to ensure fish habitat offsetting is achieved. Recent revisions to the Fisheries Act have expanded fish habitat banking as a viable offsetting option. Fish habitat banking is considered a preferred method for offsetting because positive environmental outcomes are achieved before habitat loss occurs. Tacora will have the opportunity, by offsetting the loss of fish habitat due to tailings expansion, to develop a fish habitat bank to offset future habitat losses. The banked habitat would provide Tacora with regulatory compliance certainty and possibly provide economic benefits by creating the additional habitat at reduced cost.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

The Town of Wabush is located 320 km north of Sept-Iles, Quebec on the north shore of the St-Lawrence River. St. John's, Newfoundland is 1,200 km southeast of Wabush while Montreal is 1,016 km to the southwest. Labrador City is 6.4 km from Wabush and Fermont, Quebec is 32 km west of Wabush. The process facility is accessed by 2.5 km plant access road off the 503 Highway.

The Trans Labrador Highway and Route 389 connect the region to Quebec and Eastern Labrador by road. The roads connect Labrador West to Baie Comeau and Goose Bay with 8-hour and 6-hour trips, respectively. These routes are remote, with no communications infrastructure and limited rest and service areas.

5.2 Climate

Wabush has a subarctic climate with precipitations much higher than usual for this type of climate due to the persistent Icelandic Low, which gives the region some of the rainiest and snowiest weather in all of Canada. In summer, cloudiness is very high due to the lakes nearby and the unstable northerly airstreams that prevail. In a wet year, Wabush can receive up to 1,185 mm of precipitation (Environment Canada, 2012). Conversely, in a dry year, Wabush receives only 675 mm of precipitation. The temperatures range from -40ºC to +25ºC.

5.3 Local Resources

The towns of Wabush and Labrador City are well established with populations of 1,906 (2016) and 7,220 (2016), respectively. The unemployment rate of Wabush and Labrador City are of 11.1% and 8.5% respectively with an unemployment rate of 13.7% across the province of Newfoundland and Labrador (these statistics are taken from the 2016 Census from Statistics Canada). The two (2) towns are located 5 km apart from one another and contain the infrastructures and necessities to house the employees and their families who live there including, indoor shopping centers, hotels, lower, middle and high school, a community center and a hospital. Air transportation needs for the communities are served daily by the Wabush Airport.

5.4 Infrastructure

5.4.1 Roads

Various mine roads on the property provide access to all the mining areas and waste dump areas. The Scully Mine Tailings Line Road connects the process plant to the tailings storage facility and crosses the 503 Highway.

5.4.2 Process Facility

The process facility complex includes crushers, mills, spirals, hydro-sizers, dryers, separation units, load out bins, rail infrastructure and maintenance shops. The process facility is further described in Section 17 on recovery methods.

The re-start of the concentrator required the refurbishment of equipment and buildings and the installation of manganese reduction lines. Tacora has carried this out at the issuance of the current update of the NI 43-101 Report.

5.4.3 Services Buildings & Ancillary Facilities

Service buildings and ancillary facilities exist and are in operation. Engineering inspections should be performed on a regular basis to ensure the buildings are in good condition to continue operation. These infrastructures include:

- Mine fleet maintenance facility with 5 bays and 50 t overhead crane.
- Wash bay within the maintenance facility.
- Auxiliary mine maintenance facility with 7 bays and 10 t overhead crane.
- Warehouse.
- Machine shop, electrical shop, paint shop and welding shop.
- Explosives storage magazines.
- Mine dewatering equipment and sedimentation ponds.
- Fuel storage tanks.
- Administration building.

5.4.4 Power Supply & Distribution

The mine site is connected to the Newfoundland & Labrador Hydro electrical network. Power is generated at Churchill Falls, 200 km to the East. The Churchill power station has the second largest hydroelectric generating capacity in North America at 5,428 MW installed.

A 46 kV electrical grid on site electrifies the mine area powering mine equipment and pumping stations.

5.4.5 Water Supply & Tailings Storage

A pumping station and water intake structure located east of the process facility on Little Wabush Lake provides water for iron ore beneficiation and potable water consumption. In 2019, 9.3 Mm^3 of water was withdrawn from Little Wabush Lake and during a full year of operation in 2020, 17.9 Mm^3 of water was withdrawn.

5.4.6 Rail & Port Infrastructure

The Mine and Port facilities are connected by a railway. The railway (Figure 5.1) travels from the mine to a connection with the Quebec and North Shore & Labrador ("QNS&L") Railway, owned by the Iron Ore Company of Canada ("IOC"), at Wabush, NL by Arnaud's sister railway, the Wabush Lake Railway. QNS&L operates the railway between Wabush, NL and Arnaud Jct., QC. At Arnaud Junction, Chemin de fer Arnaud owns the segment to Pointe-Noire, Quebec. The railway is owned by Rail Enterprises, Inc., a division of IOC.

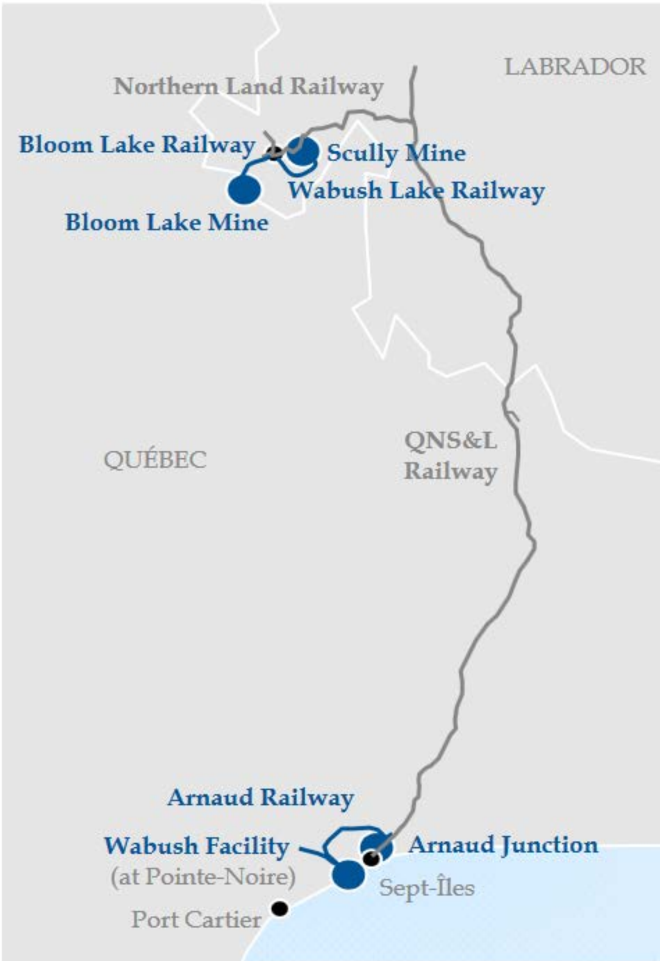
The Wabush railways are comprised of two Canadian short line railways:

- Wabush Lake Railway is 3 km long and connects the Scully Mine to the Northern Land Railway at Wabush Junction.
- The Arnaud Railroad is a 35 km common carrier, which connects to the QNS&L Railway northeast of Sept-Îles at Arnaud Junction and terminates at the Wabush port complex at Pointe-Noire. The railroad primarily transports iron ore and has the potential to transport other commodities with total capacity of 20 Mtpy.

The Arnaud Railway and port infrastructure are also a critical piece of infrastructure to other neighbouring mining operations in the Labrador Trough which need to transport materials to, and / or from, Pointe-Noire. The Arnaud Railway is the only railway connecting Pointe-Noire and the QNS&L Railway at the Arnaud Junction.

The dock facility at Pointe-Noire provides access to Dock 30 and 35 with more than 50 Mtpy of capacity and the capability to load up to 150,000 t vessels at Dock 30, and up to 400,000 t vessels at Dock 35.

Figure 5.1: Rail Network



5.5 Physiography

The mine is located in an area of undulating hills that reach elevation highs of 686 m and low areas with elevations of approximately 533 m. The ground cover consists of barren rock, marsh, and coniferous forests.

There are several lakes within the area of the mine or the adjacent boundaries including Jean Lake, Knoll Lake, Flora Lake, Wahnahnish Lake, Little Wabush Lake, Harrie Lake, Quartzite Lake, Vern Lake, and Long Lake.

6 HISTORY

6.1 Prior & Current Ownership

The Scully Mine was operated by PM, a subsidiary of Moore-McCormack Resources from 1965 to its 1986 when PM was acquired by Cleveland Cliffs Inc. ("Cliffs") who operated it from 1986 until being put on care and maintenance in February 2014. For most of its life, the mine was a joint venture owned by Stelco (37.9%), Dofasco (24.3%), Inland Steel (15.1%), Acme Steel (15.1%) and Cliffs (7.7%), but after various mergers and acquisitions in the North American steel industry the ownership was consolidated between Cliffs, ArcelorMittal and U.S. Steel Canada whereby each partner owned with joint venture percent ownerships of 26.8%, 28.6% and 44.6%, respectively. Cliffs exercised their right of first refusal in February 2010 to acquire 100% ownership of the property for approximately USD 88 million.

Cliffs shutdown the pellet plant in May 2013 followed by the mine and concentrator in February 2014. The closure was due to increased costs, reduced production rates and a dramatic decrease in seaborne iron ore prices combined with a decrease on pellet premium pricing. On November 19, 2014, Cliffs announced the pursuit of exit options from its Eastern Canadian iron ore operations and on May 20, 2015, Cliffs commenced restructuring proceedings under the CCAA, ultimately selling the Scully Mine and other Eastern Canadian assets. The Scully Mine was sold to Tacora Resources Inc. ("Tacora") in July of 2017. The APA was executed through a court supervised process under the CCAA.

6.2 Exploration & Development History

Iron ore deposits were first reported in the Wabush area in 1933. It was only in 1956, when PM became interested in the deposit that field work began. In 1957, intensive geological, metallurgical and economic investigation of the property started. A temporary camp was established and the construction of a 700 tpd pilot plant was started in 1959. During 1960 and 1961 the pilot plant produced 100,000 t of concentrate for blast furnace tests.

In 1961, contracts were awarded for the construction of a processing plant and related infrastructures with a capacity of 5.4 Mt of concentrate. Around the same time, harbor facilities construction began in Pointe Noire, Sept-Iles. The plan was to ship the ore as sinter feed; however, in 1963, a decision was made to build a 4.9 Mt capacity pelletizing plant, also in Pointe Noire. In 1965, both the Wabush and Pointe Noire facilities were completed and inaugurated in the presence of government officials. In 1967, the pelletizing plant capacity was expanded to its actual capacity of 6.0 Mt. The Scully Mine was named after Mr. V.W. Scully, former chairman of the board of Stelco Inc.

6.3 Historic Drilling

Overall, 1,005 drill holes were completed between 1969 and 2013 for a total of 77,622 m. A more complete description of the drill programs is described in Section 10.

6.4 Historical Production

The Scully Mine operated continuously from 1965 to February 2014 with the mining and concentrating at Wabush and the subsequent stage of pelletizing done at Pointe Noire near Sept-Iles. Table 6.1 presents the historical production at the Scully Mine.

The Scully Mine was first operated during the 1960's without regard to the overall manganese content of the concentrate produced. This continued into the 1980's when concentrate commercial specifications required a much lower manganese content in the final product. To achieve this lower manganese content, mine operations moved away from a blend consisting of both Middle (low manganese) and Lower Member ores (higher manganese) to one predominately made up of Middle Member ores. Problems achieving the required manganese content continued despite moving to a mostly Middle Member blend until mining shifted into the West Pit Extension during the 1990's. Cliffs recognized that operations based solely on lower manganese Middle Member ores would limit the life of the mine and thus, started extensive investigations into finding a processing approach which would solve this issue. This ultimately led to the manganese reduction circuit developed by Cliffs.

One consequence of switching to a middle member blend was softer ores than historically processed at Wabush. Once blasted, they contained mostly sand sized particles with a limited amount of coarser material. Given that the plant uses autogenous grinding, according to certain reports the decrease in the amount of coarser material negatively impacted on the concentrator performance.

The Scully Mine was operated until February of 2014 at which time the mine was idled, and all workers were laid-off.

6.5 Historic Resource Estimate - Cliffs

Mineral Resource Estimates (`MRE`) were published by Cliffs in 2013 for the Wabush property. When the decision was made to idle Scully operations, all previously reported Mineral Reserves were reclassified as mineralized material.

In their 2012 Annual Report, Cliffs stated a Measured and Indicated Mineral Resource of 200.4 Mt at 35.1% iron based on a cut-off grade of 25% weight recovery (or 16.5% Fe).

Table 6.1: Historical Production (1965 - 2014)

Year	Iron Ore mined (kt)	Waste Mined (kt)	Final Concentrate Production (kt)
2014	N/A	N/A	300
2013	8,880	5,602	2,535
2012	10,504	5,714	3,016
2011	10,068	7,072	3,122
2010	11,733	5,043	3,776
2009	7,313	2,945	2,662
2008	12,378	8,133	4,158
2007	13,172	8,125	4,638
2006	12,916	11,628	4,209
2005	16,267	9,426	5,306
2004	11,411	4,780	3,728
2003	14,005	7,195	4,974
2002	12,726	8,555	4,482
2001	12,202	8,207	4,419
2000	17,412	10,784	5,836
1999	15,887	9,316	5,337
1998	16,947	10,403	5,689
1997	15,051	8,893	5,122
1996	16,846	8,416	5,659
1995	17,523	5,446	5,344
1994	15,489	6,300	4,835
1993	14,313	5,367	4,554
1992	13,778	4,618	4,519
1991	14,306	5,220	4,595
1990	18,914	6,895	5,702
1989	19,483	6,011	6,050
1988	19,574	5,027	5,979
1987	16,068	4,801	4,948
1986	13,911	6,566	4,582
1985	16,410	7,369	5,409
1984	16,799	7,996	5,535
1983	12,153	6,280	4,117
1982	10,014	5,258	3,224
1981	16,632	8,655	5,581
1980	16,244	9,014	5,064
1979	17,651	6,395	6,062
1978	12,611	5,582	3,676
1977	17,362	6,014	5,514
1976	17,399	6,258	5,523
1975	9,133	3,409	2,870
1974	17,336	5,511	5,599
1973	17,989	4,737	5,544
1972	15,251	3,211	4,384
1971	17,751	2,688	5,743
1970	16,554	2,386	5,672
1969	10,155	2,475	3,400

Year	Iron Ore mined (kt)	Waste Mined (kt)	Final Concentrate Production (kt)
1968	16,080	2,961	5,200
1967	13,858	2,552	5,180
1966	10,745	1,271	3,902
1965	7,852	1,225	2,533

6.6 Historic Resource Estimate – G Mining Services

As part of the 2018 Feasibility Study ("FS"), G Mining Services ("GMS") published Mineral Resources for the Scully Mine using the data provided by Tacora Resources' representative at the time. The resource was effective December 6th, 2017 and the Qualified Person was Mr. Réjean Sirois, P. Eng. The geological wireframes were constructed using sectional interpretation derived from Cliffs geologists, and the estimation was bound by the geological units and sub-units of the Lower, Middle- and Upper-Member Iron Formations. Crude grades were estimated from exploration drill holes using Ordinary Kriging, and concentrate grades were estimated using Inverse Distance interpolation methods. Blastholes were included in the first estimation pass to assist in local accuracy.

The combined Measured and Indicated Mineral Resource for the Scully Deposit was estimated to 734 Mt at an average grade of 34.6% Fe and Inferred Mineral Resource to 237 Mt at an average grade of 34.1% Fe (Table 6.2). The Mineral Resources were reported within the conceptual open pit shell at a cut-off grade of 20% Fe, using a long-term iron price of USD 79/dmt. Only rock codes 22, 31, 32, 33, 34, 51, 52 and 53 were included in the Mineral Resource.

Table 6.2: Mineral Resource Estimate by GMS, Effective December 9th, 2017

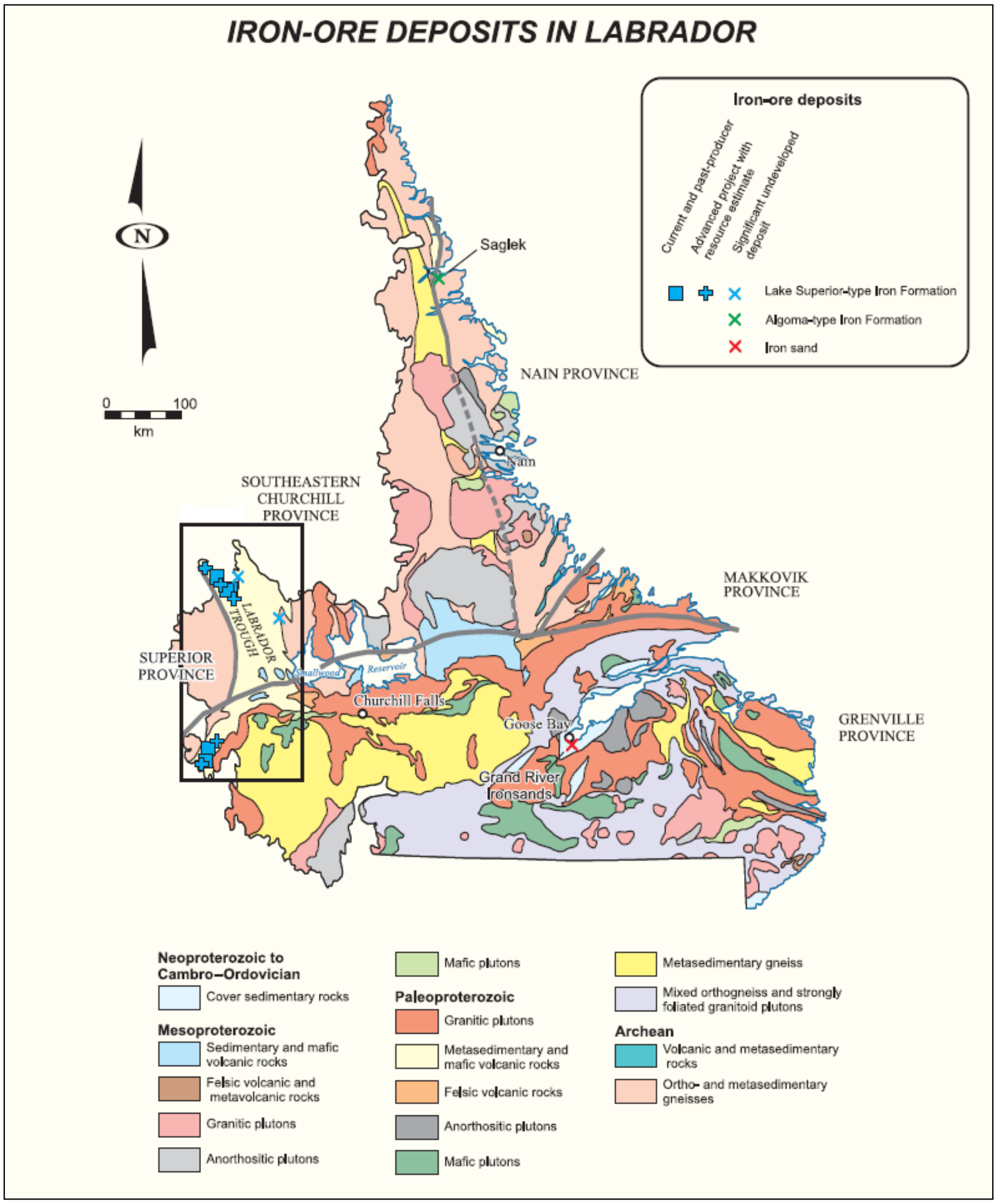
Classification	Tonnage (dry)	Fe	Mn	SAT	WTREC	Fe Conc.	SiO_2 Conc.	Mn Conc.	Sat Conc.
	kt	%	%	%	%	%	%	%	%
Measured	213,650	35.1	2.3	5.6	36.6	64.5	3.6	2.0	6.8
Indicated	520,760	34.3	2.4	5.8	35.4	63.5	3.8	2.5	7.7
Total M&I	734,410	34.6	2.4	5.7	35.7	63.8	3.8	2.4	7.4
Inferred	236,973	34.1	2.1	5.8	34.6	64.0	3.8	2.1	8.9

7 GEOLOGICAL HISTORY AND MINERALIZATION

7.1 Regional Geology

The Scully Mine iron deposit lies at the southern end of the Labrador Trough within the geological Grenville Province. Figure 7.1 shows the regional geological location of the Scully Mine (Conliffe et al., 2012). The Labrador Trough extends along the margins of the eastern boundary of the Superior-Ungava craton for more than 1,200 km and is up to 75 km wide at its central part. The Scully Mine Deposit is located within the Parautochton Deformation Belt of the Grenville Province of the Canadian Shield, just south of the Grenville Front. The Grenville Front, the northern limit of the Grenville Province, truncates the Labrador Trough, separating the Churchill Province greenschist metamorphic grade part of the Labrador Trough rocks from the highly metamorphosed and folded amphibolite to granulite metamorphic grade rocks, which are their equivalent in the Grenville.

Figure 7.1: Simplified Regional Geological Map (J. Conliffe et al., 2012)

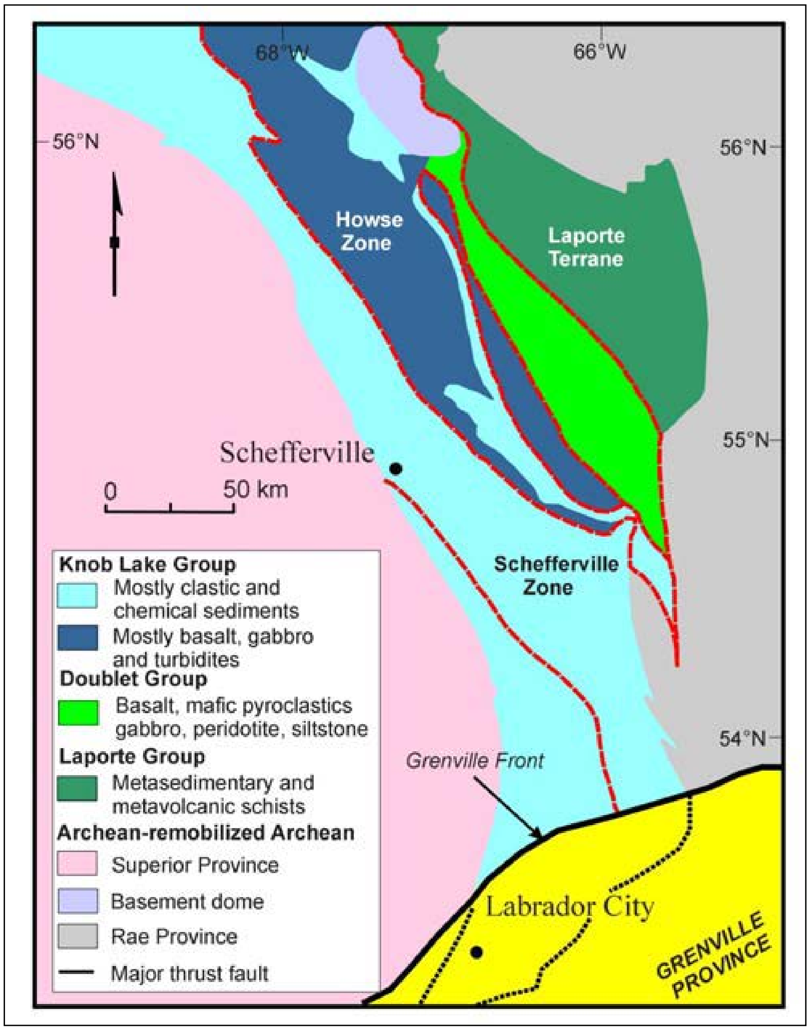


The western half of the Labrador Trough, consisting of a thick sedimentary sequence, can be divided into three (3) sections based on changes in lithology and metamorphism (north, central and south). The Trough is comprised of a sequence of Proterozoic sedimentary rocks including iron formations, volcanic rocks and mafic intrusions known as the Kaniapiskau Supergroup. The Kaniapiskau Supergroup consists of the Knob Lake Group in the western part of the Trough and the Doublet Group, which is primarily volcanic, in the eastern part. The Kaniapiskau Supergroup within the Grenville is highly metamorphosed and complexly folded and is named the Gagnon Group. It occurs as numerous isolated segments. From the base to the top, it includes a sequence of gneisses and schists, a group of chemically precipitated sediments, and more schists, including some distinctive aluminous varieties. Gabbro sills intrude parts of the Gagnon Group, and granites are found in the gneiss.

The Central or Knob Lake Range section extends for 550 km south from the Koksoak River to the Grenville Front located 30 km north of Wabush Lake. The principal iron formation unit, the Sokoman Formation, part of the Knob Lake Group, forms a continuous stratigraphic unit that thickens and thins from sub-basin to sub-basin throughout the fold belt.

The southern part of the Trough is crossed by the Grenville Front. The rocks in the Grenville Province to the south are highly metamorphosed and complexly folded. Iron deposits in the Grenville part of the Labrador Trough comprise Bloom Lake, Lac Jeannine, Fire Lake, Mounts Wright and Reed, and the Luce, Humphrey and Scully Deposits in the Wabush area. The high-grade metamorphism of the Grenville Province is responsible for recrystallization of both, iron oxides and silica in primary iron formation, producing coarse-grained sugary quartz, magnetite, specular hematite schists (meta-taconites) which are of improved quality for concentrating and processing. Figure 7.2 shows the simplified geological map of the Labrador Trough.

Figure 7.2: Simplified Regional Geological Map of the Labrador Trough (from Gross, 2009)



In the region, at least two (2) stages of deformation are recognized. The first stage produced linear belts which trend northwest, such as the well-defined structural trends in the central part of the Labrador geosyncline; the second stage formed linear belts which trend east to northeast, parallel with the major structural trends developed in the Grenville Province. Folds now present reflect both stages of deformation in form and orientation. For example, in the Wabush Lake area, folds trend N20ºE and in the central part of the area, around Lamelee Lake and Midway Lake, N35°W. Isoclinal and recumbent folds overturned to the west or southwest are common, and it is inferred that this deformation produced thrust faults striking northwest and dipping east. Structures developed during the earlier stage of deformation are believed to have been similar to those now seen in the central part of the Labrador geosyncline, and it is highly probable that the structures produced by this early stage of deformation in the south and those in the central and northern regions were the result of the same orogeny.

The second stage of structural deformation took place during the Grenville orogeny between 0.8 and 1.2 Ga years ago. Its effects are not so intense north of Wabush Lake near the margin of the Grenville belt as they are throughout the region to the south. Near the margin of the Grenville belt cross-folds trending east or northeast appear to be superimposed on the earlier northwest-trending structures. Around Mount Wright and farther south, the trend of the overall structure is east to northeast and the prevailing dip of foliation is 55°N. Tightly folded and faulted structures developed during the earlier stage of deformation were further deformed by folding and faulting during the Grenville orogeny. Oblique sections through the resulting complex fold structures are exposed at the present erosion surface. Many of the minor folds appear to plunge steeply to the northwest, but the axes of these folded folds are not straight for any appreciable distance.

Regional structures developed during the Grenville orogeny play out against the stable craton area of the ancient Superior Province. Folds and faults along the northwest margin of the Grenville Province trend west, and the general pattern of folds overturned to the south or southeast formed in conjunction with north-dipping reverse faults indicates overriding of the northerly blocks towards the southeast. The relative amount of movement between adjacent fault blocks is suggested by the position of iron-formation in local structures.

The iron-formation and associated metasedimentary rocks, which were derived from an assemblage of continental shelf-type sediments, do not appear to extend south beyond a line trending northeast from the Hart-Jaune River linear to Plaine Lake and northeast to Ossokmanuan Lake. Granite-gneisses, charnockites, and anorthosites are part of the rock assemblage south of this line. These typical deep-seated Grenville rocks may have been thrust northwest along a system of faults that coincide with this line. The large suite of gabbro intrusions in the area between Wabush Lake and Ossokmanuan Lake probably were intruded along faults in this linear zone.

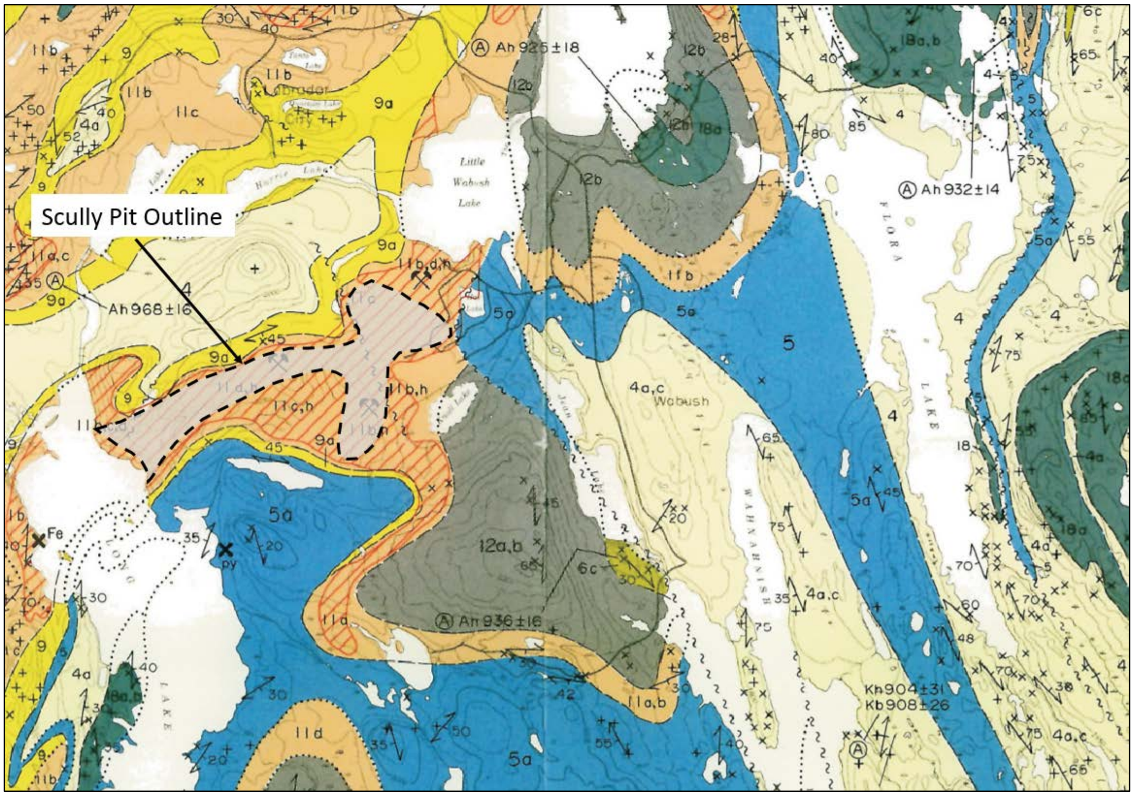
7.2 Local Geology

Information relating to the local geology at Scully is derived from O'Leary (1972) who explains in detail each individual unit and their properties.

7.2.1 <u>General</u>

The Scully Mine lies within the Labrador Trough in Western Labrador. The Sokoman Formation is an iron formation that consists of three (3) iron bearing formations, named the Upper, Middle and Lower Iron Formations. The Sokoman Formation is more than 300 m thick near the Scully Mine and has been subjected to two (2) episodes of folding and metamorphism during the Hudsonian and Grenville orogenies, resulting in a complex structural pattern in the Wabush Area. The younger Menihek and Shabagamo Formations and the older Denault, Attikamagen, and Wishart Formations all outcrop in the vicinity of the mine site. The mineral deposit that defines the Scully Mine consists of folded and faulted stratigraphic beds of iron-bearing units within the regional Sokoman Iron Formation. A local geological map and legend are presented in Figure 7.3.

Figure 7.3: Schematic Geological Map of the Scully Region Showing an Approximate Current Pit Outline



12 — **Menihek Formation:** *12a, dark grey to black schist, phyllite and slate, commonly graphite-bearing; 12b, quartzofeldspathic schist and gneiss, commonly aluminosilicate and/or graphite-bearing.*

11 — **Sokoman Formation:** *11a, carbonate iron formation; 11b, silicate and silicate-carbonate iron formation; 11c, oxide iron formation; 11d, ferruginous quartzite; 11e, quartz-garnet-two amphibole ± pyroxene iron formation; 11f, cherty magnetite greywacke, 11g, cherty magnetite iron formation with tuff bands and fragments; 11h, leached iron formation, original lithotype unknown in some cases.*

9 — **Wishart Formation:** *9a, coarse-grained, white, crystalline quartzite; 9b, pelitic schist; 9c, quartz pebble conglomerate with pelitic schist matrix.*

5 — **Denault Formation:** *5a, dolomitic and calcitic marble with variable content of quartz and calc-silicate minerals, including tremolite, diopside, talc and phlogopite; 5b, dolomitic marble with inter-banded chlorite schist.*

4 — **Attikamagen Formation:** *4a, biotite-bearing quartzofeldspathic schist; 4b, biotite-bearing quartz-K feldspar schist; 4c, migmatitic quartzofeldspathic gneiss; 4d, porphyroclastic augen schist; 4e, metagreywacke-siltstone and slate; 4f, grey to black phyllite.*

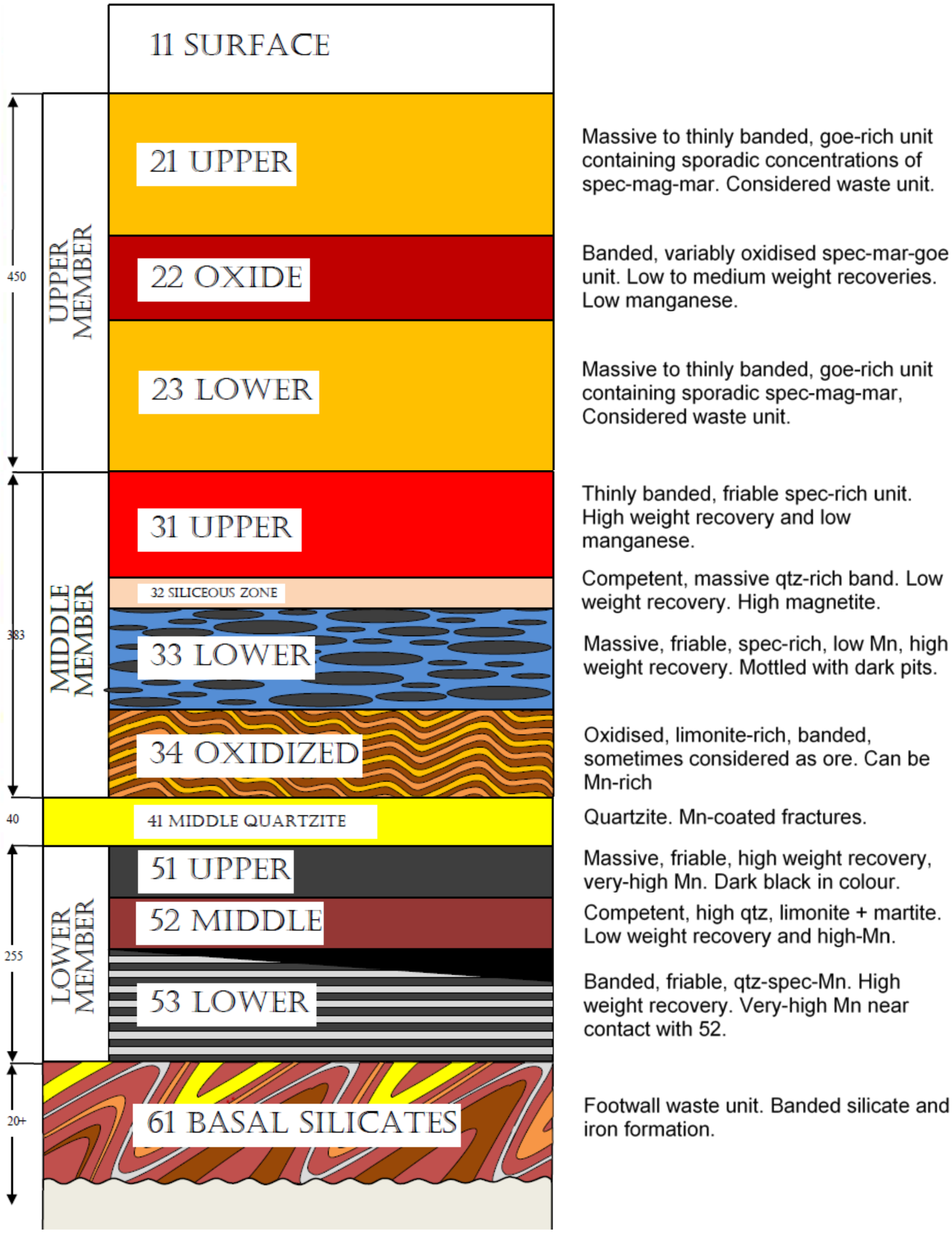
7.2.2 Mineralization

The ore minerals are hematite (specularite), magnetite, and martite. The waste minerals are quartz and hydrated iron oxides such as limonite and goethite. Manganese oxides also occur in bands or are disseminated throughout the iron-bearing units.

A stratigraphic column is presented in Figure 7.4. All units are overlain unconformably by glacial till (overburden) and underlying the Lower Member is a silicate-rich iron formation known as the Basal Silicates (considered the footwall waste unit). Unfolded, the average thickness of the formation is about 800 feet. Each member is split up into separate numeric sub-members, which reflect the changes in ore characteristics (mineralogy, hardness, manganese abundance).

The following information has been sourced from O'Leary (1972), which contains detailed descriptions of each ore member.

Figure 7.4: Stratigraphic Column of the Sokoman Formation at Scully



7.2.2.1 Basal Silicates

Near surface, the basal silicates comprise narrow bands of quartz, limonite and goethite, with pyrolusite in bands running parallel to the bedding. Characteristically, these have high iron assays with very low weight recovery. Fresh samples from drill holes show a layered quartz-amphibole assemblage. Green biotite and chlorite are also common, and garnets become more abundant near the base of the member. The footwall contact is shown in the pit wall in Figure 7.5.

Figure 7.5: Contact Between Basal Silicates (Unit 61) and the Lower Member (53) in the Pit Wall

7.2.2.2 Lower Member Ore

This is generally a hard, dense, blue quartz-hematite rock, with little or no limonite (except where fibrous silicates along the bedding planes of the ore are altered). The ore is distinctly banded with segregation of hematite, magnetite and quartz. Magnetite is common, increasing toward the base of the unit. Vugs are rare and abundant manganese oxides occur in bands and disseminated throughout the ore. Near the base, the layers of hematite and magnetite give way to an amphibole assemblage, which near the surface has altered to limonite. The unit is characterized by intermediate iron and iron unit recovery values. The Lower Member contains an internal thick interbed of limonite-rich ore which presents low weight recoveries (sub-unit 52).

Typical examples of Lower Member Ore are shown in Figure 7.6.

Figure 7.6: Core Photos of Lower Member Ore

7.2.2.3 Middle Quartzite

This is a narrow but persistent band of quartzite which runs the entire length of the orebody. Iron minerals, including limonite and goethite, and more rarely hematite, and pyrolusite fill lenses, joints and cracks. Because of its distinct lithology, this horizon has been widely used as a marker horizon in the structural interpretation of the orebody.

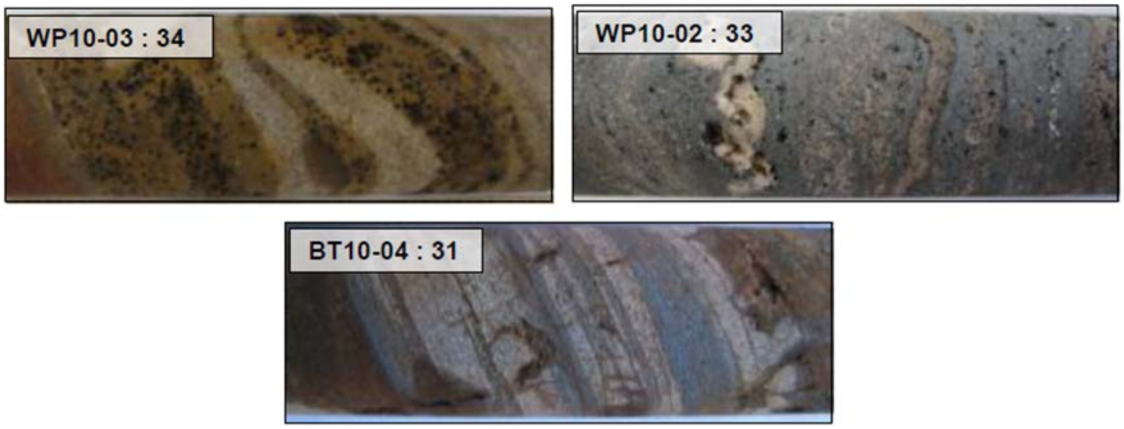
7.2.2.4 Middle Member

This is invariably a coarse-grained, blue specularite-quartz rock. Hematite often makes up more than 50 percent by weight of the rock. The ore is usually soft and friable, breaking down to constituent grains on drying. This is normally explained by intense weathering and removal of silica. Elsewhere, similar leaching of the ores at Knob Lake (O'Leary, 1972) is demonstrated in part by the presence of depressions and crevices filled with lacustrine clays and argillite and containing fossil plants of Cretaceous age. Evidence of leaching of this nature on a similar scale is not found at Wabush Mine.

Figure 7.7: Pit Wall Showing the Contacts Between the Lower and Middle Member Sub-Units, and the Middle Quartzite.

The magnetite content of the member is low but increases rapidly toward the base. Manganese is present as earthy pyrolusite in small vugs which vary in size from half a cm to 5 cm diameter. It also occurs disseminated through the ore, as distinct bands and included in the hematite and magnetite grains, both as separate mineral phases and in solution in the hematite and magnetite crystals. The ore is noticeably banded and has a sub micaceous high lustre. Hematite and quartz are often completely segregated into bands. Massive, hard, mixed quartz-hematite bands, with a pink hue, are also common. Hematite in this association is non-micaceous and has a dull lustre. Hematite and quartz are cemented by a fine-grained magnetite matrix. The nature of the contact between Middle Member ore and the Middle Quartzite varies, but there is often a layer of limonite-rich, oxidized ore known as sub-unit 34 (Figure 7.7) which is sometimes considered waste. Typical examples of Middle Member ore are shown in Figure 7.8.

Figure 7.8: Core Photos from Middle Member Ore



7.2.2.5 Upper Member

This lies immediately beneath the hanging-wall carbonaceous quartzite and is a quartz-goethite-limonite rock with varying amounts of magnetite and specularite. Quartz grains are stained with limonite, giving this member a dirty brown appearance, and bands of colloidal goethite up to 15 cm wide are relatively common. The Upper Member varies from a hard-fine-grained through coarse grained and friable to a soft earthy variety. It is thought that the limonite coatings of quartz grains have been produced by weathering near surface. The member is characterized by low iron and a poor weight recovery, making it the least attractive of the three (3) ore members.

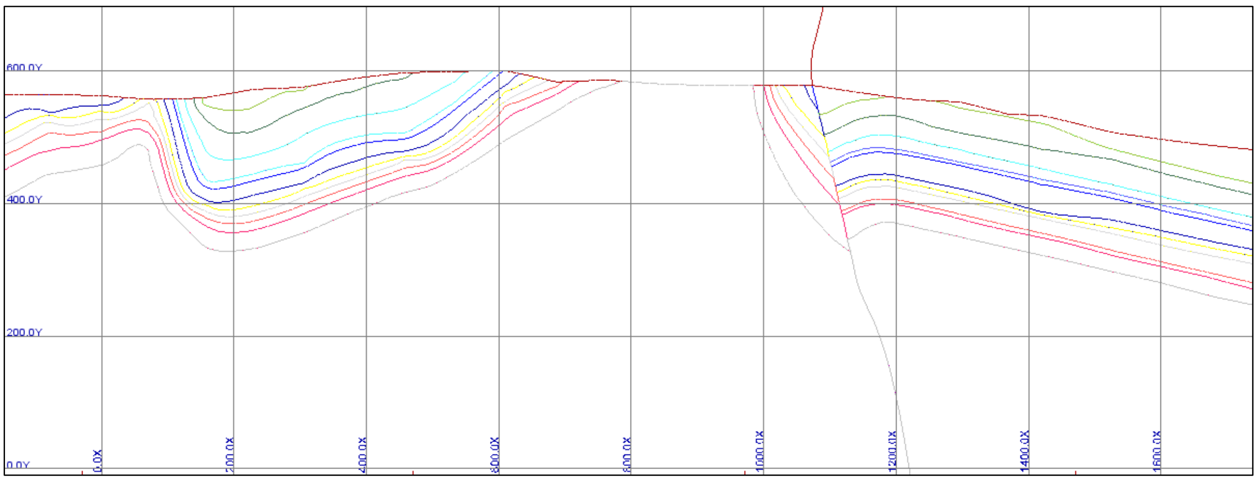
7.2.3 Structural Geology

The Scully Deposit can be divided into two (2) distinct structural areas. Bounded to the east by the Wabush Lake lies a series of northwest-trending folds. This trend continues as far west as the west end of Knoll Lake, where the folds transition to an east-west trend.

The interpretation by O'Leary (1972) explains a series of simple folds in the east plunging gently to the southeast and cut by an almost vertical fault zone, 75 m wide, which is believed to be barren of ore minerals. The area to the west is described as a succession of a synform, an antiform and a second synform to the south. The axes plunge east in the eastern part of the fold system, and west in the western part.

The most prominent structural feature in the East Pit, geologically, and as far as mining considerations are concerned, is a reversed fault which runs approximately in a northwest direction through the orebody (Figure 7.9). This fault dips steeply to the east, with the basement rocks thrown up some 75 m on the western side. The fault is marked by a series of clay seams, varying in color from pink to light cream. In addition, the fault is characterized by an elevated level of manganese, giving assays as high as 6 or 7 percent. The sooty black appearance of the ore against the lighter clay provides a striking contrast and the fault can be traced quite easily along strike as mining advances. A few small parallel fault zones have been traced.

Figure 7.9: Schematic Section Through East Pit Showing Reverse Fault Affecting Stratigraphy



The Scully Deposit is heavily influenced by the presence of folding, which occurs along three (3) principal axes: F1, F2 and F3.

F1 folds with axes striking northwest are asymmetrical to overturned, with alternately steep and gently dipping limbs. The axial planes dip steeply south. In the East Pit, the axial trend is north-northwest; east of the fault, it is almost north-south.

F2 folds occur along axes which run east-west in the West and South Pit areas, and west-southwest in the East Pit. In the West and South Pit, the folds are broad open syncline / anticline / syncline structures. The axial planes dip steeply south and probably reach vertical. Folds plunge to the east in the eastern part of the area, and west in the western part resulting in a domal setting. In the East Pit, the folds of this system plunge gently and slightly modify the plunge of the F1 folds.

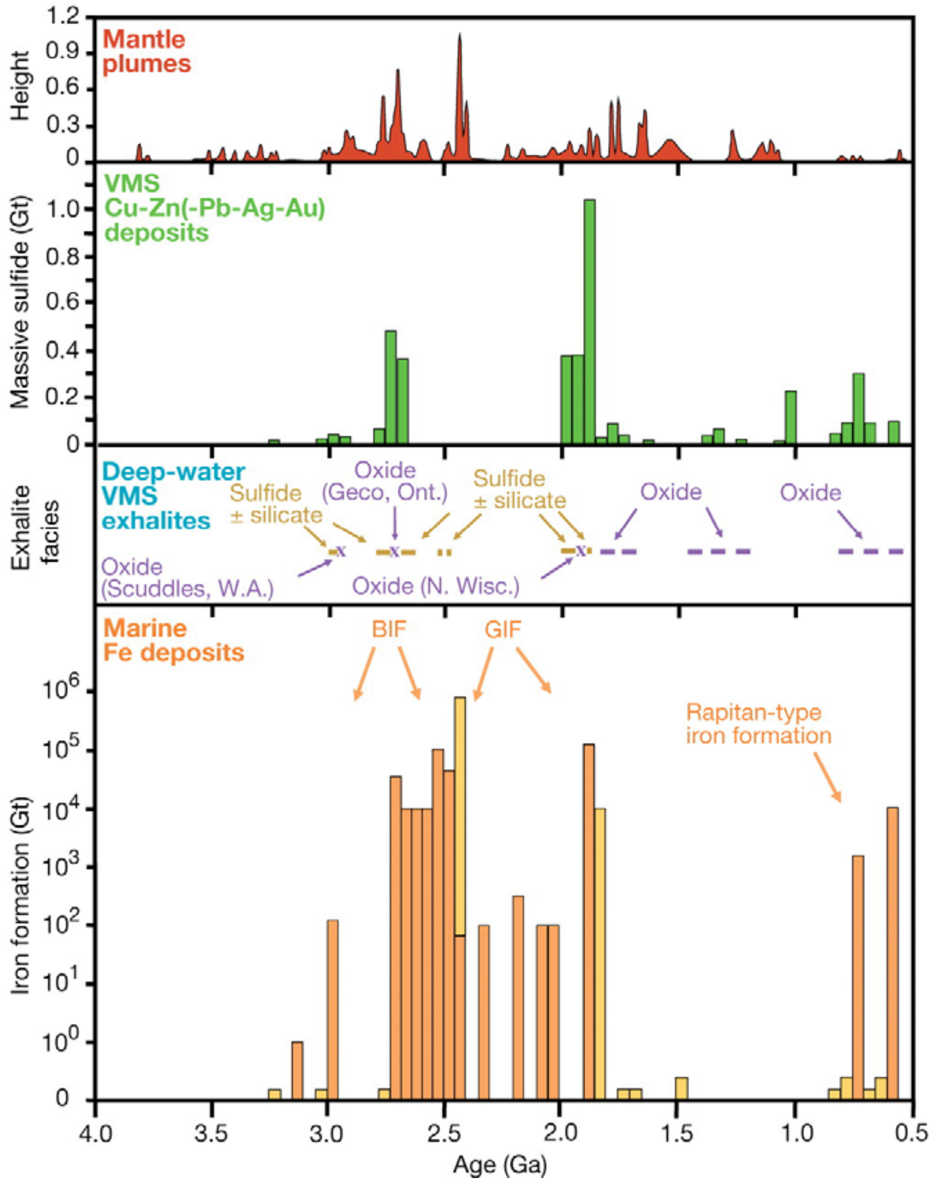
F3 folds striking north-south are only seen in the West Pit area. They alter the axes of the F1 and F2 folds so that plunges change from an easterly to a westerly direction within a hundred feet.

8 DEPOSIT TYPES

The Scully Deposit mineralization style is a deposit typical of the Superior-Lake type of Iron Formations.

The peaks in iron sedimentation took place between ~2.65 and 2.32 Ga and again from ~1.90 to 1.85 Ga. Their deposition is linked to geochemical and environmental evolution of Earth, the Great Oxidation Event ("GOE") at ca. 2.4 Ga, the growth of continents as well as to mantle plume activity and rapid crustal growth (see Figure 8.1).

Figure 8.1: Time Distribution of the Iron Formation Deposition (from Bekker et al., 2010)



The Labrador Trough contains four (4) main types of iron deposits:

- Soft iron ores formed by supergene leaching and enrichment of the weakly metamorphosed cherty iron formation; they are composed mainly of friable fine grained secondary iron oxides (hematite, goethite, limonite).
- Taconites, the fine-grained, weakly metamorphosed iron formations with above average magnetite content and which are also commonly called magnetite iron formation.
- More intensely metamorphosed, coarser-grained iron formations, termed metataconites which contain specular hematite and subordinate amounts of magnetite as the dominant iron minerals.
- Minor occurrences of hard high-grade hematite ore occur southeast of Schefferville.

Secondary enrichment included the addition of secondary iron and manganese which appear to have moved in solution and filled pore spaces with limonite-goethite. Secondary manganese minerals, i.e., pyrolusite and manganite, form veinlets and vuggy pockets. The types of iron ores developed in the deposits are directly related to the original mineral facies. The predominant blue granular ore was formed from the oxide facies of the middle iron formation. The yellowish-brown ore, composed of limonite-goethite, formed from the carbonate-silicate facies, and the red painty hematite ore originated from mixed facies in the argillaceous slaty members.

All iron ore deposits in the Labrador Trough formed as chemical sediments on a continental margin which were lithified and variably affected by alteration and metamorphism that had important effects upon grade, mineralogy and grain size. Faulting and folding led to repetition of sequences in many areas, increases the surface extent and mineable thicknesses of the iron ore deposits. Underlying rocks are mostly quartzite or mica schist. Transition from these rocks and the mineralized iron formation may take place over up to 10 m vertically. All rock sequences have been heavily metamorphosed by intense folding phases that are part of the Grenville Orogen.

Iron formation sequences range commonly from 25% to 40% iron oxide, mainly hematite of the specularite type with minor amount of magnetite (remainder mostly quartz) and can have true thicknesses (ignoring minor intercalated bands of schist and quartz rock) of up to 200 m. It is these sequences which are of economic importance.

For iron formation to be mined economically, the iron content must generally be greater than 30%, but also iron oxides must be amenable to concentration (beneficiation) and the concentrates produced must concentrate specifications (with minimal deleterious elements, such as silica, aluminum, phosphorus, sulphur and alkalis). Iron formations repeated by regional folding produce favorable strip ratios due to repeated synclines, and this structural setting is typical of the Wabush area.

9 EXPLORATION

Only limited information exists regarding non-drilling exploration activities at the Scully Mine. No data relating to geological mapping, surface sampling or other non-drilling prospecting activities was located by G Mining Services ("GMS") in the historical data archives. This is perhaps due to the outcropping nature of the orebody and the early onset of mining which negated the need for early-stage prospecting activities. GMS can also confirm that no additional exploration data has been collected since the mining activities resumed in 2019.

Numerous magnetic and gravity surveys have been conducted at Scully, including a historic magnetic survey from the 1960's and two recent surveys over the Boot Pit area. Details of these surveys are presented below.

9.1 Gravity Surveys

In November 2006, a gravity survey was carried out within the mining lease, west of the Wabush townsite. The purpose of this survey was to detect potential economic iron ore bodies around the pits. The survey was undertaken by Geosig Inc. A total of 549 gravity stations were taken along with coordinates and altitude measurements. The survey focused on the Boot Pit area, south and north of West Pit Ext North, West Pit, South Pit and east of South Pit. The survey revealed several anomalies suggesting the presence of buried masses of iron beneath shallow cover sediments. Bouguer anomalies and the first vertical derivative are shown in Figure 9.1.

9.2 Magnetic Surveys

A historical magnetic survey was undertaken on the Boot Pit in the 1960's (actual year unknown). No further information is available for this survey, and the data is only available as a scanned hard copy.

In October 2010, a detailed ground magnetometer survey was undertaken between mine grid sections 5000E and 7500E (in feet). The purpose of the survey was to provide details for structural interpretation on Boot area. The survey was undertaken with a line spacing of between 100 and 150 ft, using a CDGPS walk-mag setup. A drilling campaign was undertaken on the Boot area in 2011 to follow up on these anomalies and better understand the geology. Figure 9.2 presents the data from this survey.

Figure 9.1: Bouguer (top) and First Vertical Derivative (bottom) Images of the Gravity Survey over Scully Deposit

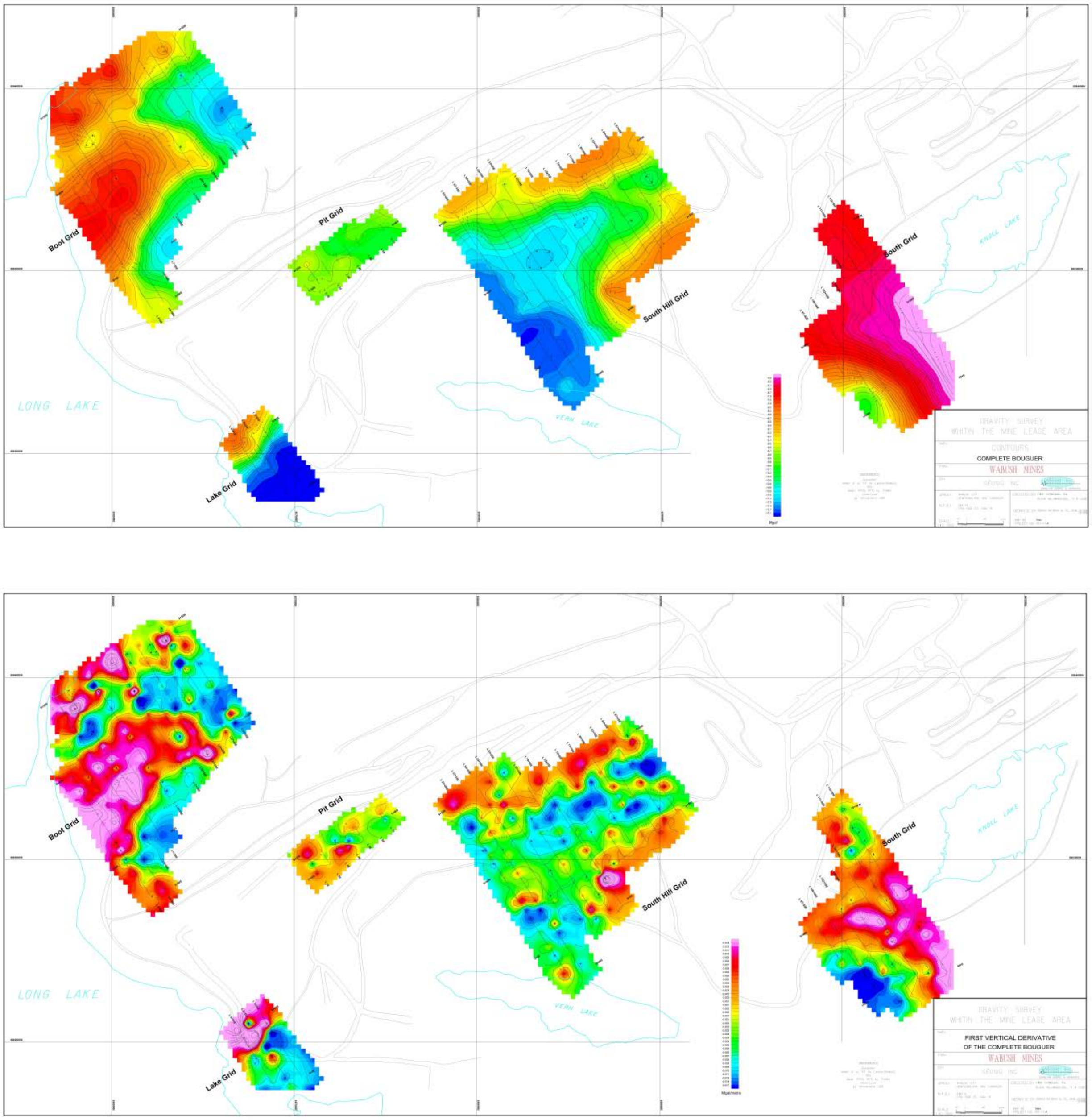
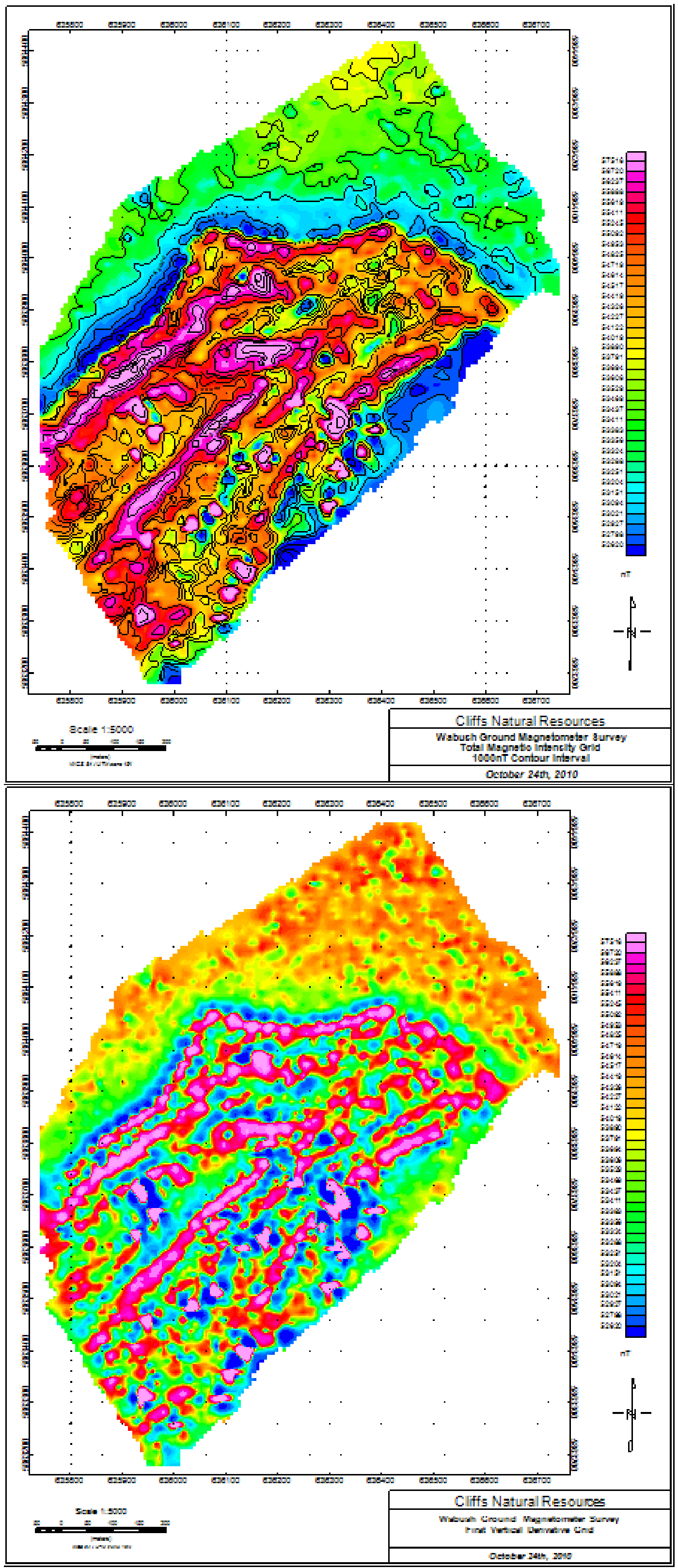


Figure 9.2: Total Magnetic Intensity (top) and First Vertical Derivative (bottom) Images from the 2010 Ground Magnetometer Survey (shown in UTM coordinates)



10 DRILLING

All data relating to drilling undertaken at the Scully Deposit are in the Scully Mine Grid coordinates system (feet). G Mining Services ("GMS") received drilling information in PDF format classified by year and drill hole. Data is reasonably complete from 1969 onwards; however, a large proportion of the drilling in the drilling database pertains to an unknown period, referred to as "historical drilling" in this Section 10. GMS can confirm that since the release of the 2018 Feasibility Study ("FS") and the recommencement of mining operations in 2019, no additional exploration drilling has been conducted on the property. Therefore, for the purpose of this 2021 FS Update, the exploration drilling database remains unchanged.

Available information includes general notes, laboratory results and logs. Drilling information was recorded manually until 1986, including the laboratory results. From 1986 onwards, data was recorded digitally in Microsoft Excel™ format, and is generally more complete. From 2005 onwards, some drilling reports are available which summarize the drilling methods, core photos, and other supporting data.

10.1 Previous Drilling

Information pertaining to the 1969-2013 drilling programs is presented below. A total of 1,005 drill holes were performed on the property for a total of 77,622 m. After the production of the block model, GMS found 6 drill holes from 2013 (WPX13 prefix) which were not included in the drill hole database provided. These drill holes were not included in the resource estimate.

10.2 Pre-1969 Drilling

Out of the 1,005 drill holes present in the database, 507 holes do not include proper identification to retrace the year they were drilled.

GMS believes that these drill holes were drilled in the initial stages of the mine development. From 1969, drill holes began to be prefixed with the year of drilling and continued using that nomenclature until the mid-2000s. These holes appear to be early drilling defining the initial, near surface resources at Scully. According to these drill hole IDs, they were likely drilled during different phases. Initially, in the West Pit South, South Pit and the East Pit, the drill hole sequence shows a systematic drilling spacing of 150 m which covers the pits. An additional infill drill hole sequence shows a drilling spacing of 75 m, which allowed for a better definition of the ore body.

10.3 Drilling Programs 1969-2005

Most of the post-1969 drilling was undertaken to develop the resources of the East Pit, South Pit and the West Pit South, which is shown in Figure 10.1, Figure 10.2 and Figure 10.3. The topography shown was surveyed in September 2017. Numerous drilling contractors were used: Longyear, Dominik Drilling, I.R Drillmaster, Forage Mercier, Lantech, Major Ideal and Logan. Most of the drilling was undertaken using NQ size drill rods.

A summary of drilling by year is presented in Table 10.1, which was constructed using the original drill logs provided in the data pack.

Table 10.1: Drilling Summary

Year	Number of Drill Holes	Hole Prefix(es)	Total Meters (converted from feet)	Type of Drilling	Comments
1969	15	69-	1,687	Diamond Drilling	Canadian Longyear Limited - Mostly East Pit
1970	14	70-	1,015	Diamond Drilling	East Pit and South Pit
1971	24	71-	3,059	Diamond Drilling	I.R. Drillmaster - West Pit South and South Pit Development
1972	8	72-	358	Diamond Drilling	I.R. Drillmaster - Mostly East Pit Along Trend
1973	41	73-	3,392	Diamond Drilling	I.R. Drillmaster, Inspiration - South Pit Development, East Pit and East Pit East
1974	47	74-	1,785	Diamond Drilling	I.R. Drillmaster - Mainly West Pit South, South Pit and East Pit
1976	50	76-	3,918	Diamond Drilling	I.R. Drillmaster, Dominik - West Pit South, South Pit and East Pit Development and Definition
1969	15	69-	1,687	Diamond Drilling	Canadian Longyear Limited - Mostly East Pit
1970	14	70-	1,015	Diamond Drilling	East pit and South Pit
1971	24	71-	3,059	Diamond Drilling	I.R. Drillmaster - West Pit South and South Pit Development
1972	8	72-	358	Diamond Drilling	I.R. Drillmaster - Mostly East Pit Along Trend
1973	41	73-	3,392	Diamond Drilling	I.R. Drillmaster, Inspiration - South Pit Development, East Pit and East Pit East
1974	47	74-	1,785	Diamond Drilling	I.R. Drillmaster - Mainly West Pit South, South Pit and East Pit
1976	50	76-	3,918	Diamond Drilling	I.R. Drillmaster, Dominik - West Pit South, South Pit and East Pit Development and Definition
1972	8	72-	358	Diamond Drilling	I.R. Drillmaster - Mostly East Pit Along Trend
1973	41	73-	3,392	Diamond Drilling	I.R. Drillmaster, Inspiration - South Pit development, East Pit and East Pit East
1974	47	74-	1,785	Diamond Drilling	I.R. Drillmaster - Mainly West Pit South, South Pit and East Pit
1976	50	76-	3,918	Diamond Drilling	I.R. Drillmaster, Dominik - West Pit South, South Pit and East Pit Development and Definition

Year	Number of Drill Holes	Hole Prefix(es)	Total Meters (converted from feet)	Type of Drilling	Comments
1974	47	74-	1,785	Diamond Drilling	I.R. Drillmaster - Mainly West Pit South, South Pit and East Pit
1976	50	76-	3,918	Diamond Drilling	I.R. Drillmaster, Dominik - West Pit South, South Pit and East Pit development and definition
1982	14	82-	894	Diamond Drilling	I.R. Drillmaster - West Pit Ext. North
1983	20	83-	1,184	Diamond Drilling	East Pit Development
1991	41	91-	4,453	Diamond Drilling	Longyear - West Pit Ext. North, Boot Area and East Pit East
1994	24	94-	2,270	Diamond Drilling	Dominik Drilling - South Pit, East Pit and West Pit Ext. North
1996	14	96-	2,427	Diamond Drilling	Dominik Drilling - South Pit
1997	20	97-	3,334	Diamond Drilling	Forage Mercier - South Pit Exploration for South Extension
1998	45	98-	3,211	Diamond Drilling	Lantech - South Exploration of West Pit South
1982	14	82-	894	Diamond Drilling	I.R. Drillmaster - West Pit Ext. North
1983	20	83-	1,184	Diamond Drilling	East Pit Development
1991	41	91-	4,453	Diamond Drilling	Longyear - West Pit Ext. North, Boot Area and East Pit East
1994	24	94-	2,270	Diamond Drilling	Dominik Drilling - South Pit, East Pit and West Pit Ext. North
1996	14	96-	2,427	Diamond Drilling	Dominik Drilling - South Pit
1997	20	97-	3,334	Diamond Drilling	Forage Mercier - South Pit Exploration for South Extension
1998	45	98-	3,211	Diamond Drilling	Lantech - South Exploration of West Pit South
1996	14	96-	2,427	Diamond Drilling	Dominik Drilling - South Pit
1997	20	97-	3,334	Diamond Drilling	Forage Mercier - South Pit Exploration for South Extension
1998	45	98-	3,211	Diamond Drilling	Lantech - South Exploration of West Pit South
1996	14	96-	2,427	Diamond Drilling	Dominik Drilling - South Pit
1997	20	97-	3,334	Diamond Drilling	Forage Mercier - South Pit exploration for South Extension

Year	Number of Drill Holes	Hole Prefix(es)	Total Meters (converted from feet)	Type of Drilling	Comments
1998	45	98-	3,211	Diamond Drilling	Lantech - South Exploration of West Pit South
2001	7	2001-	1,070	Diamond Drilling	Major Ideal - Exploration for extension of West Pit Ext. North
2004	27	2004-	2,798	Diamond Drilling	Logan - Development & definition of East Pit East
2005	33	WP05, WX05, EW05, EE05, SW05, HL05	3,060	Diamond Drilling	Forage Mercier - Boot area, West Pit South, East Pit for Development & Definition
2008	22	WX08, SP08, BT08, EPE08, EPW08	1,510	Diamond Drilling	Lantech - All Pits and Boot Area for Ore Hardness Research
2010	13	BT10, WP10	1,768	Diamond Drilling	Lantech - Boot area and West Pit South for Exploration, Ore Control and Modeling Purposes
2011	23	BT11, WP11	2,208	Diamond Drilling	Lantech - Exploration, Ore control and Modeling Purposes
2012	13	WPX12, SP12, BT12, EPE12	1,758	Diamond Drilling	Orbit Lantech - Boot area, South Pit and East Pit East
2013	6	WPX13	585	Diamond Drilling	Logan
Unknown	490		29,878	Unknown	Unknown
Total	**1,005**		**77,622**		

Figure 10.1: East Pit and East Pit East Historical Drilling

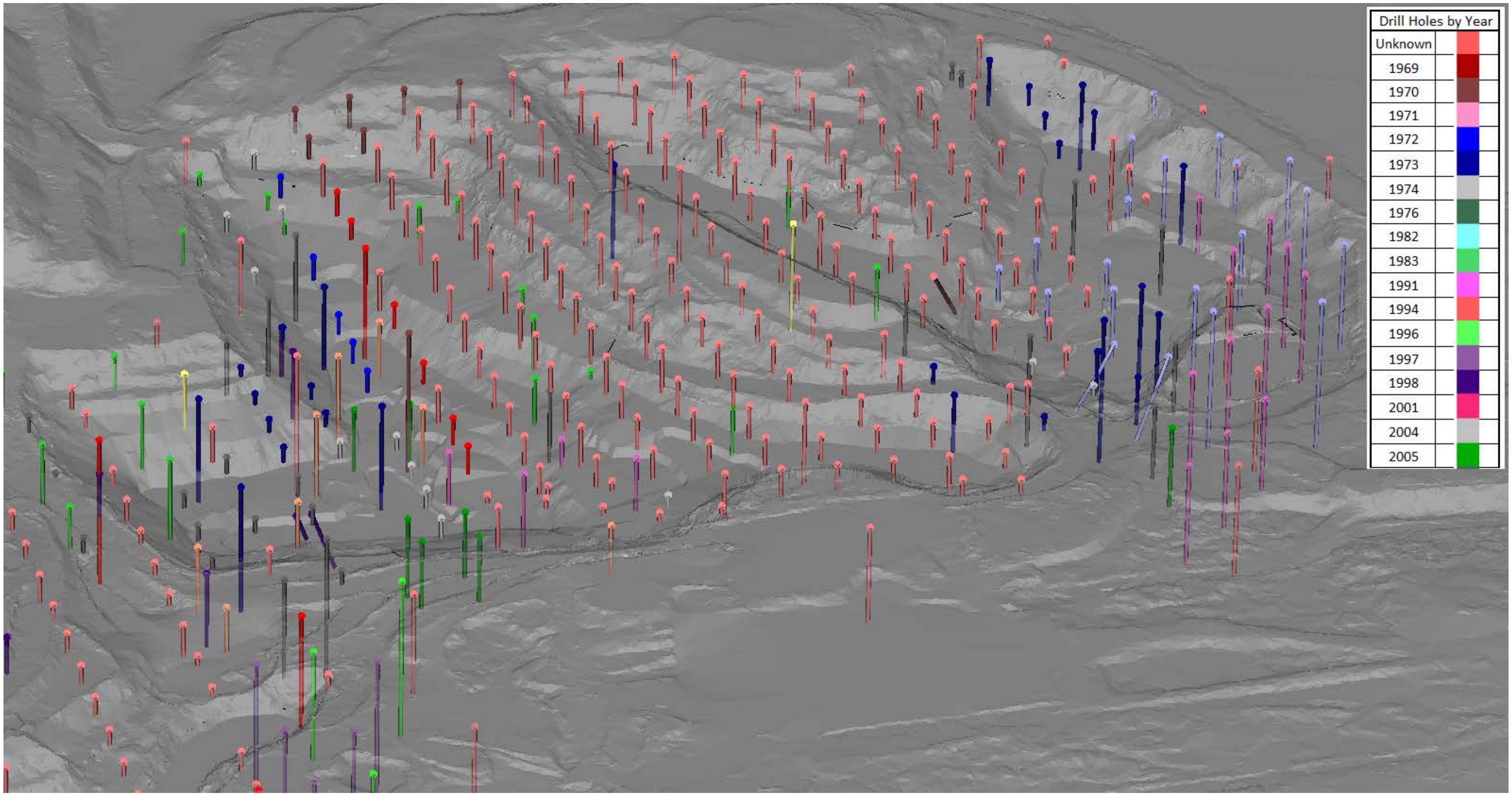


Figure 10.2: South Pit Historical Drilling 1969-2005

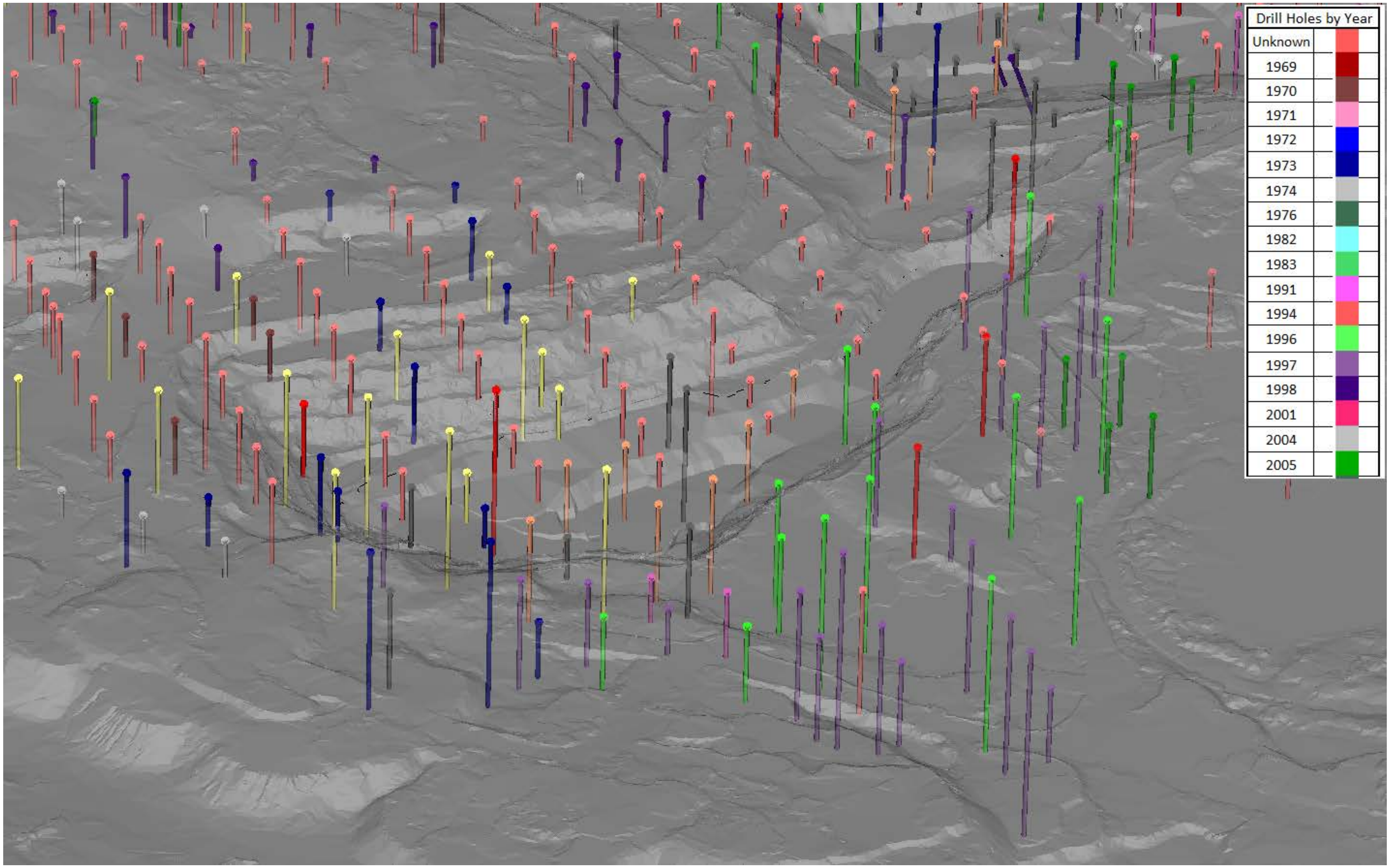
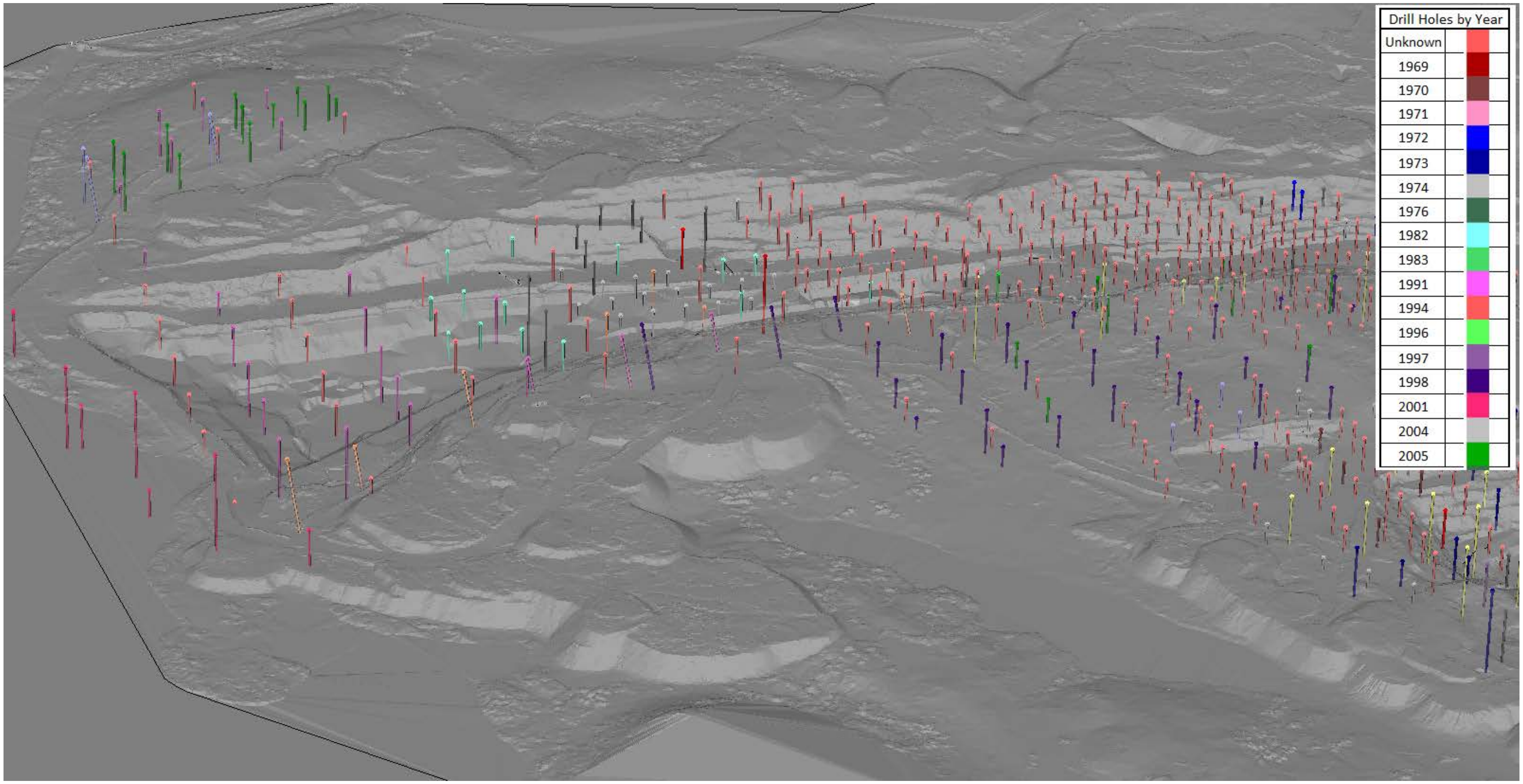


Figure 10.3: West Pit and Boot Area Historical Drilling 1969-2005



10.4 Drilling Programs Around the Boot Area

A total of 48 holes covering 6,089 m were drilled on the Boot area, an unmined syncline of the middle and lower members. Figure 10.4 shows the location of these drill holes. Drilling of the Boot Pit is relatively recent (compared to other areas), with most of the drilling undertaken in the 2000s.

10.5 Drilling Program 2008 – Geotechnical Program

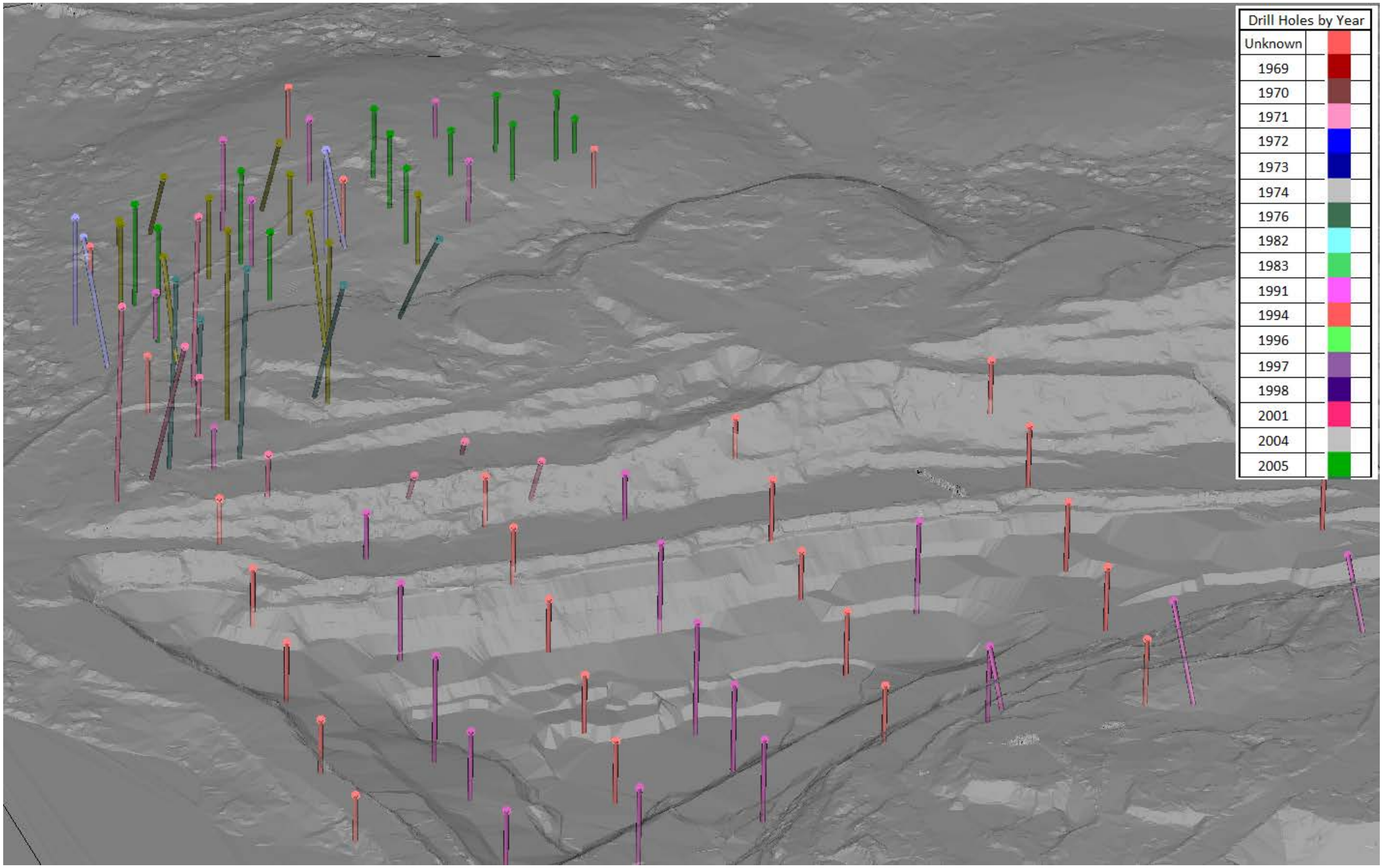
The 2008 diamond drill program goals were to provide samples for an ore hardness research project conducted by Corem Consulting. Due to budget limitations, the Corem research project was stopped in early 2009. After a request to continue the ore hardness project internally, work on the 2008 drill core was completed by CTG geologists in 2009.

CTG carried out the logging and sampling of the 22 diamond drill holes totaling 1,510 m for standard assaying by the Scully lab. New initiatives were developed to replace the Corem ore hardness research and facilitate the design of a new ore competence index at Scully.

10.6 Drilling Program 2010 and 2011

A total of 21 drill holes amounting to 3,306 m were drilled for exploration and / or control and modelling purposes in the West Pit and Boot areas of Scully Mine. The goals were to obtain closer spaced and deeper diamond drill information in the West Pit and to aid in structural modelling in Boot area. The West Pit holes encountered Lower Member ores, while the Boot holes encountered Middle and Lower Member ores, and basement lithologies. Drilling was contracted to Lantech and Orbit Lantech Drilling.

Figure 10.4: Drilling Programs on Boot Area 1991, 2004, 2005, 2010, 2011 and 2012



10.7 Drill Hole Locations

GMS was unable to find references or procedures regarding how drill hole collars were surveyed. Pre-1991 drill collars appear to have been rounded to match the drill lines (250 feet spacing), therefore the precision of these drill collar locations is considered poor. GMS found evidence of significant rounding of XY collar coordinates in the database. A hand-written survey sheet containing high-precision coordinates was found from 2004; however, they were found to be rounded to the nearest drill section in the database.

The more recent drilling in the database shows better precision, with drill collar survey precision to the nearest foot (0.3 m) and often to decimal precision.

Although varying levels of imprecision may be present for the historical drill holes in the database, the newer drill hole collars show sufficient precision for this style of deposit.

10.8 Downhole Surveys

Minimal information regarding down hole surveys is available in the data package provided to GMS. Several survey sheets referring to acid tests were located for 2004 drill holes. In addition, drilling reports (Jongewaar, 2010) mention that a flex-it tool was used to record drill hole dips, however the azimuths were rejected due to magnetite interference. Downhole geophysical surveying was undertaken in 2011 by DGI Geosciences (Jongewaar, 2011) in the surrounds of the Boot Pit, but this focused on televiewer data rather than downhole survey information.

10.9 Core Shack

From the initial days of exploration drilling (late 50s) until the mid-1980ss, the drill core and various samples were stored in the Hay Lake core shack. Having fallen into a state of disrepair, the core generated from the 1990’s drilling campaign was moved the South Pit powder magazine. In 2005, the powder magazine building was deemed unsafe. Upon the completion of the drilling program in late 2008, no practical storage area existed for drill core. The core was subsequently stored inside the classifier building. In the winter of 2009, samples and drill core were disrupted by warehouse personnel. This resulted in the destruction of some of the sample materials before they could be analyzed. As of 2017, the core samples on site seem to be scattered into various areas (pers. comm. Réjean Sirois, VP, Geology and Resources, GMS).

10.10 Core Logging

The core was logged using standard verified methods. Rock types were identified, and intervals were measured. Geological logging considered the general color of the rock, the relative percentage of constituents, the grain size distribution, the alteration, the contact with other rocks, the texture and the variation of these elements, when significant. Geotechnical features in the core, such as core recovery, rock quality designation (“RQD”), fractures and joints, foliation, granulometry, friability, rock strength, and weathering, were also described in the logs. The mineralized units to be sampled were identified depending on the mineral content and considered geological contacts. They were also named using a specific nomenclature relating to the stratigraphic column.

Core logging and Sampling is discussed further in Section 11.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The information contained within this section was sourced from a series of standard operating procedures provided to G Mining Services ("GMS") by representatives of Tacora Resources Inc. ("Tacora"). The procedures outline practices for the logging, sampling and analysis of both exploration and blasthole drill holes, all of which was all conducted exclusively on the Scully Mine Site. Procedures were generally put in place between 2002 and 2007, and there is little information available regarding sampling, preparation and analysis before this time.

As there has been no new exploration drilling conducted on the Property since the 2018 Feasibility Study ("FS") and the recommencement of mining activities in 2019, no new information relating to quality assurance / quality control ("QA / QC") or sampling procedures is available.

11.1 Core Logging

Core logging was determined to be the responsibility of the mine geologist at Scully and is outlined in the procedure *0092002 Diamond Drill-Core Logging (Revised12.01.09).doc*. The procedure was updated in April 2009 to reflect the change from the original data management system "medsystem" to the more centralized database within the Maptek Vulcan software environment. In addition, a procedure exists for logging blastholes (*0092003 Blast Holes Logging (Revised11.25.09).doc*), which explains the techniques for logging drill cuttings.

11.1.1 Exploration Drill Core Logging

Proposed drilling was initially classified into one of five (5) categories: new deposit, delineation, in-fill, structural and grade information. Once the drilling purpose was determined, factors such as recovery, overburden depth, angle of drilling were considered when selecting drill core diameter, and a cost estimate was presented. Three (3) to four (4) contractors were invited to submit bids, with the final selection based on factors such as cost, experience and availability. Drill hole sites were surveyed and cleared, core shack and drilling supplies were made available, and the laboratory was notified.

Upon arrival of contractors, safety procedures were conveyed to drilling contractors, and drilling was conducted using best practices to meet the requirements of Scully Mine. The majority of drill core at Wabush is of NQ diameter (4.8 cm).

Core was logged using a logging sheet in Microsoft Excel (example shown in Table 11.1), giving a full description of core recovery, structure, oxide distribution, grain size, friability, degree of oxidation, clay bands, brecciation, folding and faulting. Drill core was then divided into composites for sampling. Composite lengths were determined by geological contacts, grade variations, oxidation or simply by reducing thick zones into smaller more manageable composites. The procedure emphasizes that composites should be greater than 15 feet (4.6 m) and less than 30 feet (9.1 m). An example of drill core is shown in Figure 11.1.

Composites were bagged and tagged twice (one tag inside the bag, one tag on the outside) showing drill hole ID and the downhole interval. Supporting information was forwarded to the laboratory in the form of a sampling sheet (example shown in Table 11.2). The procedure emphasizes that composites are to be analyzed for five (5) elements: Fe%, Magnetite% (Satmagan), Mn%, SiO_2% and WTREC% (weight recovery).

Core logging and sampling intervals were transferred into the Maptek Vulcan database and the original spreadsheet was saved onto the computer network. A hard copy of each diamond drill log was stored in the geologist's office.

Figure 11.1: Example of Drill Core Prepared for Logging

Table 11.1: Example of Drill Logging Sheet for Exploration Drilling

Wabush Mines
Development Drill Hole Log Sheet

Drill Hole #: BT08-20

Page: 1 of 2

Location	Boot Area	Angle	-90	Date	Aug-09
Easting	5145.0	Depth	173.9	Drilled by	Lantech Drilling
Northing	23728.0	Size	NQ	Logged by	JD Lubben
Elevation	1907.0	Split			

Code: Blank = 0, Tr = 0, Tr-Min = 1, Min = 2, Min-Mod = 3, Mod =4, Mod-Maj = 5, Maj = 6

Geology		Footage			Geotechnical Description						Oxide Distribution					Detailed Core Description
Lith	Rock Code	From	To	Feet	Core Rec.	RQD 2"	RQD 4"	RQD 8"	Dip	Graphic log	Spec	Mag	Mar	Goe	Mn	Remarks
		0.0	10.0	10.0	0.00	0.00	0.00	0.00	-		NA	NA	NA	NA	NA	0.0' - 10.0': Casing depth to 9.8' and overburden recovered to 10.0': waste zone.
		10.0	22.0	12.0	70.83	28.47	11.81	0.00	72		2	3	3	2	0	10.0' - 22.0': red and grey, med-grained chert interbedded with thin (up to .25") to thick (+1") martite/magnetite/+/- spec, irregular and defined bands; zone has low Mn and carries a moderate magnetism throughout; core is hard and competent with occasional broken zones hosting gt/lim staining and slightly increased amounts of spec.; weak to no rxn with H2O2 indicating a low-Mn signature for this zone; bottom of interval is marked by the end of magnetism and the start of coarser-grained chert; @13.0' banding 72 degrees from C.A.; Photo taken at 14.0'.
				0.0												
				0.0												
				0.0												
Middle Member	34			0.0												
				0.0												
		22.0	35.0	13.0	55.77	32.05	21.79	7.69			3	0	3	2	1	22.0' - 64.5': red and brown, coarse-grained meta-chert Fefm with irregular and diffuse Fe-oxide bands; the bands host spec/mart/red hem.; this zone is non-magnetic; minor rxn with H2O2 is noticed indicating a weak Mn concentration; occasional thick (2-3") zone of fractured, white, quartz veins; interval is moderately fractured as a whole (increasing with depth) with broken zones hosting strong lim/gt staining and a "sandy" texture; platy spec and crystalline gt within fracture zones and in vugs within cherty-rich horizons; no bedding/banding angles discernable in this zone; Photo taken at 37.0'.
		35.0	46.0	11.0	77.27	47.73	24.24	9.09			3	0	3	3	1	
		46.0	64.5	18.5	29.28	8.56	0.00	0.00			4	0	3	3	1	
				0.0												
				0.0												
				0.0												
		64.5	83.0	18.5	54.95	9.01	0.00	0.00			2	1	2	4	1	64.5' - 114.0': Resume the red and grey, moderately thick (.25 - +1") banded Fefm with mixed oxides within bands; bands contain low- to med-spec and minor Mn; interval is variably fractured with highly variable alteration expressions throughout; bottom contact marked by the end of thick bands and an increase in coarse spec. and higher Mn. 64.5' - 83.0'; highly fractured and broken, friable core with strong lim/gt staining; no rxn with H2O2; occasional zones with a gt-sand texture; deep-red and yellow in color; no bedding angle are discernable; Photo taken at 75.5'. 83.0' - red, brown, yellow, competent and hard, thin to thick (~.25" - 2") banded, low-spec., variably magnetic, meta-cherty Fefm with low-Mn; interval is variably fractured with broken zones hosting locally strong gt/lim and Mn alteration; these zones can also host coarse quartz veins; banding thicken and is more distinct at the base of the interval; basal contact is also marked by a distinct increase of spec. within thinner bands and the increase in Mn content; @99.0' banding 55 degrees from C.A., @ 114.0' banding 69 decrease from C.A.; Photo taken at 96.0'.
		83.0	100.0	17.0	61.27	35.29	15.69	0.00	55		2	3	4	3	1	
		100.0	114.0	14.0	88.10	58.33	16.07	0.00	69		3	3	4	3	2	
				0.0												
				0.0												
				0.0												
Middle Quartzite	41			0.0												
				0.0												
				0.0												
				0.0												
				0.0												
				0.0												
				0.0												

Table 11.2: Sampling Sheet Written by Geologist, and Filled by Laboratory Staff

Wabush Mines
Development Drill Hole Assay Sheet

Drill Hole #: BT08-20 | Page: 1 of 1

Location	Boot Area	Angle	-90	Date	
Easting	5145.0	Depth	173.9	Drilled by	Lantech Drilling
Northing	23728.0	Size	NQ	Logged by	JD Lubben
Elevation	1907.0	Split			

Code: Blank = 0, Tr = 0, Tr-Mn = 1, Mn = 2, Mn-Mod = 3, Mod = 4, Mod-Maj = 5, Maj = 6

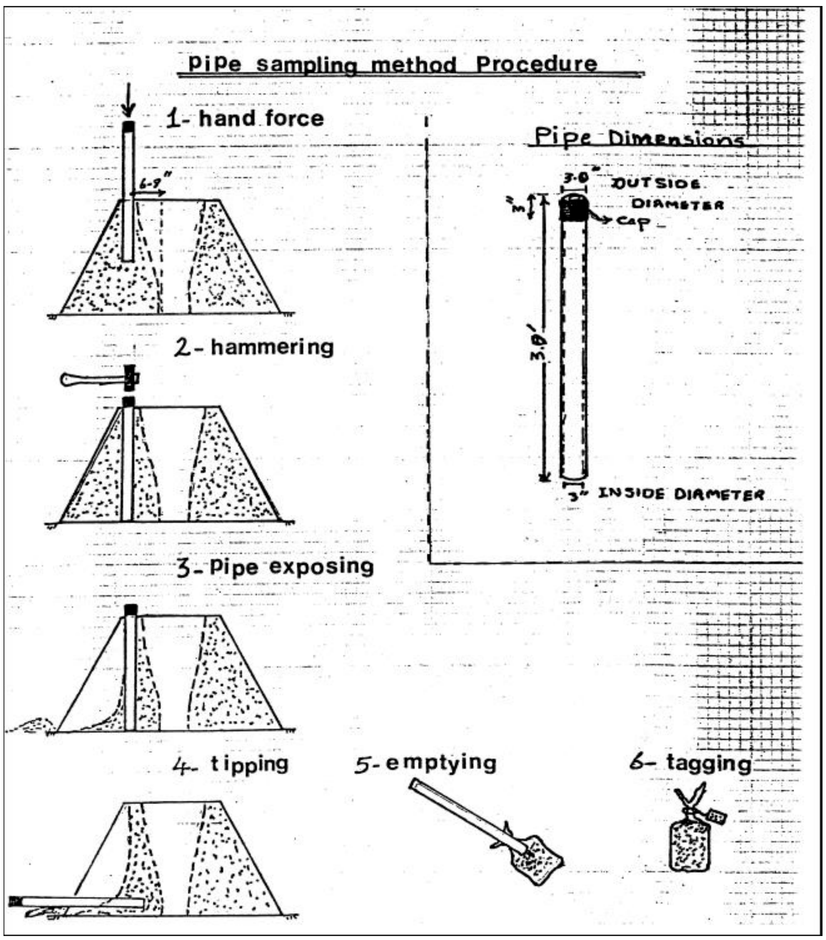
Geology		Footage			Sample	Crude			Concentrate					Tails	WtRec		Detailed Core Description
Lith	Rock Code	From	To	Feet	Sample ID	Fe	Mn	Mag	Fe	Mag	Mn	SiO_2	Wtrec %	Fe	Calc WtRec %	WtRec% Diff.	Remarks
NA	NA	0.0	10.0	10.0	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	NA	
		10.0	22.0	12.0	3225	34.97	0.60	4.20	61.94	7.90	0.89	6.30	26.61	23.71	29.453	-2.84	
		22.0	35.0	13.0	3226	34.15	0.24	1.20	64.81	1.90	0.24	2.43	9.72	27.07	18.76	-9.04	
		35.0	46.0	11.0	3227	35.36	0.11	1.10	64.43	2.20	0.24	2.34	12.83	30.39	14.6	-1.77	
		46.0	64.5	18.5	3228	34.61	0.13	0.90	64.59	1.90	0.18	2.93	9.45	31.17	10.293	-0.84	
		64.5	83.0	18.5	3229	36.41	0.17	0.70	65.81	1.50	0.17	2.10	7.53	32.92	10.611	-3.08	
		83.0	100.0	17.0	3230	29.92	0.21	1.30	62.78	2.40	0.42	5.22	16.28	25.61	11.595	4.68	
		100.0	114.0	14.0	3231	36.58	0.35	1.80	61.71	3.70	0.60	3.83	20.13	28.75	23.756	-3.63	
		114.0	128.5	14.5	3232	33.22	0.14	0.70	62.16	1.00	0.23	6.03	36.73	13.80	40.157	-3.43	
		128.5	144.4	15.9	3233	41.77	0.20	1.80	63.34	2.20	0.22	4.93	42.14	25.41	43.132	-0.99	
		144.4	160.0	15.6	3234	37.74	0.13	2.00	64.74	2.80	0.17	4.31	35.39	22.98	35.345	0.05	
		160.0	173.9	13.9	3235	44.15	0.17	1.00	64.59	0.90	0.07	3.90	50.56	20.02	54.14	-3.58	
				0.0											#DIV/0!	#DIV/0!	
				0.0											#DIV/0!	#DIV/0!	
				0.0											#DIV/0!	#DIV/0!	
				0.0											#DIV/0!	#DIV/0!	
				0.0											#DIV/0!	#DIV/0!	
				0.0											#DIV/0!	#DIV/0!	
				0.0											#DIV/0!	#DIV/0!	
				0.0											#DIV/0!	#DIV/0!	
				0.0											#DIV/0!	#DIV/0!	
				0.0											#DIV/0!	#DIV/0!	
				0.0											#DIV/0!	#DIV/0!	
				0.0											#DIV/0!	#DIV/0!	
				0.0											#DIV/0!	#DIV/0!	

11.1.2 Blasthole Logging and Sampling

The procedure for logging and sampling blastholes follows as illustrated in Figure 11.2.

During the drilling phase, the rotary drill operator / blasters samples the drill cuttings of each blast hole using a three (3) foot long plastic pipe with a three (3) inch inside diameter. The sample pipe is inserted vertically into the drill cuttings at the highest part of the cone (about 6-8” from the drill hole) and forced downward with a hammer until it reaches the bottom of the cone. The pipe is than dug out and tipped horizontally to keep the cuttings from falling out and emptied into a sample bag, which is tagged and numbered with the grid blast hole numbering system; one (1) tag goes inside the bag and another is attached outside. The sample is approximately 5-8 lbs in weight.

Figure 11.2: Schematic of Blasthole Sampling Methodology



According to the procedure, blasthole logging is the responsibility of the Chief Mining Engineer; however, it is unclear who is in charge of the actual logging (likely the drilling operators).

From the vertical channel cut from the blast hole cone, information was gathered and recorded in a field notebook by drill number and material logged relating to the position of the vertical channel (i.e., top, middle, bottom) or the percentage of the vertical channel. A rough visual estimate was made relating to crude Fe %, Mn % and other properties, and was recorded as the following values: Low, Average or High. Samples were gathered as previously discussed and sent to the laboratory. Logging information was plotted up and checked off against the survey record sheet to ensure all blastholes had been recorded (Figure 11.3).

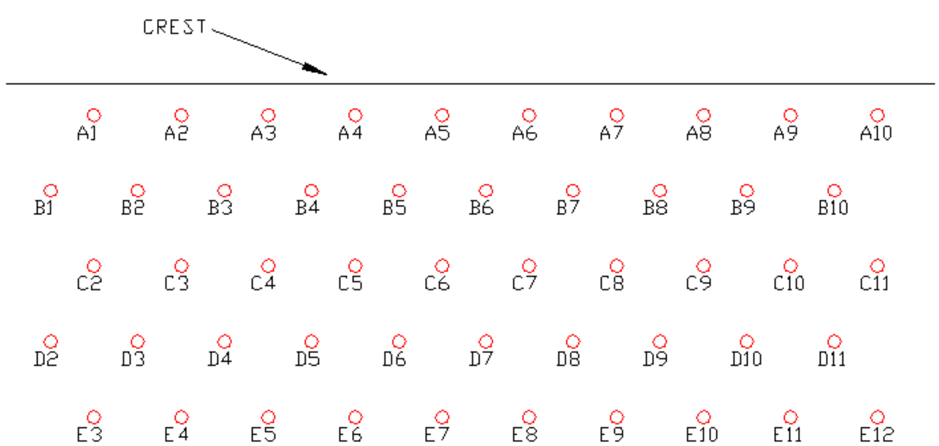
Laboratory analyses were compared with the visual estimates (when analyses become available). Any large discrepancies relating to weight recovery between the visual estimates and laboratory results were followed up and samples were re-analyzed as required.

Shovel maps were produced, and data was uploaded to the computer network. Ore, overburden, and member contacts were marked on the shovel maps, in addition to faults and Mn-rich and magnetite-rich zones.

Figure 11.3: Example of Blasthole Sheets Used for Logging

WABUSH MINES Cliffs Mining Company, Managing Agent	PROCEDURE	Page No.:4 of 6 Date: 11/25/2009
Title: BLAST HOLES LOGGING Section: **ISO – Process Control**		Document No. **0092003**

DRILL MAP



11.2 Core Sampling, Security and Chain-of-Custody

All drilling, core sampling, sample preparation and analyses were undertaken at the Scully Mine site since the mine started operating. Drill core logging, sample selection and compositing were undertaken under the supervision of the Mine Geologist, with the laboratory technicians responsible for the sample preparation and subsequent analyses. Blasthole sample collection and logging were the responsibility of the chief mining engineer and the drilling operators. All sample materials were kept on the Scully Mine site during the sampling and analysis process. In 2008, an ore characterization research project was initiated, and samples were prepared from three (3) drill holes to be sent to COREM Consulting in Quebec City. However, due to budget constraints, this project was curtailed before any samples were sent off-site.

GMS could not find any information relating to drill core or samples that were sent off-site to umpire laboratories, therefore, the chain-of-custody remained confined to the Scully Mine site at all times.

11.3 Sampling Summary

The procedures outlining sample collection and preparation are considered sufficient for the style and scale of deposit mined at Scully. GMS believes that the data collected since the procedures were employed (2002) is of sufficient quality and quantity to have confidence in its application during resource estimation.

GMS notes that no sampling procedures are available pre-2002, therefore, GMS has lower confidence in the data. As sampling and logging data was not available in a centralized database (detailed logging, core recoveries, etc.), GMS could not further assess the impact in a spatial context, and thus was not further considered.

11.4 Analytical Laboratories

Since operations began at Scully, all sample analysis has been undertaken using the laboratory located on site. The laboratory consists of shaker tables (Wilfley tables), XRF analyzers (Philips) and Satmagan analyzers, operated by various employees of Cliffs Natural Resources.

GMS is not aware of any other laboratories that analyzed samples from Scully and has not found any evidence of umpire assaying or check assays. In addition, GMS did not find evidence that the Scully laboratory was certified or audited externally.

11.5 Sample Preparation and Analytical Procedures

Core sample preparation is described for exploration samples in procedure *0103039 Exploration Sample Preparation.doc*, which states that laboratory attendants are responsible for the collection preparation and analysis of samples, in accordance with the sampling intervals provided by the mine geologist. In addition, procedure *0103029 Blasthole sampling.doc* is available, outlining the steps followed during blast hole sampling. The procedure states that drilling operators or blasters are deemed responsible for the collection of samples from blast holes and delivering them to the laboratory. The section manager – mine operations provide oversight of this process, ensuring that the procedure is followed correctly.

Samples were transferred from the bags (or buckets) into pans, along with the tags which were found on the outside and inside of the sample bags. Samples were dried in the oven overnight. After drying, each sample was crushed in a jaw crusher and then a roll crusher and screened through a 20-mesh screen. Any material + 20-mesh was fed through the roll crusher again until 100% of material passes through. The material was rolled thoroughly on the rolling mat to homogenize the sample before splitting. Splitting was undertaken using a riffle splitter to produce a 50 g head sample, a 2000 g shaker table sample and finally 3000 g is retained for storage. If the sample was greater than two (2) bags in size, then each bag was treated separately and only ½ of the riffled material was retained from each bag, re-homogenized on the rolling table, and a final re-split was undertaken to obtain the desired portions for analysis.

11.6 Analysis

As mentioned in Section 11.5, the laboratory produced two (2) sample splits (50 g and 2000 g) which followed two (2) different sample streams resulting in three (3) final sets of analyses: Crude (head) grades, Concentrate grades and Tails grades. XRF sample preparation, fused bead creation and subsequent analysis was described within the file *John Libal – Xrf procedure.docx*, which outlines the entire process.

11.6.1 <u>Crude Grades</u>

The 50 g sample (already crushed to -20 mesh during sample preparation) was pulverized and a subsample of 25 g was taken, processed in a scatter box and passed through 200 mesh. Material not passing 200 mesh was re-sent to a scatter box and processed again through 200 mesh until all the sample passes. This 25 g subsample is then transferred into a manila packet and labelled accordingly with a shift and sample matrix type.

Reagents were prepared (10% Citric acid solution and Lithium nitrate solution), and fused beads were created using a 10:1 ratio of flux (50% Lithium Tetra Borate, 50% Lithium Meta Borate) to pulverized sample. A 0.8 g of pulverized sample was mixed with 8 g of flux. The mix was then placed in a platinum crucible with 1 ml of reagent (Lithium nitrate) and fused using a Phoenix machine. The resulting bead was then removed, and the sample ID was written on top. Apparatus was subsequently cleaned using citric acid and sonic baths.

Sample beads are sent to the XRF operator. The XRF machine was calibrated daily using certified laboratory standards, where 20 standards were analyzed to ensure accuracy is maintained. If standards did not meet the calibration verification criteria, the laboratory supervisor was notified. A Philips PW2400 was used to analyze sample beads.

11.6.2 Weight Recovery and Concentrate Grades

The following information is found within a procedure called *Preparation of the Mine Sample – Document No 0103036 (shaker table test.pdf)*, which contains the operational settings of the Wilfley tables used to produce concentrate and tailings.

Concentrate analysis was undertaken using a Wilfley table (shaker table). The 2000 g samples are presented to the Wilfley table as a size of -20 mesh (<0.8 mm) at a uniform rate through the vibrating feeder. Water is fed at a regulated pressure of 62 gauge. The slope of the table was usually run at the steepest slope and remained constant throughout the test. Concentrate and tailings were collected in separate collection pans and were carefully decanted at the end of the process to avoid any loss of material. The two (2) resulting samples are filtered, dried in the oven, and weighed. Weights are recorded, and the percent weight recovery is calculated based on the two (2) sample weights as shown by the equation below:

$$Weight\ Recovery = \frac{Concentrate\ Weight}{(Concentrate\ Weight + Tailings\ Weight)} \times 100$$

The concentrate and tailings are then rifled individually down to 50 g and sent to the XRF sample stream for analysis (as described in Section 11.6).

11.7 Specific Gravity Measurements

No specific gravity measurements from the Scully Mine were available. Specific gravity was not routinely measured as part of exploration sampling, assumedly due to an emphasis on concentrate weight recoveries and grades (the final product).

11.8 Quality Assurance and Quality Control Procedures

There is little mention of regular QA / QC procedures within the data package supplied to GMS. Apart from the certified standards used during the calibration of the XRF equipment, no evidence of blanks, certified standards or field duplicates into the sample stream have been observed. Since operations commenced in 1964, it appears the Scully laboratory did not adopt typical laboratory standardization, and no consistent QA / QC protocols were implemented during the life of the mine. Typical QA / QC protocols would include the regular insertion of blank material, externally certified reference material ("CRMs") and field duplicates collected from drill core; and it is essential to maintain the integrity of the assaying process. Without a strong QA / QC protocol, inconsistencies or bad-practices during the sampling process cannot be detected and will remain inherent in the data.

GMS embarked on an independent re-assaying program to confirm the validity of the assay database, which is discussed in detail in Section 12.

Although little QA / QC data exists, GMS is encouraged that comprehensive sampling and analysis procedures were put in place from 2002 onwards, and that the mine has been operational for many decades without significant issues with grade reconciliation.

11.9 Conclusions

GMS considers the previously described sampling and analysis procedures as acceptable for use in resource estimation considering the style of mineralization and the size of the Scully deposit. However, there are numerous outstanding issues which need to be addressed to improve the performance of the laboratory and reduce the potential for significant errors. These are discussed in Section 26 as recommendations.

In summary, the laboratory at the Scully Mine requires significant improvements to ensure that future production is guided by robust data, and contaminant levels (Mn%, magnetite%) can be accurately analyzed for mine planning purposes.

12 DATA VERIFICATION

12.1 Introduction

Due to the lack of sampling and assaying quality assurance / quality control ("QA/QC") protocol in the past at the Scully Mine, G Mining Services ("GMS") undertook a comprehensive check-assay and data verification program to validate the diamond drilling database to determine if it is suitable for resource estimation purposes. The information presented below represents a summary of the re-sampling program.

Since the release of the 2018 Feasibility Study ("FS") and the recommencement of mining operations in 2019, no further data verification work has been completed.

12.2 Outline of Re-Sampling Program and Sample Selection

A list of samples from the available core boxes and coarse rejects was prepared by GMS. The selection of samples was undertaken randomly within the foreseen future mining pit design. A total of 3,369 samples were found to be located within the future mining areas.

GMS believes that a selection of between 165 and 200 samples (representing between 5 and 6% of all data) will be sufficient to perform the comparison between historical assay grades within the current drilling database, and the re-assays derived from core boxes and coarse reject bags. Blanks, Standards and Duplicates were inserted within the samples flow to adhere to a valid QA/QC protocol (around 40 QA/QC samples were inserted).

The re-sampling process took place at site between August 17th and August 29th, 2017. GMS Qualified Person ("QP") was present at site between August 17th and 18th to supervise the process. A written procedure was prepared by GMS to ensure that the process was well documented and followed the Canadian National Instrument 43-101 (NI 43-101) guidelines. A total of 167 samples were collected (105 core samples and 62 coarse rejects), and 44 QA/QC samples (12 duplicates, 17 CRMs and 15 blanks) were inserted in the sample flow. A total of 10 core samples were collected to assess the bulk density of the various rock-types.

Mr. Anthony Cranford from Tacora Resources Inc. ("Tacora") was appointed as responsible for the 2017 re-sampling program and was assisted by two co-workers (Figure 12.1 and Figure 12.2).

Figure 12.1: Re-Sampling Program 2017 (Scully Mine) - Collecting Drill Core for Sampling

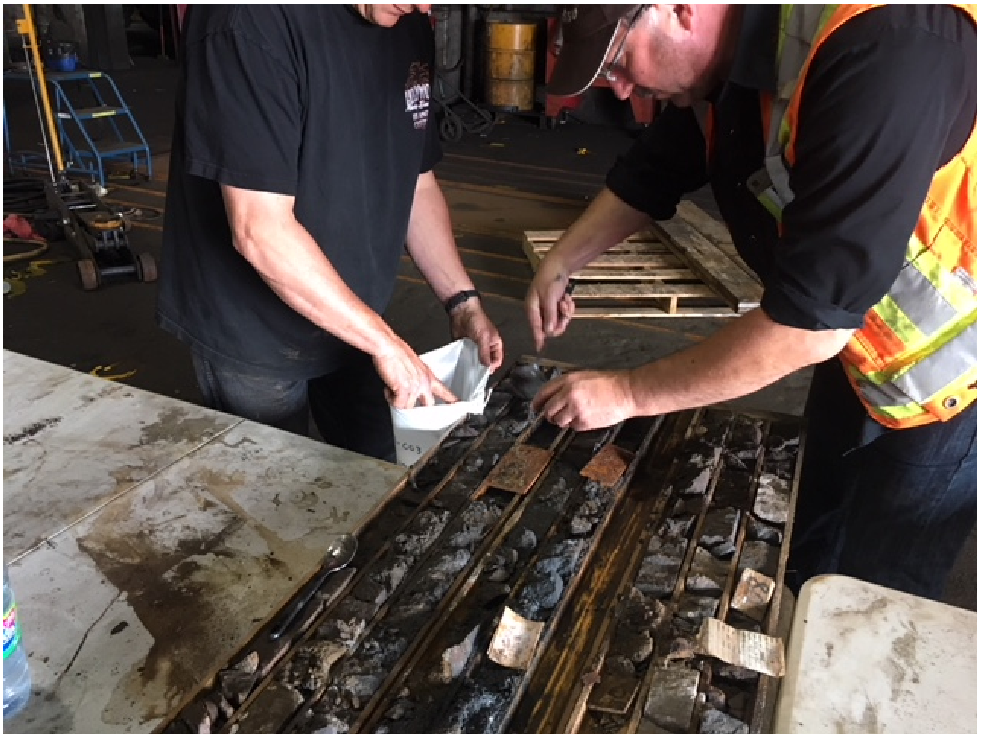
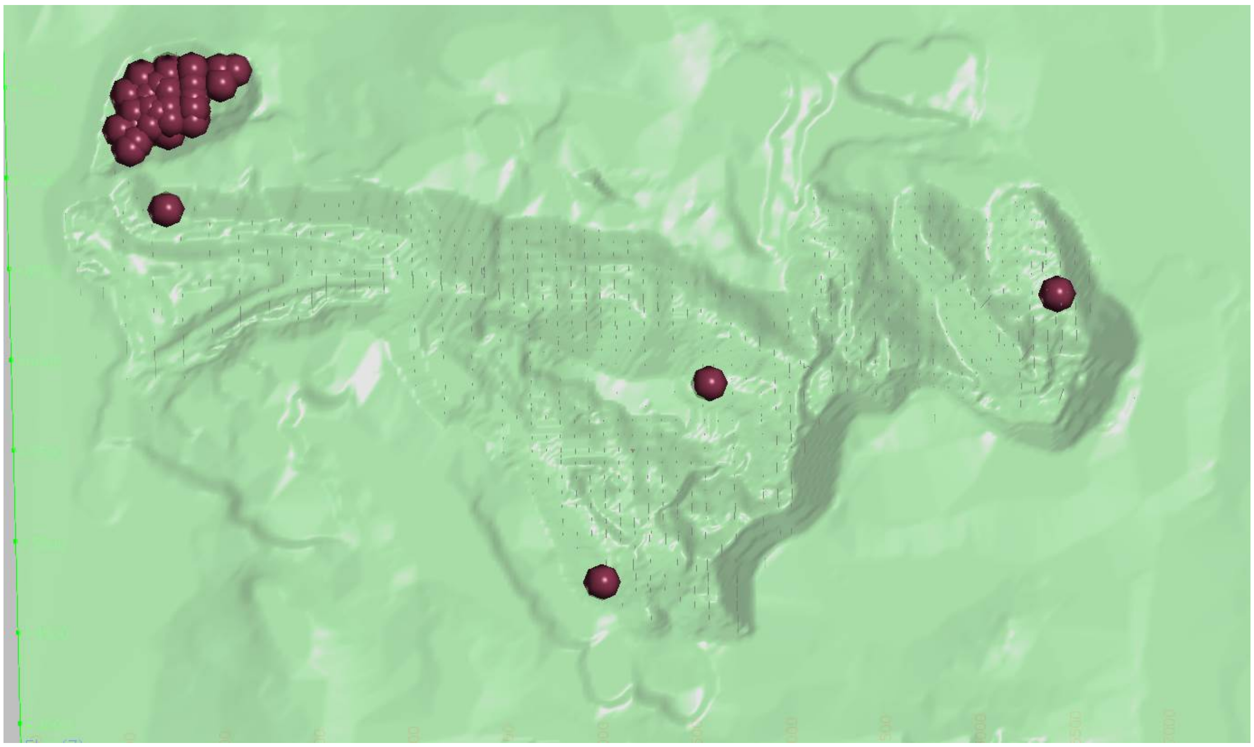


Figure 12.2: Re-Sampling Program 2017 (Scully Mine) – Collecting Coarse Rejects for Sampling

Tacora selected SGS Canada, Lakefield, Ontario ("SGS") as the independent assay laboratory and Mr. Tom Watt from SGS as Project Coordinator. The final assay results were received on October 11 and 12, 2017. GMS received the assay results in PDF and CSV formats.

GMS reviewed the assay results and prepared the following comparisons between the assay results contained within the Scully database and the SGS reanalyses results. The location of the samples is presented in Figure 12.3. Most of the selected samples are located inside the Boot Pit, where more recent drilling was conducted; drill core is in relatively good condition.

Figure 12.3: Re-Sampling Program 2017 (Scully Mine) – Samples Location



12.3 Comparison Between Scully Database and SGS Assay Results – Crude Fe%

The comparison between the assay results for Crude Fe% is presented in Figure 12.4 and Figure 12.5. GMS observes that although the correlation is good (R^2 of 0.41), the general trend shows that on average the SGS Fe % results are around 7% lower (2% lower in grade value) than the Scully database. GMS does not believe that this difference is critical since the Scully Mine is a past producer. However, the reader should be aware that the Fe% grade of the Scully drilling database is possibly slightly over-estimated compared to the SGS reanalyses. SGS is a certified laboratory, and to the best of GMS's knowledge, the Scully laboratory was not certified.

Figure 12.4: Correlation for Fe% Between the Scully Database (X-Axis) and the SGS Reanalyses (Y-Axis) – All Data

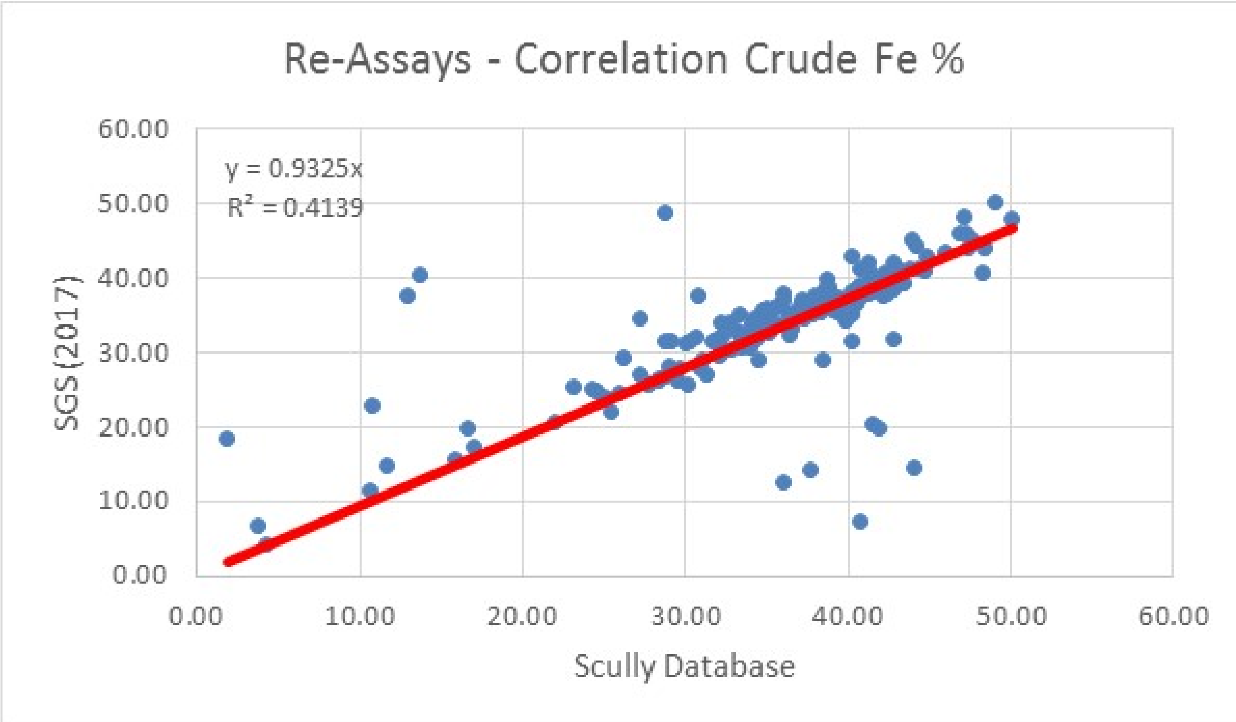
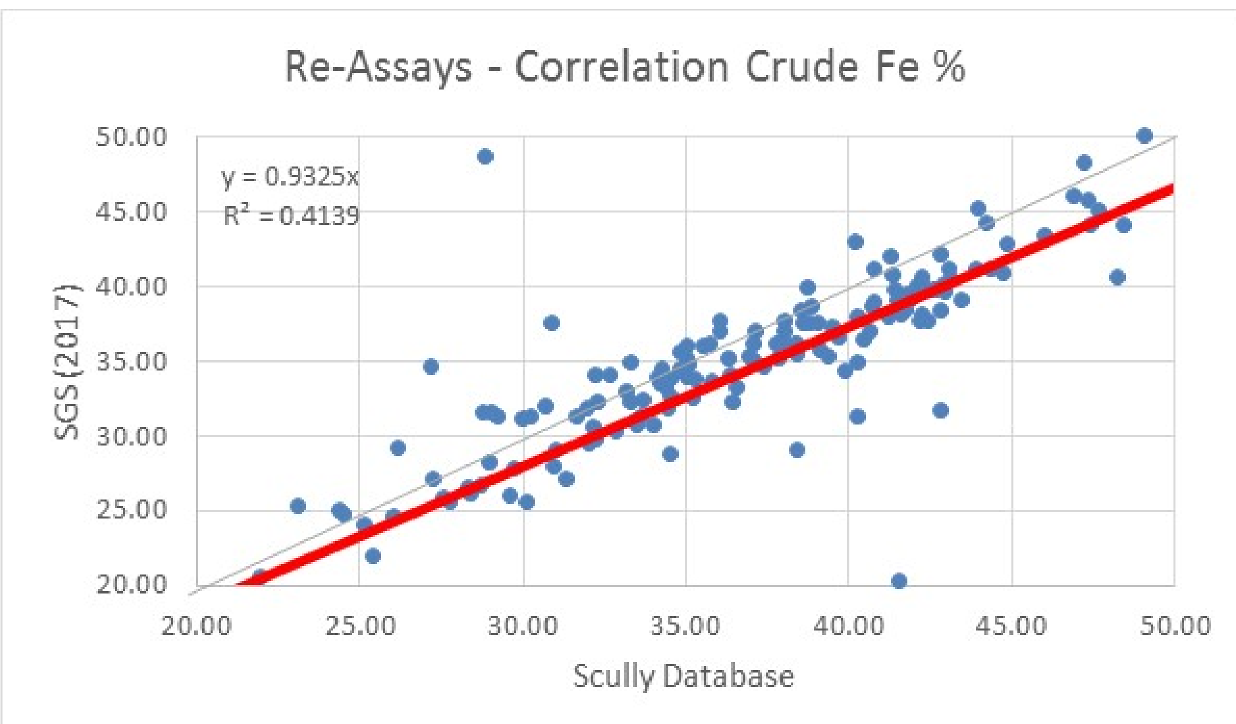


Figure 12.5: Correlation for Fe% Between the Scully Database (X-Axis) and the SGS Reanalyses (Y-Axis) – Close-Up of Data Between 20% and 50% Fe



12.4 Comparison Between Scully Database and SGS Assay Results – Crude Mn %

The comparison between the assay results for Crude Mn% is presented in Figure 12.6 and Figure 12.7. The correlation between the assay results is good at an R^2 of 0.69. At higher Mn grades, the SGS results are generally lower than the Scully database, however this could be strongly influenced by outliers.

Figure 12.6: Correlation for Mn% Between the Scully Database (X-Axis) and the SGS Reanalyses (Y-Axis) – All Data

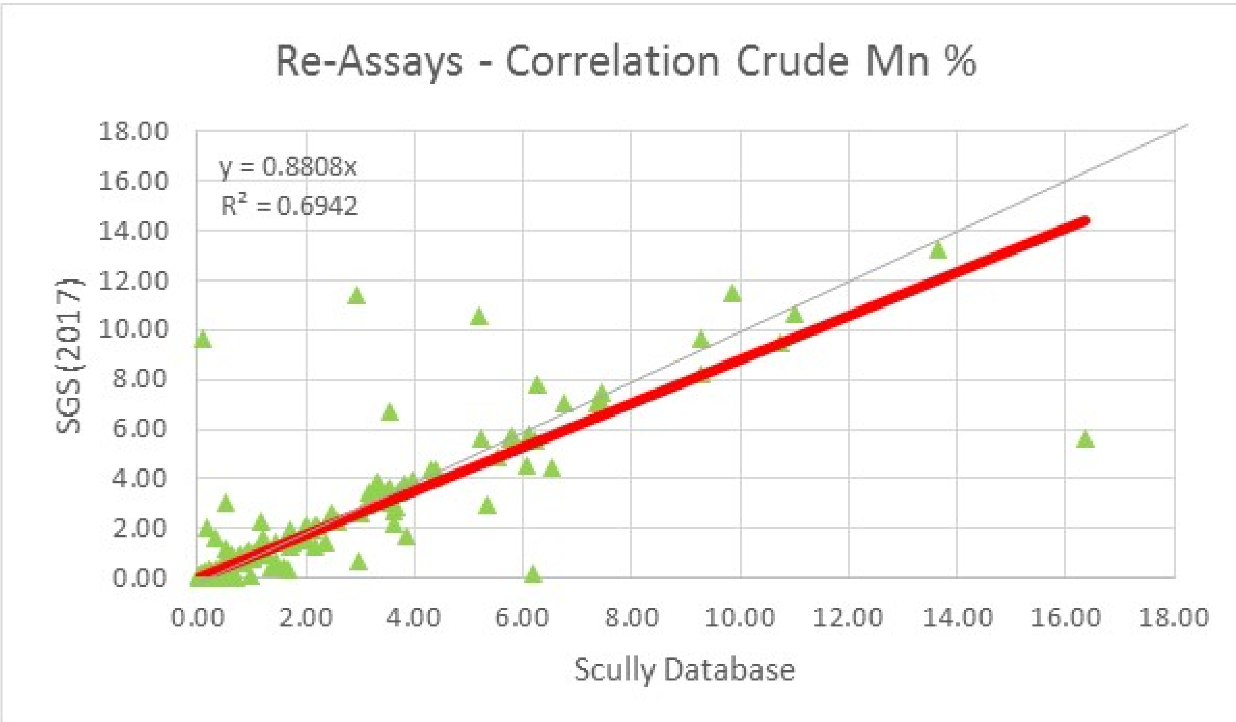
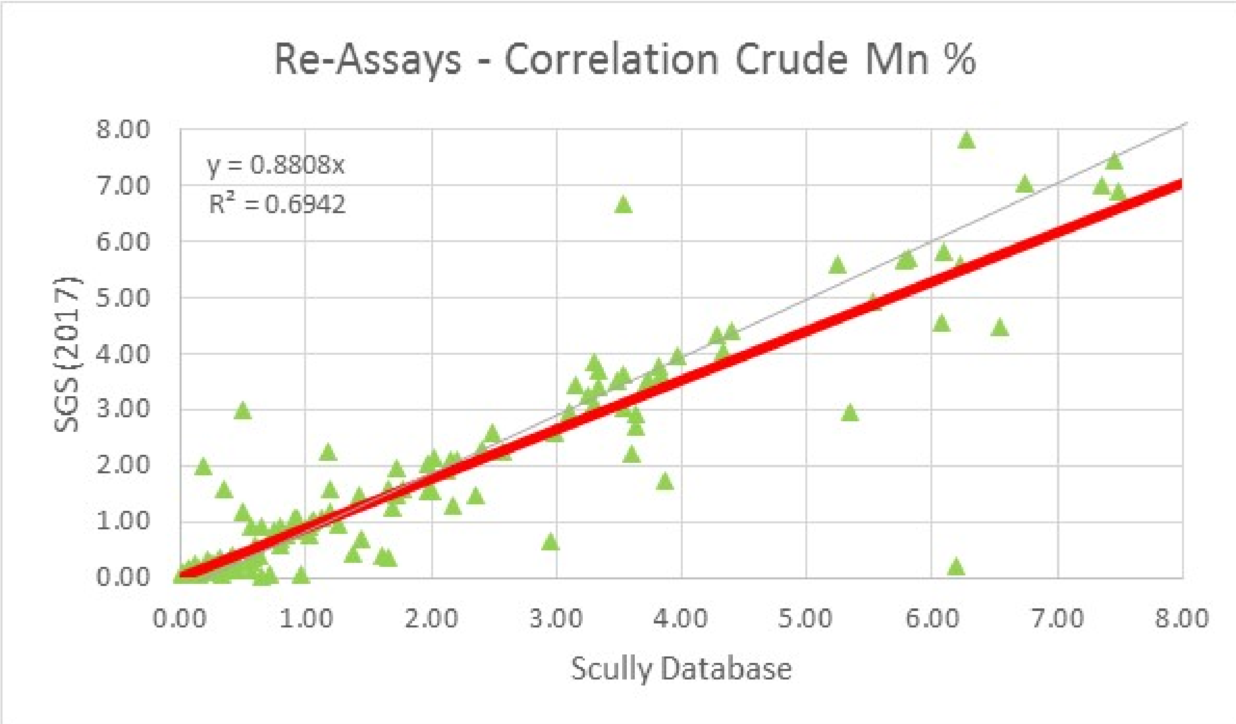


Figure 12.7: Correlation for Mn% Between the Scully Database (X-Axis) and the SGS Reanalyses (Y-Axis) – Close-up of Data Between 0% And 8% Mn



12.5 Comparison Between Scully Database and SGS Assay Results – Crude Magnetite %

The comparison between the assay results for Crude Magnetite % is presented in Figure 12.8 and Figure 12.9. The correlation between the assay results is good at an R^2 of 0.65. At higher Magnetite % grades in the SGS results, we observe lower Magnetite % assays in the Scully database.

Figure 12.8: Correlation for Magnetite % between the Scully Database (X-Axis) and the SGS Reanalyses (Y-Axis) – All Data

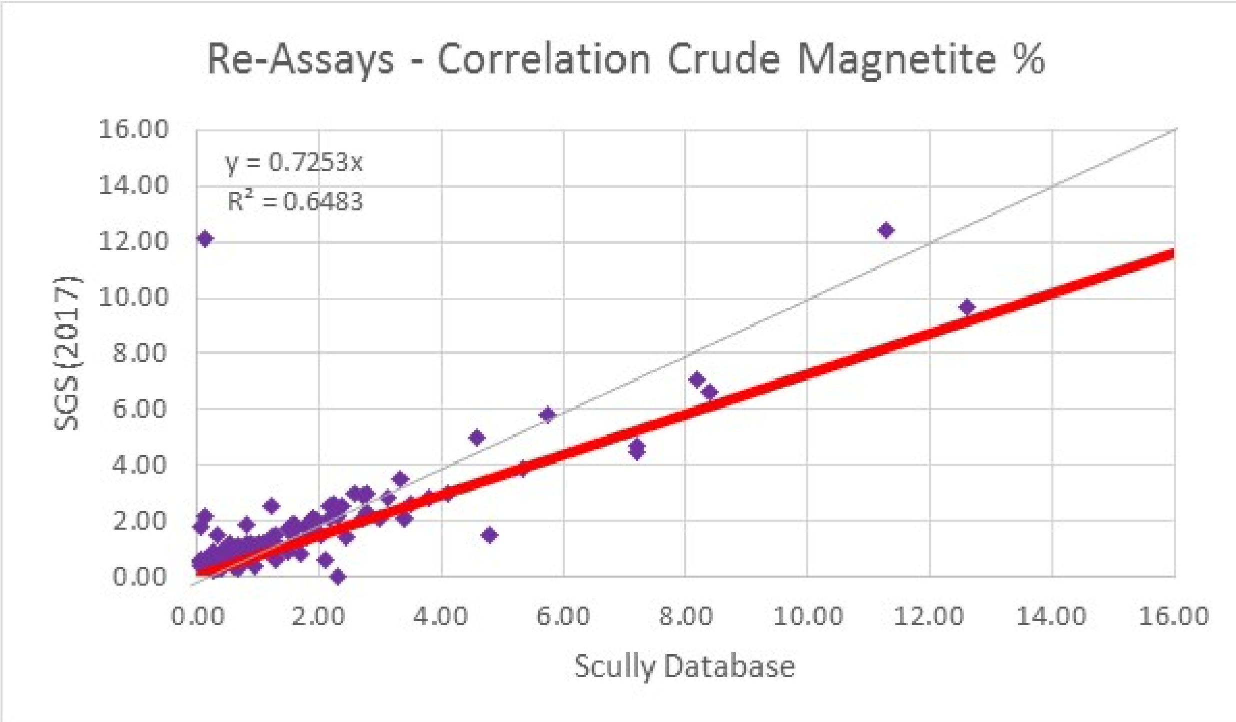
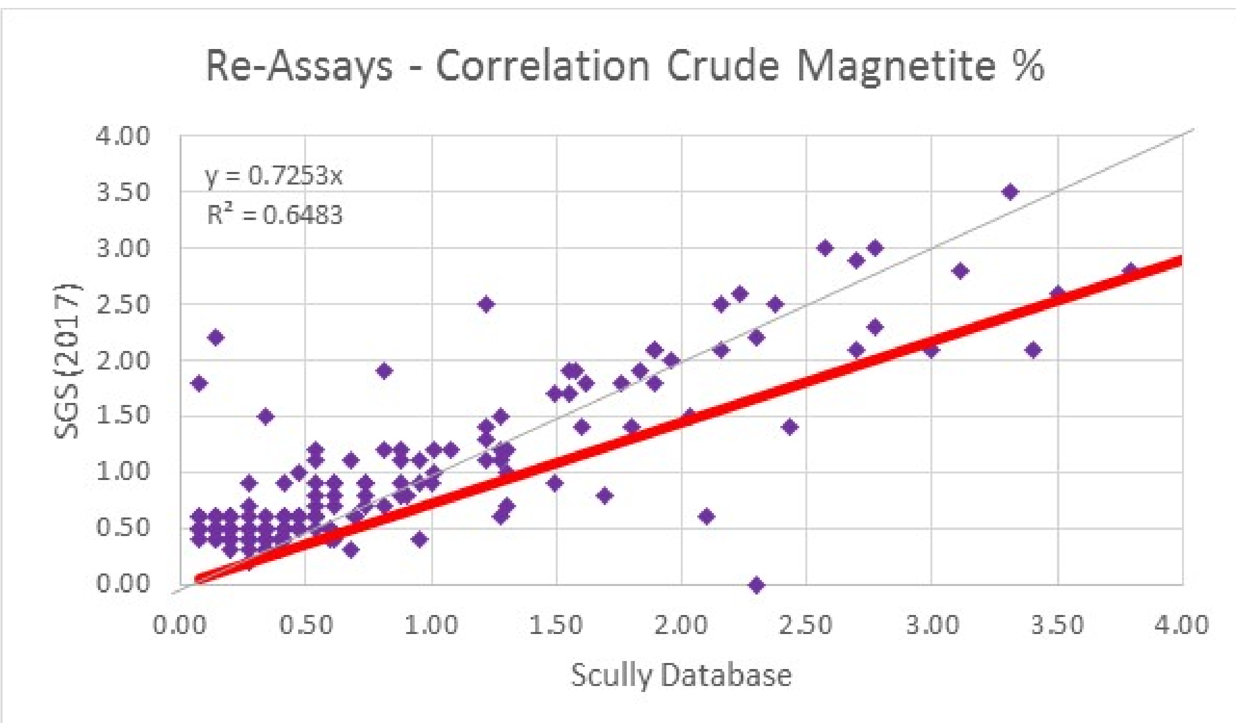


Figure 12.9: Correlation for Magnetite % between the Scully Database (X-Axis) and the SGS Reanalyses (Y-Axis) – Close-Up of Data Between 0% and 4% Magnetite



12.6 Quality Assurance and Quality Control

In addition to check assays comparing the Scully laboratory to the certified SGS laboratory, GMS inserted regular blanks, CRMs and duplicates to assess the performance of the SGS laboratory during the re-assay campaign. Only results for crude Fe% are shown in this Technical Report.

The results showed no areas of concern, as they all fall within acceptable limits for QA / QC purposes. The results of Certified Reference Material ("CRM") 463 are presented in Figure 12.10 for Crude Fe%. All results are within +/- 3 SD, and no failures were observed. Figure 12.11 shows no issues with the blanks that were inserted within the flow of samples, as all of them are at the detection limit of the analyzing instrument. The duplicates all performed well, and an example is shown in Figure 12.12.

Figure 12.10: Assay Results for CRM 463 – Crude Fe%

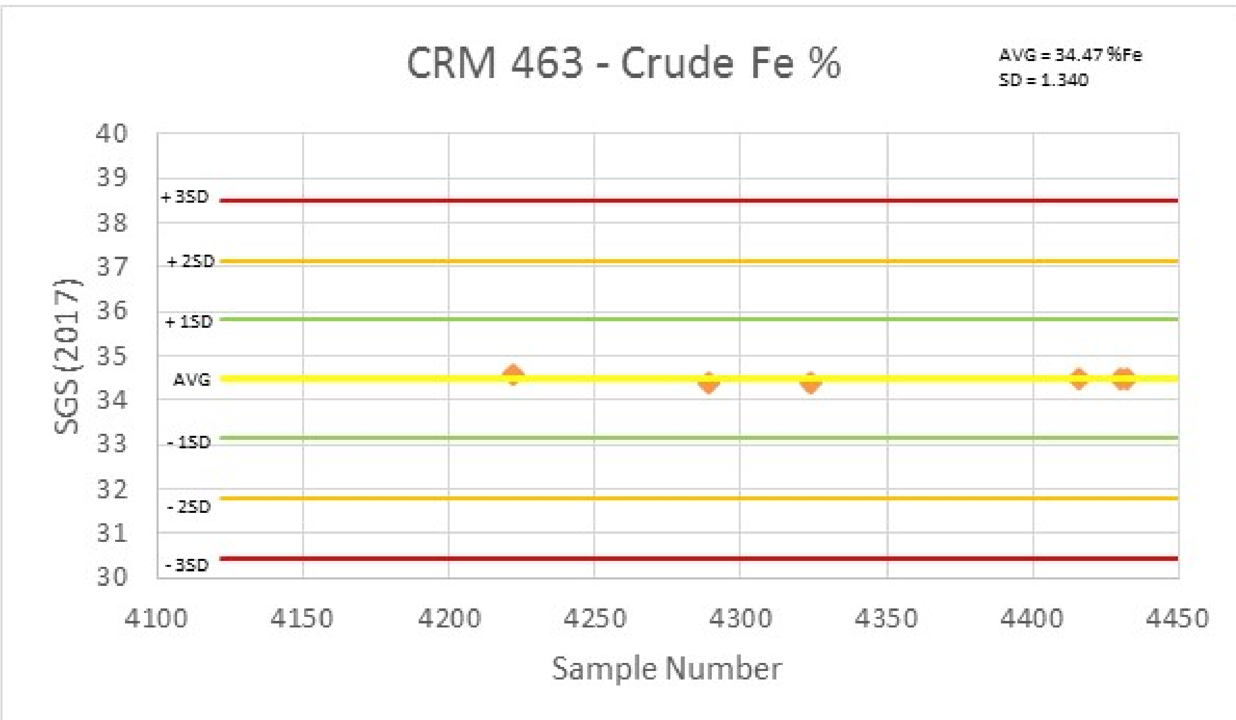


Figure 12.11: Assay Results for Inserted Blanks – Crude%

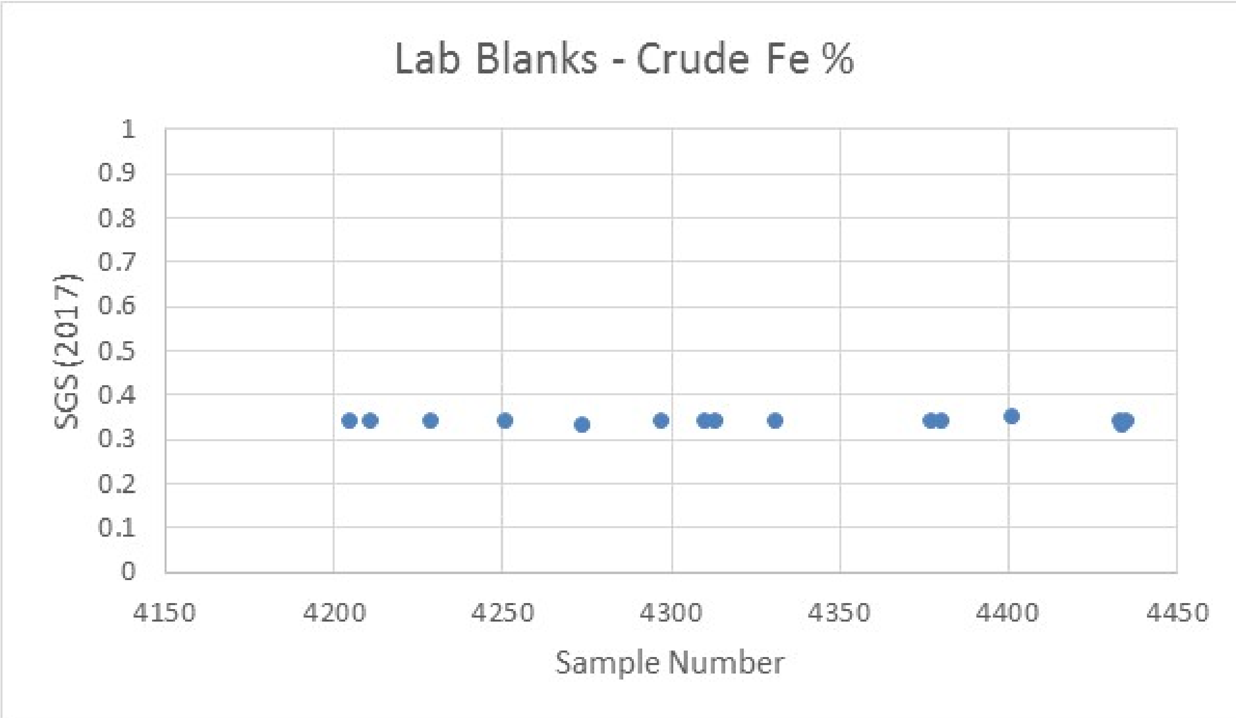
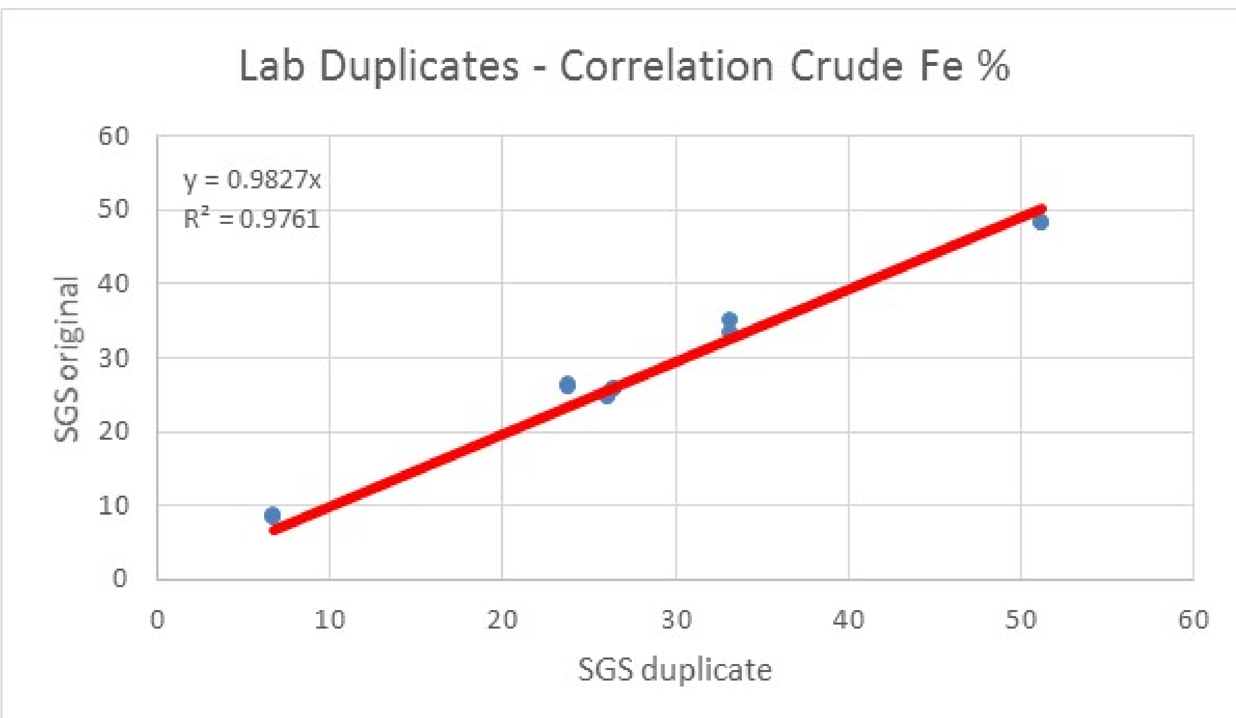


Figure 12.12: Assay Results for Duplicates Submitted to SGS – Crude Fe %



12.7 Bulk Density

The results of the 10 samples measured for bulk density is presented in Table 12.1. Bulk density was measured using the Archimedes method (a water displacement method where the sample is weighted in air, and then weight in water). Bulk density is calculated as the mass of the sample in air divided by the volume (difference between the mass in air and water). The bulk density measurements show that there is a correlation between crude Fe% and density (R^2 of 0.35) when ignoring a single outlier.

Table 12.1: Bulk Density Measurements on Drill Core from the Middle and Lower Members

Hole-ID	From	To	Length	Rock Type	FE	MN	MAG	New Sample Number	Density
BT 12-01	169	177	8	33	35.30	0.01	0.27	4426	2.55
BT 11-03	110.0	120.0	10.0	53	29.09	5.78	1.01	4270	2.88
BT 11-04	560	573	13	51	36.07	3.60	2.77	4355	2.97
BT 11-03	41	50	9	52	34.30	1.69	1.89	4424	2.97
BT 11-01	320	326	6	51	29.24	3.71	0.61	4357	3.00
BT 11-05	340.0	350.0	10.0	52	28.85	0.21	0.81	4276	3.18
BT 12-01	376.0	386.0	10.0	34	43.00	0.39	4.12	4292	3.21
BT 11-05	630	639	9	53	38.41	2.40	2.30	4353	3.25
BT 11-05	330	340	10	52	35.76	0.14	1.49	4346	3.41
BT 11-01	360.0	370.0	10.0	34	40.79	0.42	2.70	4252	3.48

12.8 Recommendations

GMS outlines its recommendations relating to the re-assaying program below:

- The correlation between the Scully drilling database and the SGS assay results for Fe% is acceptable, but the trend appears to show that the Scully drilling database slightly over-represents (overestimates) Crude Fe%. GMS does not consider this critical, due to the operating history and emphasis on concentrate grades. However, the reader should be aware of this conclusion.
- Blanks, duplicates and CRM results were returned in line with expectations.
- No bulk density database exists for the Scully Mine. The collection of 10 samples during the re-sampling program shows that the values are acceptable for an iron deposit of this style, however, 10 measurements is considered limited and not conclusive. GMS recommends that bulk density

sampling should be done systematically every 20 m downhole on all new drilling, to capture the variability between rock types. A piece of core of around 10 cm in length should suffice for testing using the water displacement method. Core should be waxed prior to testing if porosity is a concern.

12.9 Conclusion

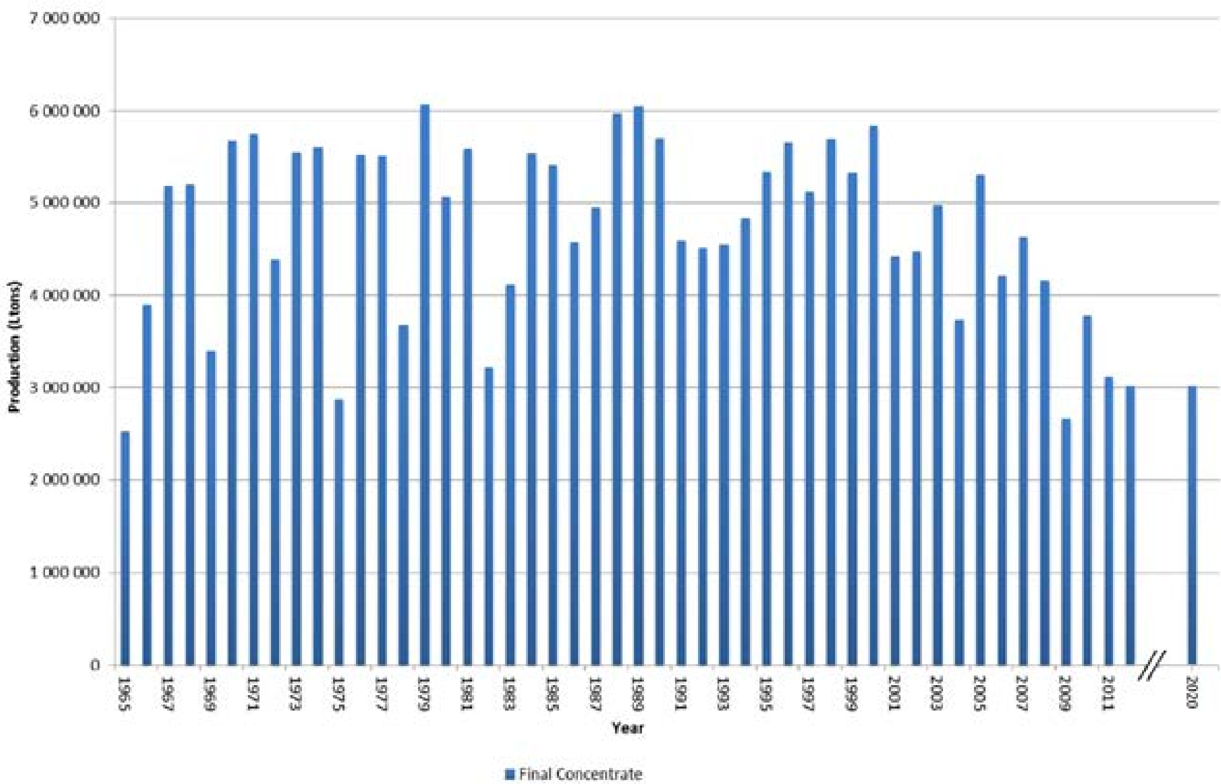
GMS believes that the 2017 re-assaying program was completed according to CIM Guidelines for Best Practices and conform to NI 43-101 requirements.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The Scully Concentrator had been in operation from 1965 to 2014, when the mine was shut down by Cliffs Natural Resources, and then was back in operation in 2019 after the mine was acquired by Tacora Resources Inc. ("Tacora"). Twice in its history the plant produced more than 6 Mt of concentrate per year as shown in Figure 13.1. Idling the pellet plant in Sept-Iles as part of the transformation chain with the plan of selling concentrate to the seaborne market instead of pellets removed all constraints that historically came from pellet production. Without those constraints of pellet production, the equipment capacities at the Scully Mine support concentrate production in excess of 6 Mtpy.

Figure 13.1: Historical Concentrate Production



13.2 Production Improvement Initiatives

To increase performance and availability of the plant, some modifications were made prior to restarting the operations in 2019. Among the most important is the addition of four (4) manganese removal lines.

Since the plant restart in June 2019 and following the availability and concentrate production results obtained in 2020, some opportunities were identified to reduce significantly plant downtime and increase the final concentrate production. The following modifications will be made during the year 2021 in order to reach the production target of 6 Mtpy:

- Plant and mills maintenance tactics review: Optimize the efficiency of the mill planned maintenance particularly mill relining, improve parts availability and reduce crude end feed outages.
- Mill main motor: Replace the mill starters and relays, secure additional mill motor spares and optimize the motor installation procedure.
- Mill lubrication system: Improve the mill lube systems with newly designed ones.
- Mill gear replacement: Replace the mill gear on two (2) mills to resolve vibration issues. The vibration is significant, and the mill throughput has to be reduced to avoid major failure.
- Mill recirculation line and pump redundancy: Replacement of the pumps used to return the oversize material from the autogenous (“AG”) mill discharge back to the mill. Due to the nature of the material containing very little fines and the size of the pipe, a lot of water must be added with the slurry returning to the AG mill. This also resulted in a poor pump reliability. A new pump type will be tested, and pump redundancy will be added.
- Crusher discharge conveyor belt: Replacement of the belt that caused significant downtimes since the conveyor feeds all the mill lines.
- Scavenger spirals: Add new scavenger spirals to process the hydrosizer overflow which would increase the recovery. The gravity circuit recovery will be increased by processing the fine material separately from the coarse particles.
- Modernize mill instrumentation: Install new scales to improve the accuracy of the process controls, reporting and mass balancing. Replace spiral feed flowmeters to steady and better control the spiral feed flow. Add flowmeters and control valves to better control the amount of water added with fresh ore to the AG mill feed chute.
- Drum filter upgrade: Replace cyclones to increase mill availability when the filters are down.
- High tension scavenger line: Restart the high-tension scavenger line that is part of the manganese recovery plant to recover additional material from the manganese tails.

Considering these modifications, the targeted mill availability is 89.5% as of 2022. In 2021, the mill availability will be gradually increased to reach that target by the end of the year. This target is highly dependent on the success of the above listed improvement initiatives.

13.3 Feed Material

Core sample characterization was made in 2012 at COREM (Report No T1298). Table 13.1 is extracted from this report and shows the JKSimMet simulation results for the major units fed to the process plant.

Table 13.1: Pit Sub-Unit JKSimMet Simulation Results

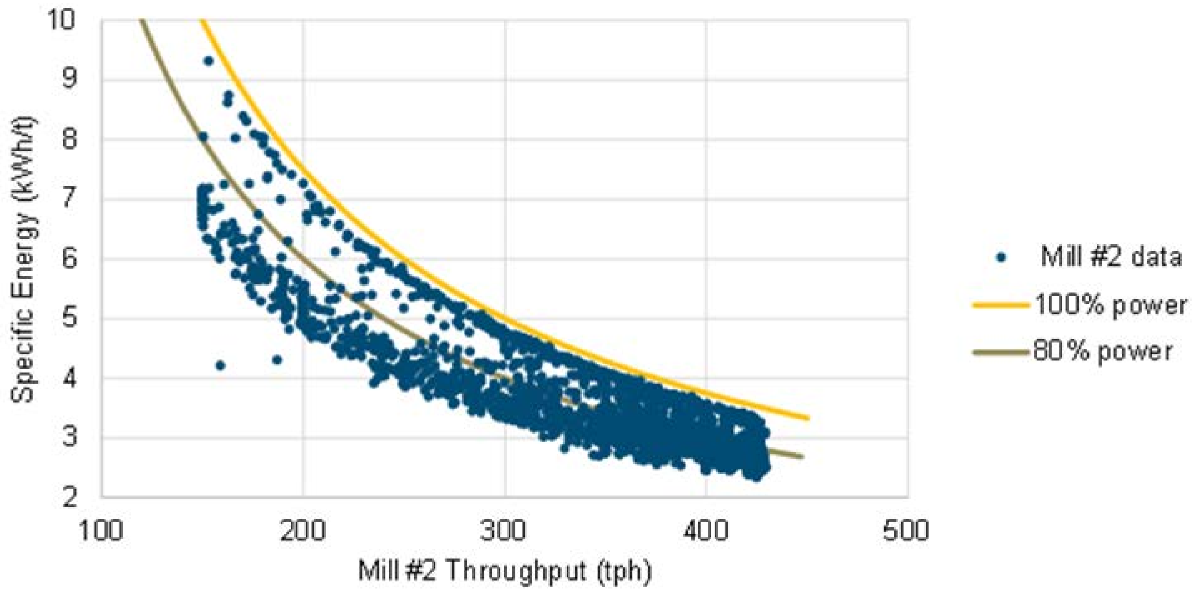
Sample	Sample characteristics					Sample grindability			
	A	b	t_a	Density (g/cc)	A.b	Energy (kW)	Fresh Feed (tph)	Grindability (kWh/t)	Load (%Vol)
SP-51	75.98	1.75	0.72	3.27	132.72	655	260	2.52	19.61
SP-52	78.22	1.37	0.53	3.08	107.52	626	216	2.90	19.58
SP-53	75.45	2.69	1.68	3.35	203.34	671	307	2.19	19.62
WP-51	91.32	7.63	1.50	3.57	696.33	742	738	1.01	19.60
WP-51	91.32	7.63	2.00	3.57	696.33	740	725	1.02	19.58
WP-52	71.09	2.41	1.74	3.15	171.13	638	242	2.64	19.65
WP-53	86.65	4.04	2.08	3.28	350.34	676	442	1.53	19.62
EP-51	87.94	3.20	2.18	3.40	281.30	685	380	1.80	19.62
EP-52	81.58	1.59	0.68	3.28	129.52	656	258	2.54	19.58
EP-53	81.72	3.35	1.53	3.37	273.42	684	406	1.68	19.63

Based on the reserves and the previous grind data obtained at Corem (see Report T-787, T-919 and T-1298), it is important to constrain the quantity of ore from Subunit 52 in any given year because of its hardness (A*b < 200 explained by its high magnetite content) in order to maintain the targeted throughputs. Specifically, the proportion of material from Subunit SP52 needs to be limited to a maximum of 15% and not to be blended with any other similarly hard materials from other pits. More details regarding Subunit 52 are presented in Section 14.

13.4 Material Throughput

The six (6) mills are each equipped with one 1,750-hp motor which has been modified to allow up to 1,500 kW of power draw, instead of the 1,300 kW normally drawn from such a motor. Figure 13.2 shows the throughput obtained and the corresponding specific energy calculated with the mill power for data gathered between July and December 2020 on Mill #2. This mill is one of the currently best performing mills, being operated most often at a throughput above 400 tph. The orange curve represents the limit of throughput achievable at a given ore hardness considering that 100% of the power is used, and the grey curve shows the 80% threshold of the power limit. A mill operated at the limit of its practical capacity would normally see all points located between the grey and orange curves and showing operations data in the 350-450 tph range.

Figure 13.2: Mill #2 Throughput vs. Specific Energy



In order to estimate the achievable throughput per mill, it was considered that a new control strategy and mill operating philosophy, involving a higher utilization of the mill power is achievable without significant investment.

The throughput which can theoretically be obtained when the power is increased to a conservative minimum value of 85% was determined for each point whose power is less than 85% while keeping the specific energy constant. The predicted throughput was capped at 450 tph which is a realistic value for the mill. Mill tonnages under 125 tph were removed from the mill throughput prediction in order to eliminate the values when the AG mill is down or in ramp up and during which the power use model is not reliable.

The data from Mills #2, #3, #5 and #6 were used to calculate the prediction; the other two (2) mills were not considered because of their vibration problems leading to a throughput limitation at 300 tph. This issue will be addressed in 2021 as outlined in Subsection 13.2.

The average predicted throughput obtained is 385 tph (dry solids) per mill.

Such throughput can only be reached if:

- The modifications listed in Subsection 13.2 are realized.
- A control system is put in place so the mill feed rate is automatically maximized based on power limitations, which would increase the throughput until a 90% of maximum power is reached.

- The maximal operating throughput is increased to 450 tph for all mills.
- The operating mindset in the Centralized Control Room ("CCR") is shifted to a more aggressive operating strategy aiming at maximizing tonnage at all time. An example of this new mindset is that mill feed chute blockages should be expected in a normal mill operation in order to use the mills at their full capacity. In 2020, less than 60 hours of downtime was observed for chutes blockage, i.e., less than 10 h/mill. This rate of blockage is too low and indicates that mills are operated in a way to avoid blockages instead of maximizing throughput. Benchmark mills where proper chute design is in place and where throughput is prioritized normally experience more frequent blockage, showing that the equipment is used at its full capacity. For instance, if 40 h of blockages are experienced per mill (four (4) times more than in 2020) and the throughput is increased by 15 tph, this would result in an increase in final concentrate production of 3.5%.

13.5 Wet Circuit Recovery

The wet circuit recovery is determined from the shaking table results using a drillhole database provided by Tacora. Historic reconciliations provided by Tacora suggest that there is a difference of around 5% between the weight recovery reported by the shaking table and the one observed through the processing plant, with the shaker table reporting higher weight recoveries. Considering that the addition of the scavenger spirals will damp this difference, a correction of 3% is made to the wet recoveries obtained from the drillhole database.

13.6 Scavenger Spirals

Iron is recovered in spirals thanks to the difference in density between the iron and gangue particles. However, when small particles are mixed with large particles, the separation of the iron present in the small particles cannot be done efficiently since the small particles are entrained with the gangue even if they contain iron. Thus, it is advantageous to separate the fine particles from the coarser particles and process the two (2) streams in separate pieces of equipment.

In order to increase the gravity circuit recovery, the overflow of the hydrosizer that contains fine iron, will be processed in dedicated scavenger spirals instead of being recirculated to the rougher spirals. The implementation of scavenger spirals is stagewise planned from March 2021 until the end of 2022. Dedicated spirals with specific operating parameters should allow recovering more efficiently fine iron instead of losing it in the rougher spirals.

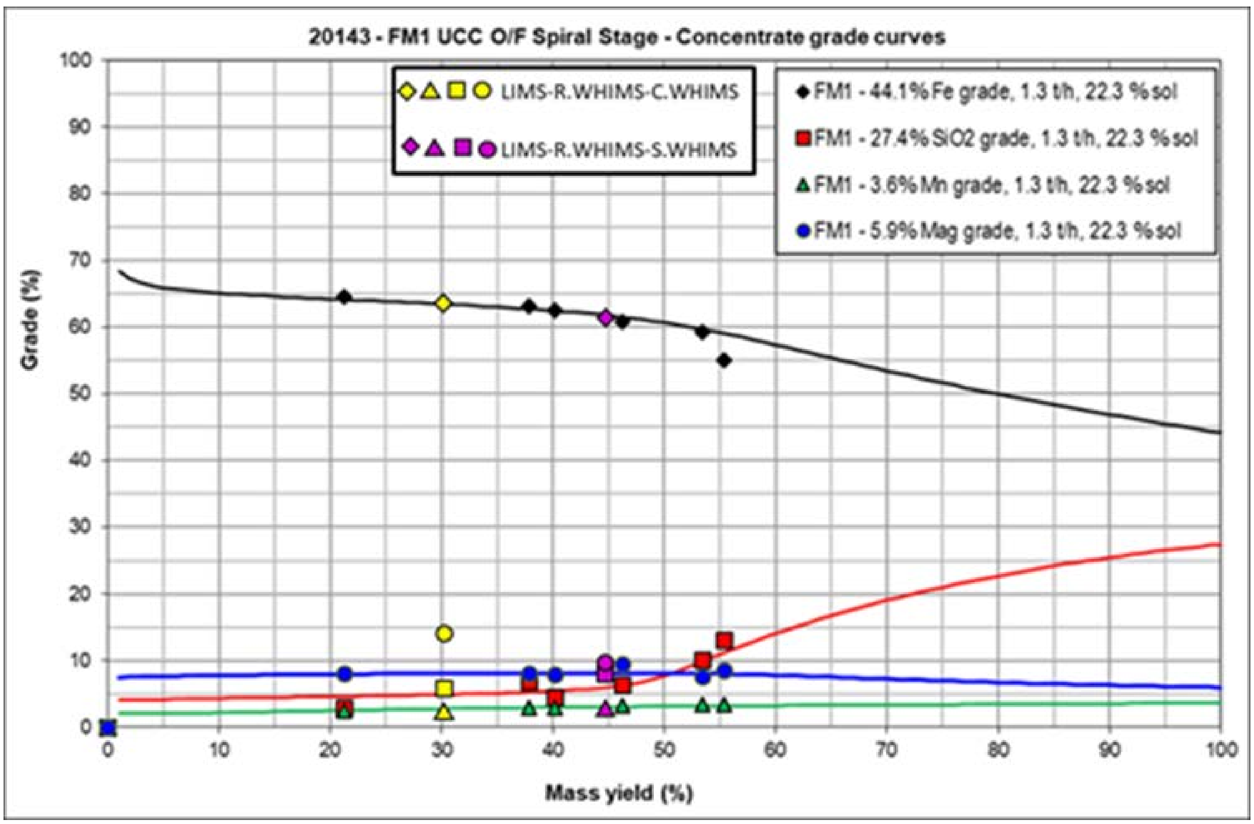
Test work realized at Mineral Technologies laboratory in 2020 demonstrated the positive impact of scavenger spirals on the final concentrate mass yield. The laboratory tests were made on the AG Mill

Discharge screened to minus 140 mesh used as feed to the scavenger spirals. This allowed to simulate the material from the Hydrosizer Overflow (HOF). The composition of the feed samples used for spiral testing is presented in Table 13.2. The results of the laboratory study in terms of grade versus mass yield are also presented in Figure 13.3.

Table 13.2: Grade Composition of Feed Samples for Spiral Testing

Feed Sample for Fine Spiral Testing	
Fe%	42.13
SiO2%	28.22
Mn%	3.92
SAT%	5.4

Figure 13.3: Concentrate Grade and Recovery Curves in Fine Spiral Testing



From Figure 13.3, using the typical Fe concentration in the plant for the scavenger spiral feed stream, the concentrate iron grade could be around 64% for a mass yield of 30%.

13.7 Manganese Removal Circuit

To reduce the quantity of Manganese in the concentrate below 2.0% and have access to a larger ore reserve, four (4) additional Mn removal lines were installed for the 2019 restart in addition to the existing two (2). The manganese removal lines include LIMS, rougher HIMS, scavenger HIMS and a high-tension scavenger circuit made of high-tension separators and cleaner HIMS.

The Pilot testing reports[1][2]explained and demonstrated the manganese and silica removal performance of the first pilot line. A weight recovery of 90% and an iron recovery of 96.5% was achieved, while a 50% manganese removal and a 44% silica removal were realised.

Pilot plant tests and the results of the first Mn removal line demonstrated[3] that the manganese removal line feed rate influences the manganese and silica removal efficiency and iron recovery[4]. Indeed, efficiency of the iron recovery is reduced at higher tonnages since a thicker layer of material is present on the drums of the magnetic separators. A maximum feed rate of around 60 tph per half line was determined from this study as higher feed rates show lower performances in manganese removal.

However, recent operation data from 2019 and 2020 provided by Tacora[5], showed that good recovery is possible even at higher tonnage (illustrated in Figure 13.4). In addition, considering that the final Mn target is high (2.0% Mn) with respect to the Mine plan average grade (2.6% Mn), MRC performances were considered unaffected by throughput for the operating range considered.

[1] Soutex Inc., Project 2156, *Mn Reduction Plant Metallurgical Summary Report*, version 1, 2008-10-16.

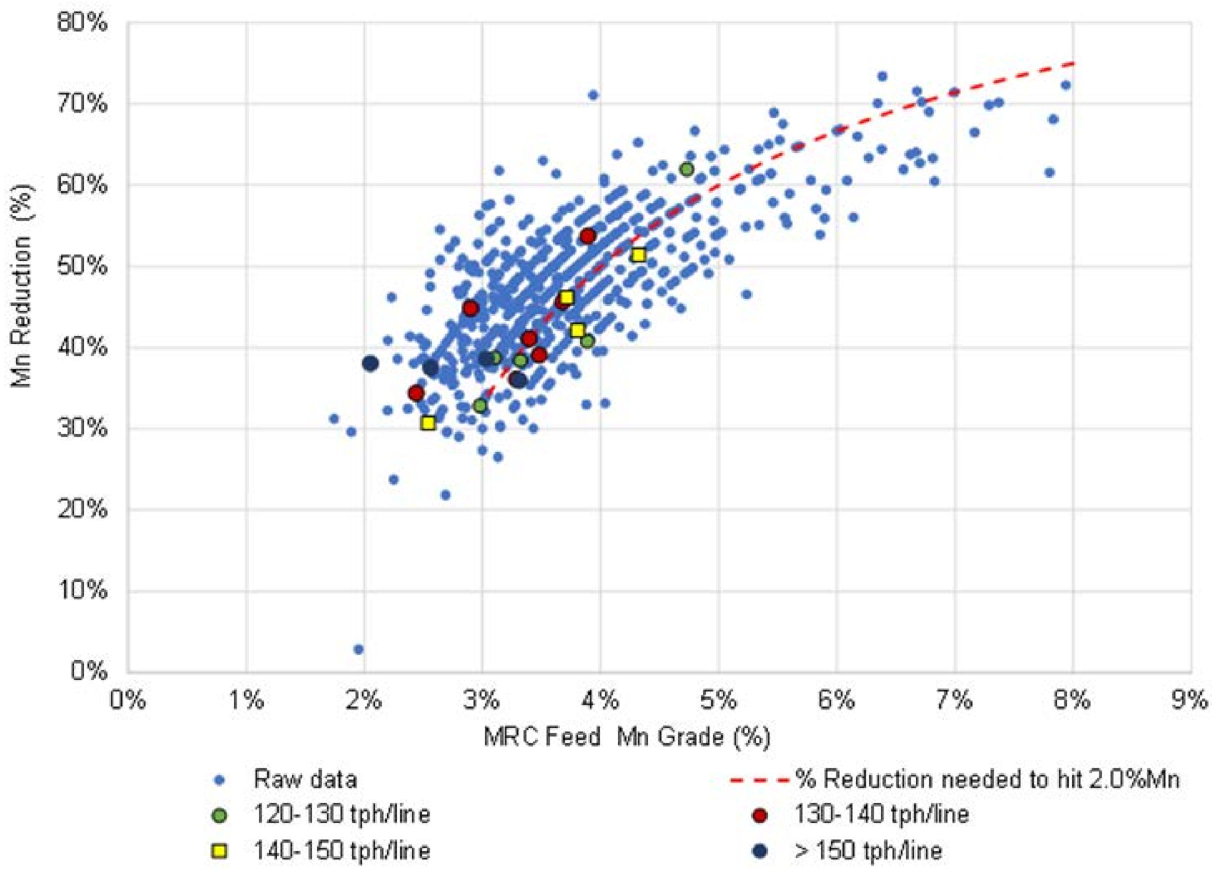
[2] Soutex Inc., Project 2156, *Technical Note on Mn Testing Review*, version 1, 2008-03-04.

[3] Soutex Inc., Project 2215*, Manganese Removal Line 1 Commissioning & Start Up Report*, version 2, May 2010.

[4] Soutex. Inc., *op. cit.*

[5] Tacora Resources, *Mine to Mill Recovery Model*, 2021/02/05.

Figure 13.4: Manganese Removal as a Function of Feed Manganese Grade



13.7.1 <u>High Tension Scavenger Circuit</u>

Since the restart of the plant in 2019, the high-tension scavenger circuit was not in use due to operational issues.

The increased recovery in the high-tension scavenger circuit had been demonstrated in the pilot plant[6] and an additional laboratory study was carried out at Tacora's laboratory in order to demonstrate the potential recovery increase with the use of the high-tension scavenger circuit. Different operating parameters were tested to produce various concentrate grades and to maximize total dry circuit recovery.

The test work was realized on the MRC tails sample. First, the material was fed to a high-tension separator and then, its concentrate was fed to a HIMS.

[6] Soutex Inc., Project 2387, *Commissioning Report of the Scavenger Circuit, Scully Mine*, version 1, February 2014.

The settings used to conduct the test work are presented in Table 13.3. Test results obtained for the high-tension scavenger separator are presented in Table 13.4.

Table 13.3: HT Bulk Settings

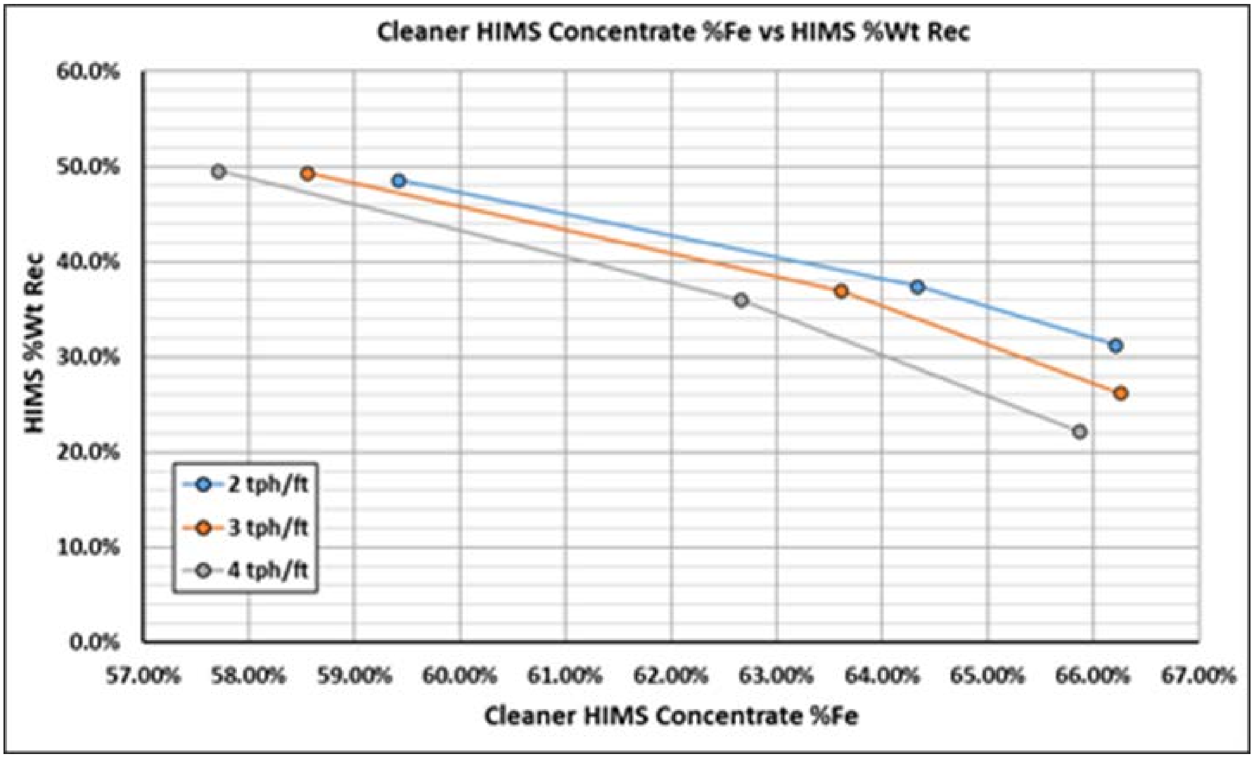
HT Scavenger Settings	
Feed Rate (MTPH/Length of Drum)	0.45
Drum Speed (rpm)	95
Concentrate Splitter Position (value)	43
Tailings Splitter Position (value)	80
Voltage (kV)	35
Electrode 1 angle (°)	48
Electrode 1 distance from drum (cm)	11.5
Electrode 2 position (°)	33
Electrode 2 distance from drum (cm)	11

Table 13.4: High-Tension Separator Testing Results

	HT Feed	HT Concentrate	HT Mids	HT Tails
%Fe	32.75%	39.04%	30.12%	2.82%
%SiO2	21.68%	6.66%	32.49%	88.35%
%Mn	16.58%	21.17%	12.20%	0.36%
%Mag	0.39%	0.20%	0.49%	0.89%
%Wt Rec	100.0%	68.3%	18.8%	12.9%
%Fe Rec	100.0%	81.56%	17.3%	1.1%
%SiO2 Rec	100.0%	20.60%	27.7%	51.7%
%Mn Rec	100.0%	86.07%	13.7%	0.3%

Different settings of feed rate, drum speed and splitter positions were used to investigate the performances of the HIMS cleaners. The results of the HIMS concentrate weight recovery as a function of the concentrate iron grade are presented in the Figure 13.5.

Figure 13.5: Results for Cleaner HIMS



The recovery performances of the high-tension scavenger separator and the cleaner HIMS were combined to obtain the overall performances of the MRC scavenger circuit. These performances are presented in Table 13.5.

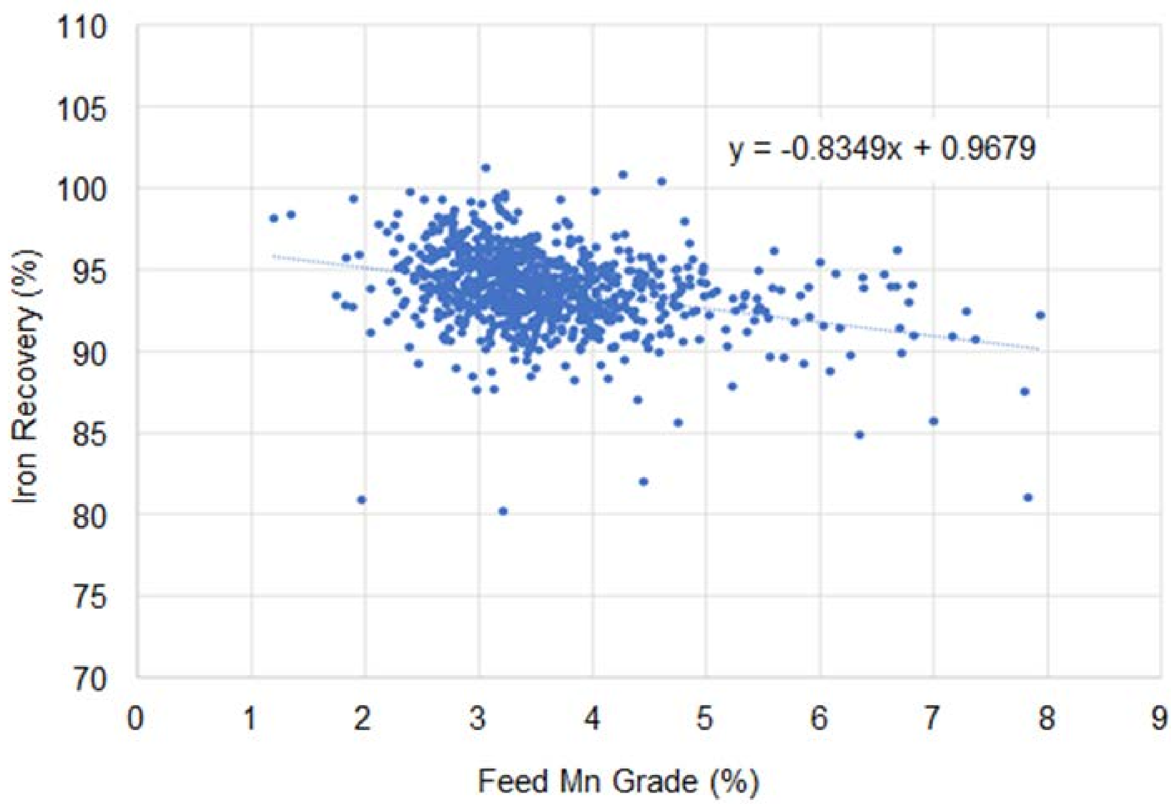
Table 13.5: Overall Recovery Results for the Scavenger Circuit

			Overall Recovery			
Test #	Feeder Setting (%)	HIMS Feed Rate (tph/ft drum length)	Overall %Wt Rec	Overall %Fe Rec	Overall $\%SiO_2$ Rec	Overall %Mn Rec
Test #1	22	1.90	33.2%	60.5%	3.5%	13.4%
Test #2	22	1.90	25.6%	51.4%	1.7%	3.9%
Test #3	22	1.95	21.4%	43.1%	1.2%	2.5%
Test #4	28	2.91	33.7%	60.5%	4.1%	14.8%
Test #5	28	2.93	25.2%	50.5%	1.9%	4.9%
Test #6	28	2.94	17.9%	36.7%	0.9%	2.1%
Test #7	34	3.61	33.8%	59.6%	3.8%	16.5%
Test #8	34	3.68	24.6%	47.3%	1.8%	5.9%
Test #9	34	3.81	15.1%	30.1%	0.8%	1.8%

13.7.2 Recovery Model

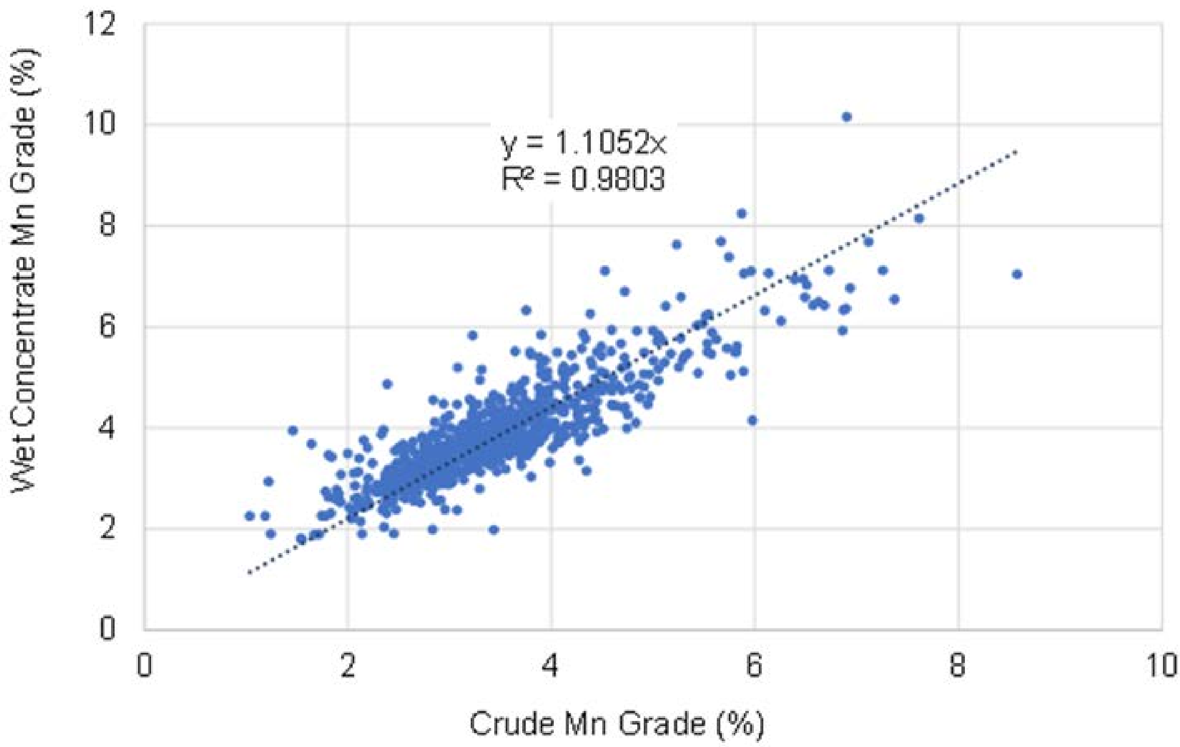
An empirical recovery model was developed to predict the MRC iron recovery and weight recovery using the operation data since restart. The trend obtained for the MRC iron recovery as a function of the manganese MRC feed grade is shown in Figure 13.6. The recovery values obtained are for the first stage of the MRC circuit: they do not include the high-tension scavenger circuit which was not in operation since the start-up.

Figure 13.6: MRC Fe Recovery Plant Data



The manganese MRC feed grade corresponds to the gravity concentrate manganese grade which can be estimated from the crude manganese grade regression as shown in Figure 13.7.

Figure 13.7: Relationship between Gravity Concentrate Mn Grade and Crude Mn Grade



The MRC iron recovery excluding the HT scavenger circuit can be calculated with the trend obtained from Figure 13.6. The additional recovery gained in the HT scavenger circuit is added to obtain the overall MRC iron recovery. The following equations are obtained:

$$Fe\ Rec_{MRC\ 1} = \left(-0.8349 * \ Mn_{grav} + 0.9679\right)$$

$$Fe\ Rec_{MRC} = (Fe\ Rec_{MRC1} + 0.42 * (1 - Fe\ Rec_{MRC1})$$

The gravity concentrate manganese grade can be calculated from the following equation:

$$Mn_{grav} = 1.1052 * \ Mn_{crude}$$

The MRC weight recovery excluding the HT scavenger circuit and including the HT scavenger circuit can be calculated as follows:

$$WRec_{MRC\,1} = \ Fe\ Rec_{MRC} * \frac{Fe_{grav}}{Fe_{final}})$$

$$WRec_{MRC} = (\,WRec_{MRC1} + 0.18 * \ (100\% - \ WRec_{MRC1}))$$

Where,

- Fe $Rec_{MRC\ 1:}$ Iron recovery in the MRC without the HT scavenger circuit (%).
- Fe $Rec_{MRC:}$ Iron recovery in the MRC with the HT scavenger circuit (%).
- W $Rec_{MRC\ 1:}$ Weight recovery in the MRC without the HT scavenger circuit (%).
- W $Rec_{MRC:}$ Weight recovery in the MRC with the HT scavenger circuit (%).
- Mn_{grav}: Gravity concentrate manganese grade (%).
- Mn_{crude}: Crude ore manganese grade (%).
- Fe_{grav}: Gravity concentrate iron grade (%).
- Fe_{final}: Final concentrate iron grade (%).

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The 2021 Mineral Resource Estimate ("MRE") of the Scully Deposit uses the same exploration drilling database and generally follows the same methodology as applied in the 2018 MRE. The key addition have been a reconciliation exercise to review the performance of the 2018 block model against production since mid-2019. Moreover, the following changes were made:

- Incorporation of 2019 and 2020 blastholes into the blasthole database.
- Modifications to the weighted average grade scripts to apply tonnage weightings rather than volume weightings. In addition, overburden blocks were removed from the weighted average script to avoid diluting surface blocks with overburden (which is stripped pre-mining).
- Mining depletion was accounted for in the block model, by adjusting block percentages to account for the end of year 2020 pit survey.
- Pit optimisations were run with updated assumptions, and we used to constrain the report the Mineral Resource.

The geological interpretation remains identical to the one used to produce the 2018 block model, and grade estimation parameters are unchanged.

14.2 Drilling Data

GMS received the drill hole database in csv format on the 31/07/2017 from representatives of Tacora Resources Inc, Grand Rapids, Minnesota. The data comprised of three (3) files; a collar file (*wab_2013_ddh_collar.cs*v), a survey file (*wab_2013_ddh_survey.csv*) and an assay file (*wab_2013_ddh_assay.csv*) which contained the relevant chemical analyses and a single logging code. In addition, a blasthole drilling database was provided on 07/09/2017 as a single csv file (*wab_2013_mod_082313_bhs_sample.csv*) which contained XYZ coordinates and chemical analyses.

No centralized QA / QC database was available and supporting drilling information was provided in .pdf format in a folder structure classified by year.

All drilling data received was converted from feet to metres using a factor of 0.3048. Down hole intervals (all in feet) were converted, and collar coordinates in the local mine grid were also multiplied by this factor.

14.2.1 Exploration Drill Holes

The drilling database provided contains 1,005 drill holes from historical drilling programs which occurred up to 2012. A total of 77,096 m of drill core covers the Scully Mine area (of which 76,643 m of downhole interval data is available).

All drilling information was used for the production of the resource block model. A list of the drill holes present in the drilling database is presented in Table 14.1 by year of drilling.

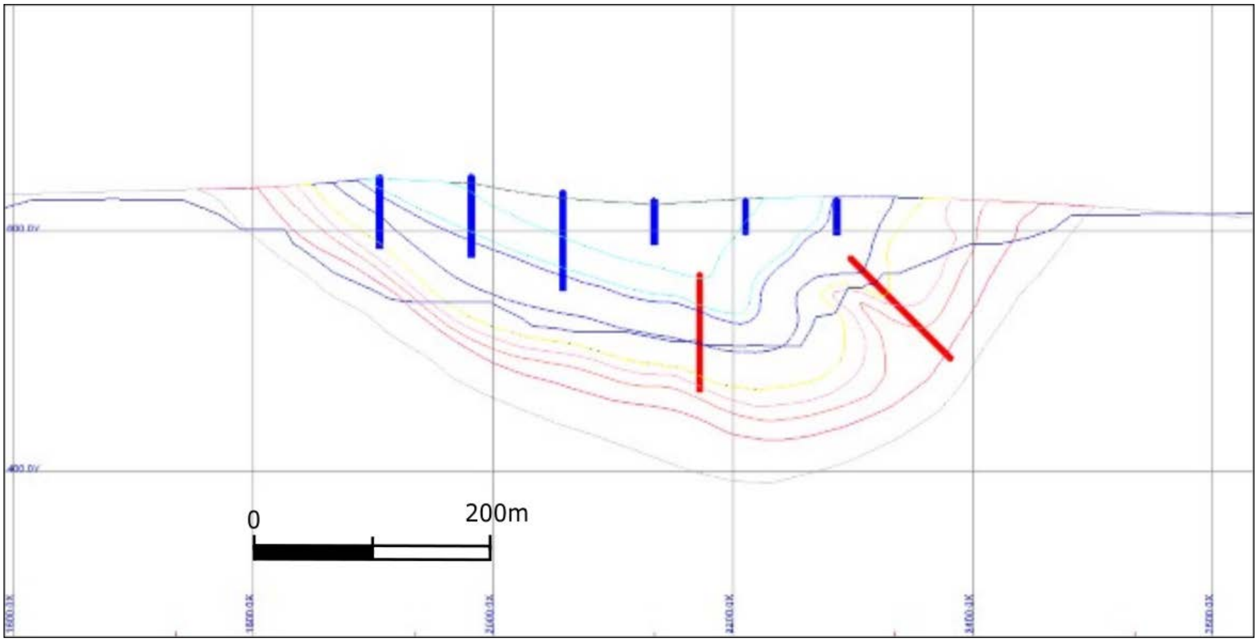
Table 14.1: List of Drill Holes Located Inside the Block Model Limits

Year	Hole ID	Number of Holes	Meterage (m)	Average Drill Hole Length
Unknown	Incremental numeric values from 1 to 1,063 with many missing values	490	29,878	61
1969 - 1974	Prefix 69-, 70- 71-, 72-, 73- and 74-	149	11,287	76
1976	Prefix 76-	50	3,918	78
1982 - 1983	Prefix 82-, 83-	34	2,078	61
1991	Prefix 91-	41	4.486	109
1994	Prefix 94-	24	2,488	104
1996 - 1998	Prefix 96-, 97-, and 98-	79	8,991	113
2001	Prefix 2001-	7	1,070	153
2004 - 2005	Prefix 2004-, EE05-, EW05-, HL05-, SW05-, WP05-, WX05-	60	5,854	98
2008	Prefix WX08-, SP08-, BT08-, EPE08-, EPW08-	22	1,462	69
2010 - 2012	Prefix BT10-, WP10-, BT11-, EPE11-, SP11-, WP11-, BT12-, EPE12-, SP12-, WPX12-	49	5,584	114
Total Number of Holes Within Block Model Limits		**1,005**	**77,096**	**77**

A large proportion of the drilling database in terms of total meterage are from an unknown time period (39% of drilled metres), and are assumed to be very early, pre-production drill holes. They are defined by an incremental numerical drill hole ID, whereas drilling after 1969 contains a reference to the year within the drill hole prefix. Most of these drill holes are located within the mined portions of the deposit, thus, do not add significantly to the geological model beneath the current pit topography.

Figure 14.1 shows an example of historical drilling (1974-1976 drill holes in blue) compared to more recent drilling (1996 drill holes in red), where the lower un-mined units of the stratigraphy are defined by more recent drilling.

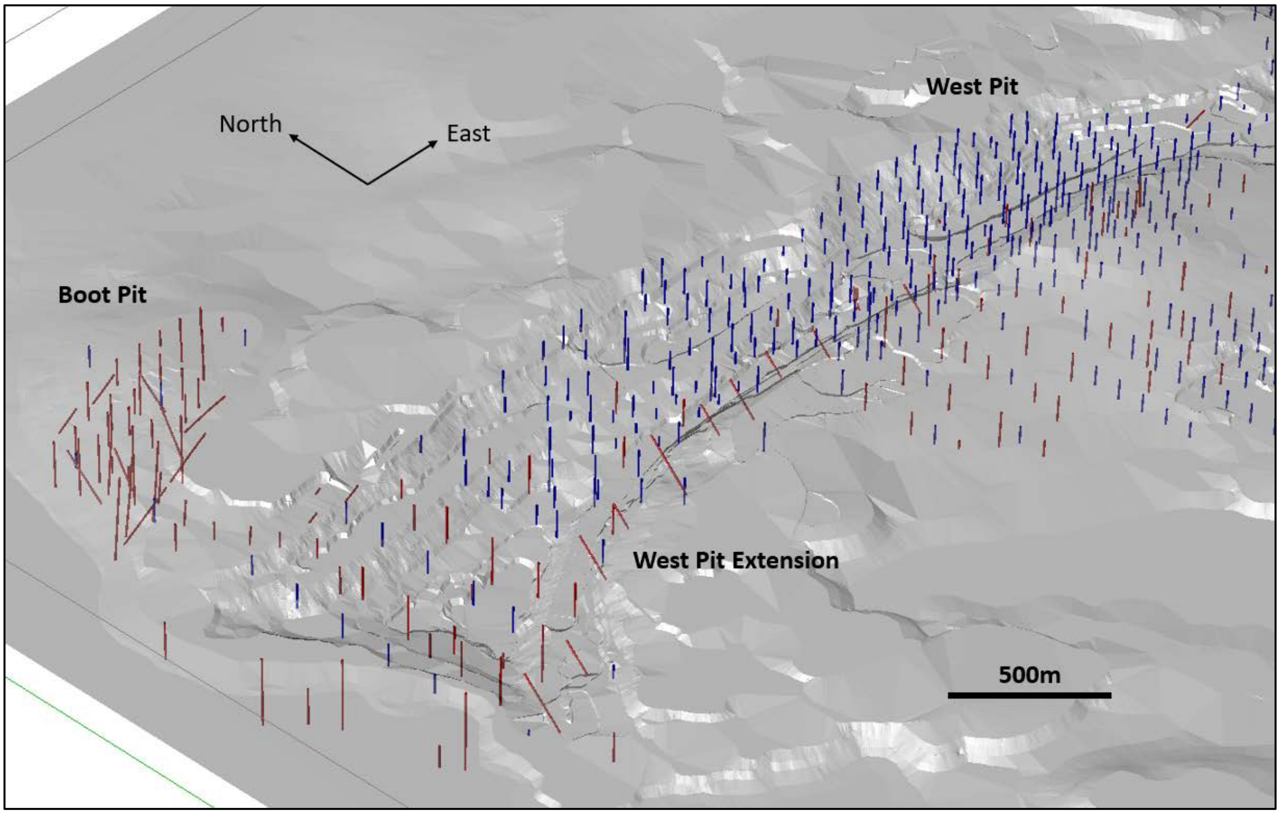
Figure 14.1: Section 2513mE Showing Historical Drilling in Blue, and Later Drilling in Red, Looking East



Historical drilling activities consisted of shallower drill holes, likely targeting near surface resources with a focus on short-term production. Since around 1985, deeper drilling has attempted to better define the extent of mineralization below surface, reflected by the deeper average drill hole length..

Figure 14.2 shows the drilling colored by year underlain by the west pit, west extension pit and boot pit. The post-1985 drilling generally defines the unmined regions or the deposit (predominantly the boot pit, and the southern wall of the west extension pit) which are considered key targets for future mining activities.

Figure 14.2: View Looking North-East Showing Pre-1985 Drilling (Blue) and Post-1985 Drilling (Red)



The assay table in the drilling database contains crude grades for Fe % and Mn %, which are present in most of drilling intervals (89% of total meterage), and crude grades of magnetite % (Satmagan) which are present in 77% of the total meterage.

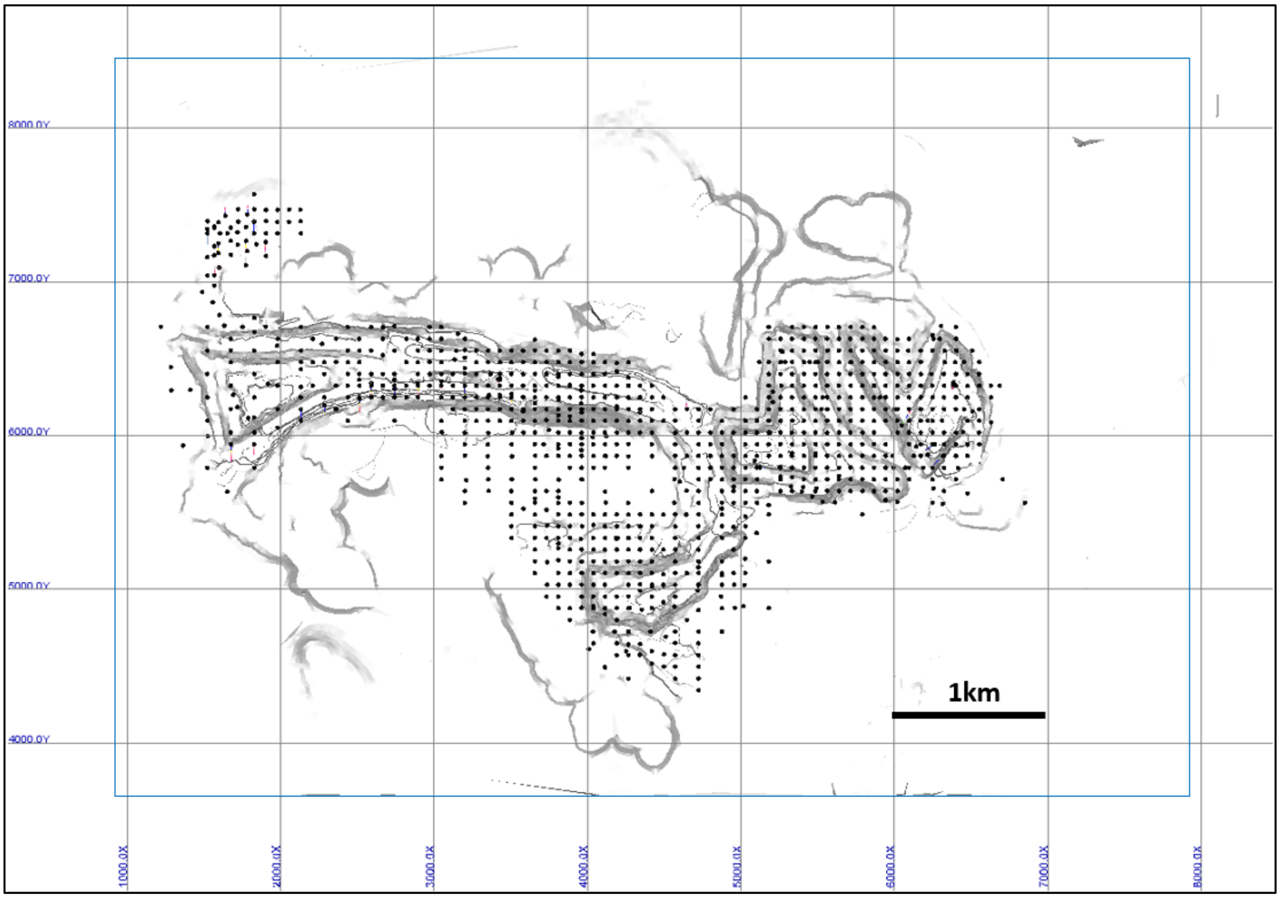
Regarding concentrate grades, iron, manganese, silica, and weight recovery are present in between 84% and 88% of the total meterage, however, magnetite % of the concentrate is only present in 27% of the total meterage. Table 14.2 summarizes the attributes found within the drilling database.

Table 14.2: Attributes Within the Drill Hole Database, and Their Abundance Compared to Total Meterage

Attribute	Description	Valid Sample Intervals	Percentage of Total Meterage
CRFE	Crude grade Fe% (head grade)	69,414 m	89%
CRMN	Crude grade Mn% (head grade)	69,051 m	89%
CRMAG	Crude grade Magnetite% (head Satmagan grade)	59,420 m	77%
WTREC	Weight Recovery, expressed as a percentage	68,615 m	88%
CONFE	Concentrate Fe%	65,334 m	84%
CONSI	Concentrate Si%	64,999 m	84%
CONMN	Concentrate Mn%	65,046 m	84%
CONMAG	Concentrate Magnetite% (Satmagan)	21,301 m	27%

Figure 14.3 shows the location of drill holes compared to the current pit topography.

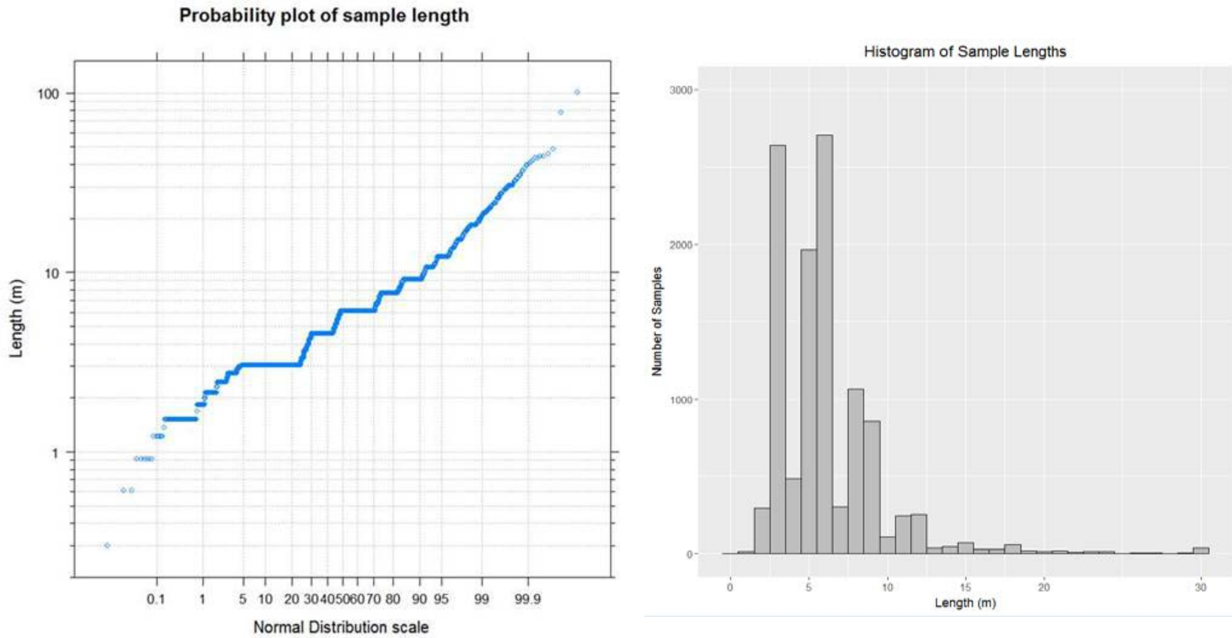
Figure 14.3: Plan View of the Diamond Drill Holes Located Inside the Block Model Area (Blue Outline)



In general, sampling of drill core has been conducted according to geological contacts at sample lengths less than the given bench height of 12.2 m (sample lengths are usually 3 m, 4.5 or 6 m); however, around 5% of the samples in the assay table have a sample length greater than 12.2 m (see Figure 14.4). Samples greater than the bench heights are considered less representative due to the high ratio between the initial sample weight and the final pulverized sample weight. On the other hand, confidence in the weight recovery percentage of concentrate increases with sample size.

Samples with a length greater than 12.2 m are generally distributed evenly across the deposit. There are insufficient samples to make a robust statistical comparison with the samples less than 12.2 m in length. GMS considers that removing these samples would have a detrimental effect on grade continuity, and a cohesive Mineral Resource classification would not be possible. Therefore, GMS believes these samples should be retained during geological modelling and estimation.

Figure 14.4: Probability Plot (Left) and Histogram (Right) or Raw Sample Lengths in the Assay Table (in metres)



The drilling database, which was imported into Geovia GEMS™ software was validated before proceeding to resource modelling and grade estimation. The following is a list of the validation checks executed on the drill hole database, and any errors found:

- Checked for duplicate drill hole collar locations and hole numbers. Six (6) pairs of drill holes were found to share the same collar coordinates and downhole positions.
- Checked collar locations for zero or extreme values. No errors were found.
- Performed visual inspection of drill holes for unusual azimuths, dips, and deviations. No errors were found.
- Cross-checked collar lengths against the final depths of the assay table. Twenty-five drill holes contained assay data longer than end-of-hole in the collar file, some caused by the conversion factor from feet to meters which produced minor rounding errors. These drill holes were lengthened to accommodate for the assay data. In addition, 31 drill holes contain end of hole depths longer than the assay data. These were cross-checked against the original drill logs.
- Ran validity checks for out-of-range values, missing intervals, overlapping intervals, out-of-sequence intervals, etc. (three (3) missing intervals identified and corrected).

- Checked the accuracy of the *length* field in the provided assay interval table. Twenty-one intervals were found to contain incorrect lengths (based off to – from calculations). Corrections were made to the data.

Although numerous errors were found in the drilling database, GMS believes the database is acceptable for resource estimation. GMS corrected any minor errors found and reimported the drilling database into Geovia GEMS™. Recommendations will be made relating to the drilling database in Section 26.

14.2.2 Blast Hole Database

The historical production blast hole database was provided to GMS in CSV format on September 7th, 2017. The database contains a single 40 ft sample per blast hole, with each sample analyzed for crude and concentrate attributes. Blastholes were converted from feet to meters (using a factor of 0.3048), and subsequently flagged with the geological wireframes presented in Section 14.3.1. New blast holes from 2019 and 2020 were also incorporated into the database.

Blasthole grades were reviewed on-section and were compared with exploration drill holes for suitability. Crude Fe% and Mn% were found to correlate well with existing drilling and geological interpretations. However, crude Magnetite % in the blast hole database was found to be significantly lower than the surrounding exploration drill holes. For this reason, the crude Magnetite % data in the blast hole database will be not considered further in the resource estimation.

14.3 Modelling

GMS was provided with various wireframes by representatives of Maptek™ Colorado, which were built based on sectional interpretations produced by Cliffs geologists before the mine ceased production in 2014. Geological sections were spaced 76 m (the nominal drill spacing, 250 ft), and were orientated depending on the strike azimuth of the deposit (sections orientated north-south for boot, west, west extension and south pits, and orientated east-west for the east pit).

Using the provided sectional interpretation and wireframes as a guide, GMS produced a new geological model using the following methodology:

- Import drill hole logging interval data and 2D cross-sections into Leapfrog GEO™ to build a geological model using implicit modelling techniques.
- Build a surface for the basal contact of each unit in the stratigraphic column using the polylines to guide the interpretation, and the drill hole geological contacts as the base information layer.

- Cut all stratigraphic units by a base of overburden surface, modelled from logged intercepts.
- Honor any faults that may offset the units (East Pit) by using separate fault blocks to constrain surface generation.
- Produce 3D solids of each stratigraphic unit for use in block modelling. The geological model will represent the deposit as "unmined" (i.e., before production commenced), to ensure that cross-validation can be undertaken against the blast hole database to measure the accuracy of the block model. For reporting purposes, only blocks below the current pit topography will be reported as Mineral Resources.

14.3.1 Geology Model

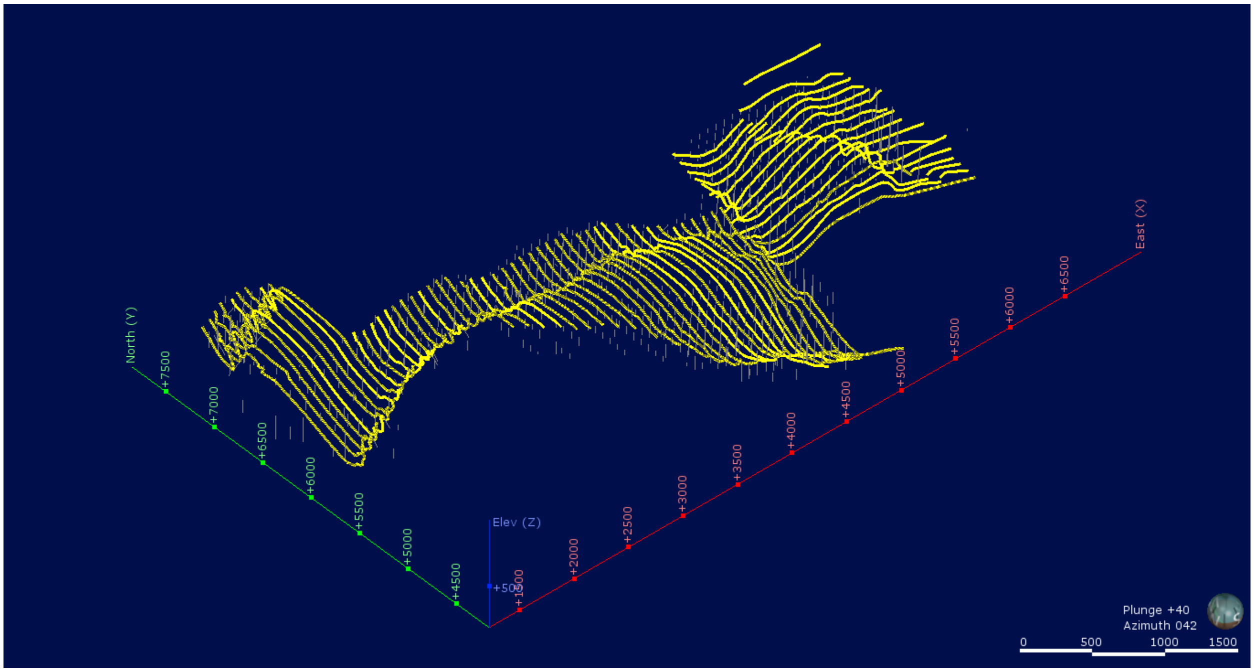
Due to the inconsistencies with the existing geological interpretation (mentioned in Section 14.3), GMS produced a new geological model using the rock code provided in the downhole assays table (*geol*). Using Leapfrog GEO™ software, a geological model was built using various stratigraphic modelling techniques and constraints. The stratigraphic column used during modelling is presented in Section 7.

All formations of the stratigraphic column were modelled:

- Upper Member (rock codes 21, 22, 23) characterized generally as low-manganese (0.6% - 0.8% Mn), goethite-rich, oxidized, specularite-poor (apart from rock code 22).
- Middle Member (rock codes 31, 32, 33, 34) characterized generally as moderate manganese (1.2%-1.6% Mn), specularite-rich, high-quality ore with high weight recoveries.
- Lower Member (rock codes 51, 52, 53) characterized as high-manganese (1.5%-3.0% Mn), specularite-rich, with high weight recoveries.
- Un-mineralized units comprising of glacial overburden (rock code 11), middle quartzite (rock code 41) and the footwall basal silicates (rock code 61).

To produce cross-sections in Geovia GEMS™, the geological wireframes provided were cut on each major drill section (76 m spacing) to produce a set of 2D interpretations, one for the base of each unit. These were imported into Leapfrog GEO™ to help guide the construction of each geological surface. An example is shown in Figure 14.5.

Figure 14.5: 2D Geological Sections Imported into Leapfrog GEO™ for the Base of Unit 34 (Base of Middle Member). Looking North-East



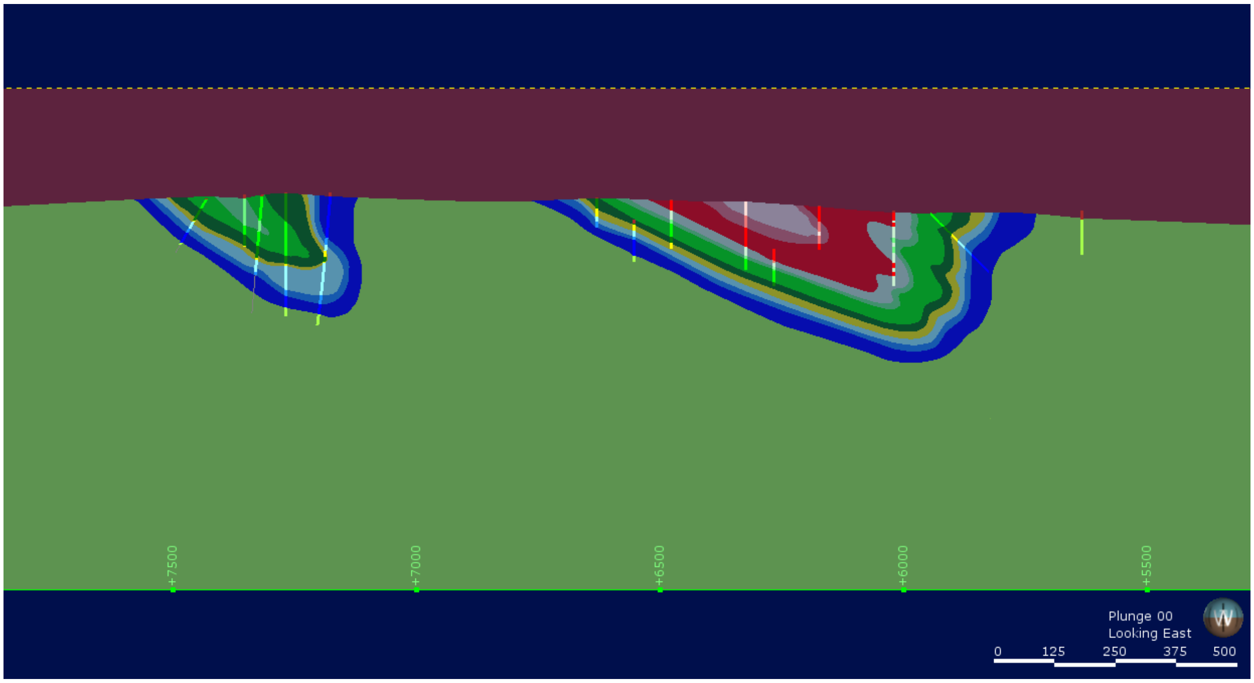
GMS reviewed the cross-sections for the base of each geological unit in Leapfrog GEO™ to ensure that polylines honored fault intersections and displacements (especially in the East Pit region). Drill hole interval data was imported and underwent further validations to ensure that the stratigraphic column was robust and any over-turned folds were well represented in the cross-sections and the drilling data.

Surfaces were then constructed using the Stratigraphic Sequence tool in Leapfrog GEO™ using a combination of drilling intercepts and 2D cross-sectional polylines in the process. The surfaces were snapped to drill hole intercepts (i.e., the surfaces pass accurately through geological boundaries in the drilling), however, polylines were used only to guide the interpretation (no snapping).

The geological model was split into two main fault blocks, which is discussed in Section 14.3.2.

All units of the stratigraphic column were constrained by the base of overburden (Section 0). A cross-section example of the new geological model is shown in Figure 14.6.

Figure 14.6: Cross-Section 1677mE in Leapfrog Geo™ Looking East Showing the Stratigraphy and Drilling



14.3.2 Fault Block Domains

The iron formations of the Scully Deposit have been affected by post-mineralization folding and faulting, resulting in two key distinct structural domains. The western domain is characterized by an asymmetrical, occasionally overturned (boot pit) E-W striking succession of synforms and antiforms, with axial planes dipping between 45 degrees and vertical to the south. No large-scale faulting has been observed in the western domain. The eastern domain is characterized by a more shallowly folded succession of N-S trending antiforms and synforms, with a significant reverse fault striking NW-SE steeply dipping to the NE. This reverse fault raises the western block, resulting in basements rocks up-thrusted to surface (Figure 14.8). The geological units in the eastern block have been deformed near vertically near the reverse fault.

GMS built a fault surface using the sectional interpretations supplied and used the fault surface to separate the geological model into two fault blocks (west and east). The reverse fault is shown in 3D in Figure 14.7.

Figure 14.7: View of the East Pit Showing the Reverse Fault (Looking North), Current Pit Topography and Drilling

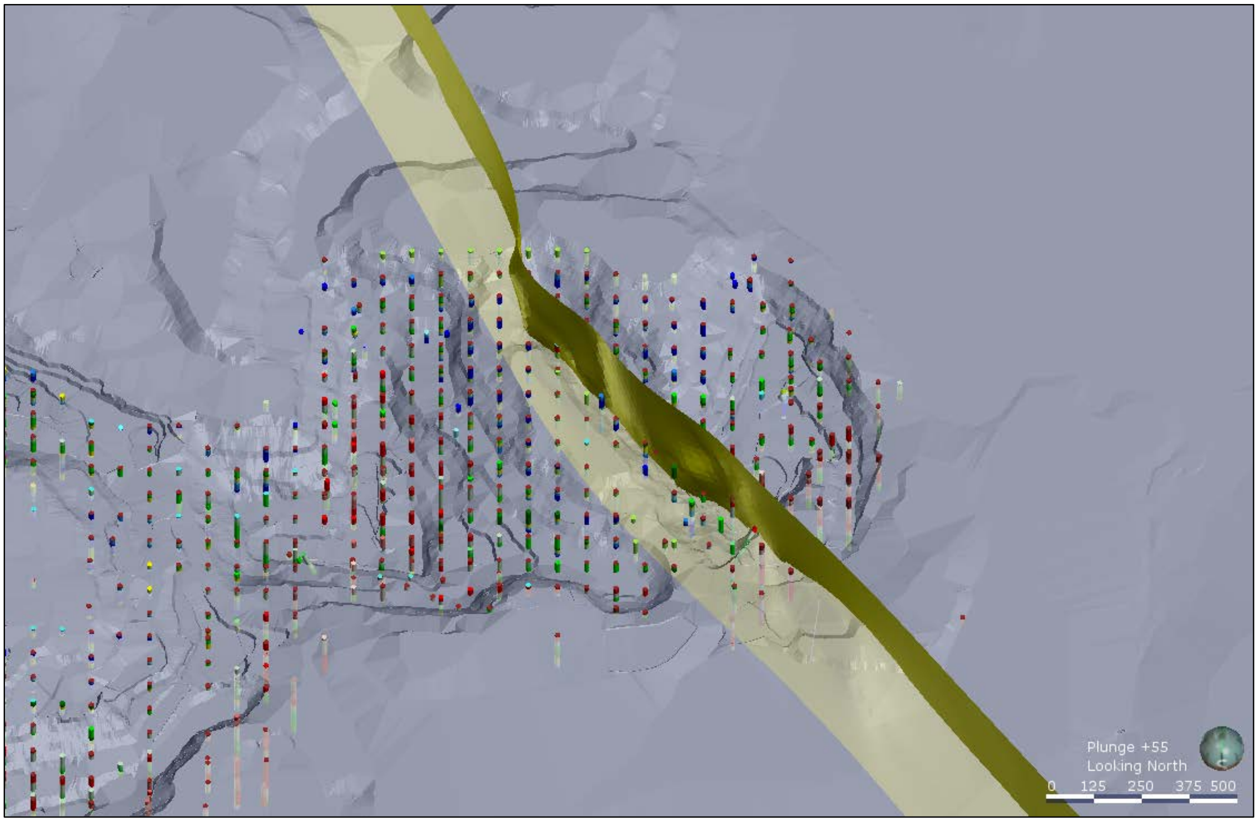
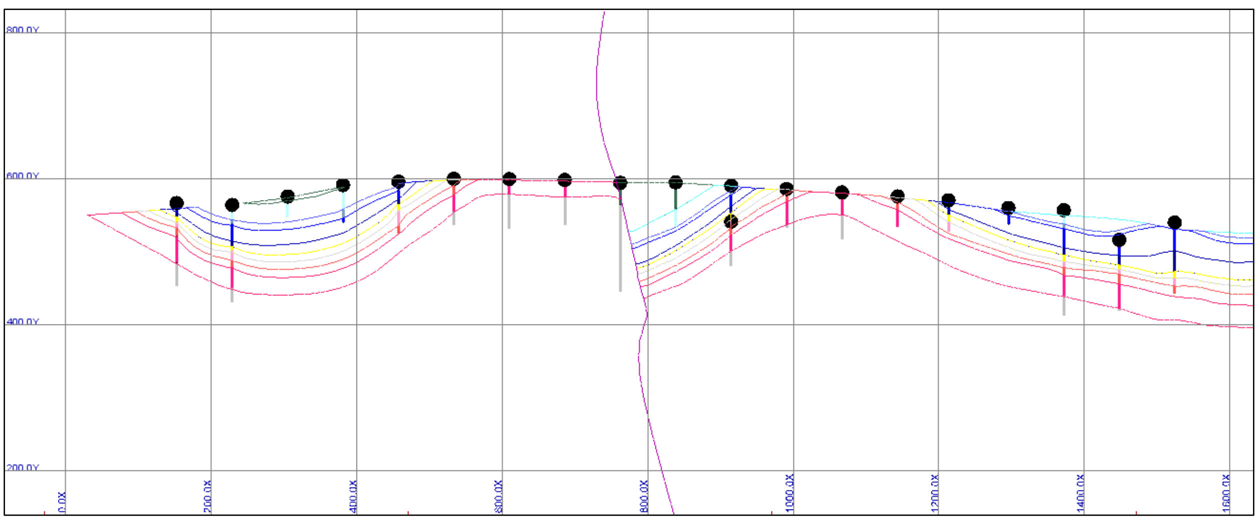


Figure 14.8: Section 6398N (Looking North) Showing Reverse Fault and Vertical Displacement of Geological Units



14.3.3 Structural Domains

Due to the folded nature of the Scully Deposit, the geological model was divided into multiple structural domains. A structural domain was modelled for each fold limb, where the orientation (dip and strike) of the geological units is comparable. Wireframes of the 14 structural domains are illustrated in Figure 14.9 (plan view) and Figure 14.10 (cross section view).

Table 14.3 lists the fold limb orientations of each structural domains. These orientations will be used as search ellipse orientations during grade estimation.

Figure 14.9: Plan View Showing the Structural Domain Definition

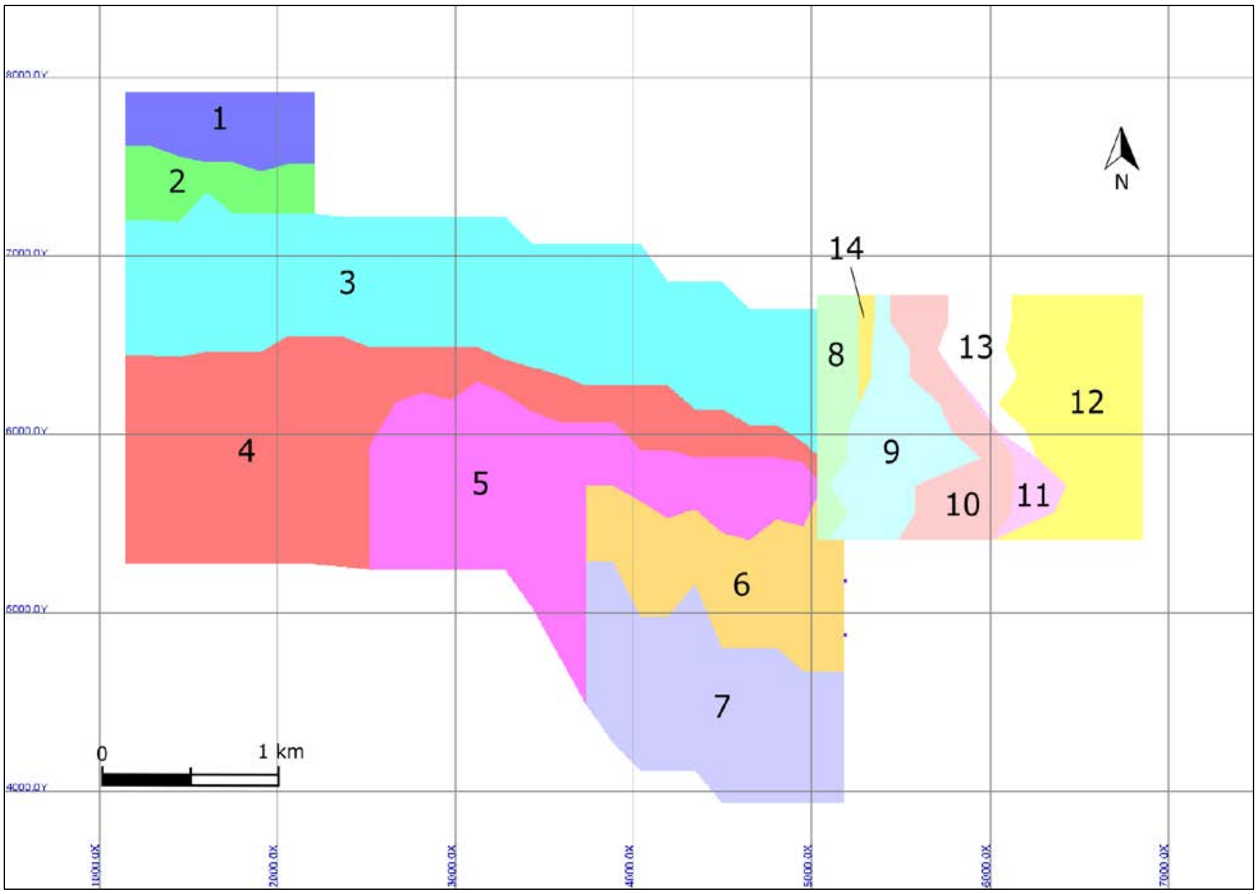


Figure 14.10: Section View (1829 mE) Looking East Showing the Structural Domains and the Geological Units

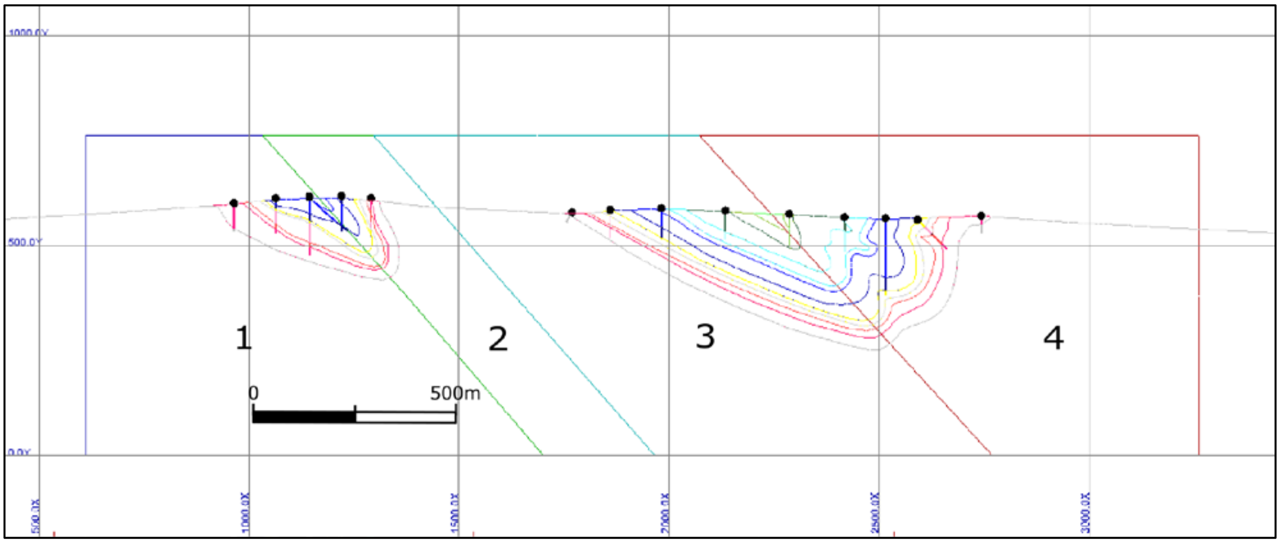


Table 14.3: List of Fold Limb Orientations Defining the Structural Domains

Domain	Plane Orientation	
	Dip Direction	Dip (negative = downwards)
1	180	-30
2	180	-65
3	185	-20
4	10	-50
5	190	0
6	190	-27
7	10	-30
8	90	-40
9	270	-25
10	80	0
11	65	-50
12	75	-20
13	250	-25
14	90	0

14.3.4 Overburden

A base of overburden surface (rock code 11) was constructed using logged drill hole intercepts from drilling conducted only from the original topographic surface. Several drill holes located within the pit contained the overburden code (11), which likely represented loose material affected by previous blasts (not true glacial till). These were filtered out, and only valid overburden intercepts were retained for modelling.

14.3.5 Topography Surface

As the geological units are constrained by the base of overburden surface, no topography was applied in the geological model. A topography will be applied and discussed when reporting the Mineral Resources later in this report.

14.4 Statistical Analysis

This section will focus mainly on the crude assay attributes which will be reported in the Mineral Resource (crude Fe, Mn, and Magnetite %). Weight recovery (WTREC) will also be assessed, as it is an important attribute for metallurgical calculations in the mineral reserve. However, this attribute is strongly influenced by the performance of the shaker table (and the crude Fe% grade) and Fe-mineralogy and should not be considered as a metallurgical recovery in the concentrator.

14.4.1 Assay Statistics

Group-wise averages of the crude assays and concentrate assays were computed using R statistical software. Assays were grouped by geological units as per the stratigraphic column shown in Section 7 and are presented in Table 14.4 and Table 14.5.

Table 14.4: Attribute Averages for Crude Assays

Stratigraphy	Rock Code	Average Downhole Thickness (m)*	Fe %	Mn %	Magnetite %	No. of Assays**
Upper Member	21	33.2	32.04	0.60	6.48	211
	22	26.5	35.33	0.69	5.40	320
	23	29.9	30.59	0.72	5.64	854
Middle Member	31	27.1	36.75	1.29	3.63	1237
	32	15.2	29.69	1.48	3.37	431
	33	25.2	38.41	1.68	5.06	1829
	34	19.0	36.77	1.60	6.78	1271
Middle Quartzite	41	12.8	23.30	2.53	3.59	623
Lower Member	51	14.6	35.66	3.06	5.37	981
	52	19.0	31.37	1.42	4.56	816
	53	27.0	36.01	3.03	5.54	1807
Basal Silicates	61	-	33.59	1.92	6.48	877

** Downhole thickness is not representative of true thickness*

***Number of assays derived from the CRFE attribute. There are 15% less samples analyzed for CRMAG compared to CRFE*

Table 14.5: Attribute Averages for Concentrate Assays

Stratigraphy	Rock Code	Average Downhole Thickness (m)*	WTREC	Fe Conc %	SiO_2 Conc %	Mn Conc %	Mag Conc %	No. of Assays**
Upper Member	21	33.2	21.96	66.19	2.93	0.71	14.65	190
	22	26.5	34.44	66.41	3.10	0.77	8.78	319
	23	29.9	20.28	66.69	3.35	0.89	11.75	758
Middle Member	31	27.1	44.85	65.96	3.16	1.29	3.27	1217
	32	15.2	27.28	64.35	3.58	1.88	2.21	406
	33	25.2	45.50	65.40	3.55	1.53	3.17	1811
	34	19.0	30.57	64.83	3.63	1.72	6.35	1212
Middle Quartzite	41	12.8	11.04	55.85	7.39	5.23	6.58	463
Lower Member	51	14.6	39.54	63.28	4.26	3.05	4.63	977
	52	19.0	26.72	64.83	3.86	1.66	3.65	780
	53	27.0	40.54	63.38	3.89	2.96	5.27	1787
Basal Silicates	61	-	16.82	61.72	4.37	2.70	15.19	704

**Downhole thickness is not representative of true thickness*

***Number of assays derived from the CONFE attribute. There are 62% less samples analyzed for CONMAG compared to CONFE.*

The averages confirm that the higher-quality iron formations are the rock codes 22, 31, 33, 34, 51, and 53, which present higher Fe grades and higher weight recoveries. The attribute that differentiates the three iron formations is manganese, which is generally low in the Upper Member (0.6% – 0.7%), moderate in the Middle Member (1.2% – 1.6%), and high in the Lower Member (1.4% - 3.1%). Weight recoveries generally correlate with crude Fe grade (higher crude grade, higher weight recovery), although this is not true for the Upper Member which is dominated by goethite and contains significant magnetite which has proven in the past problematic for recoveries.

Cumulative probability plots are shown for crude Fe% grades in Figure 14.11 separated by rock code. Due to the negatively skewed distributions associated with crude Fe% raw assays, no outliers were identified.

Probability plots were also generated for crude Mn% for each rock code with in the three Members and are presented in Figure 14.12. The plots show more typical positively skewed data (which often exhibit outliers). Potential outliers exist for crude Mn % in rock code 31 (Middle Member), however these will be readdressed after compositing.

Figure 14.11: Cumulative Probability Graphs for Crude Fe% Raw Assays Inside the Upper (Left), Middle (Right) and Lower (Bottom) Members

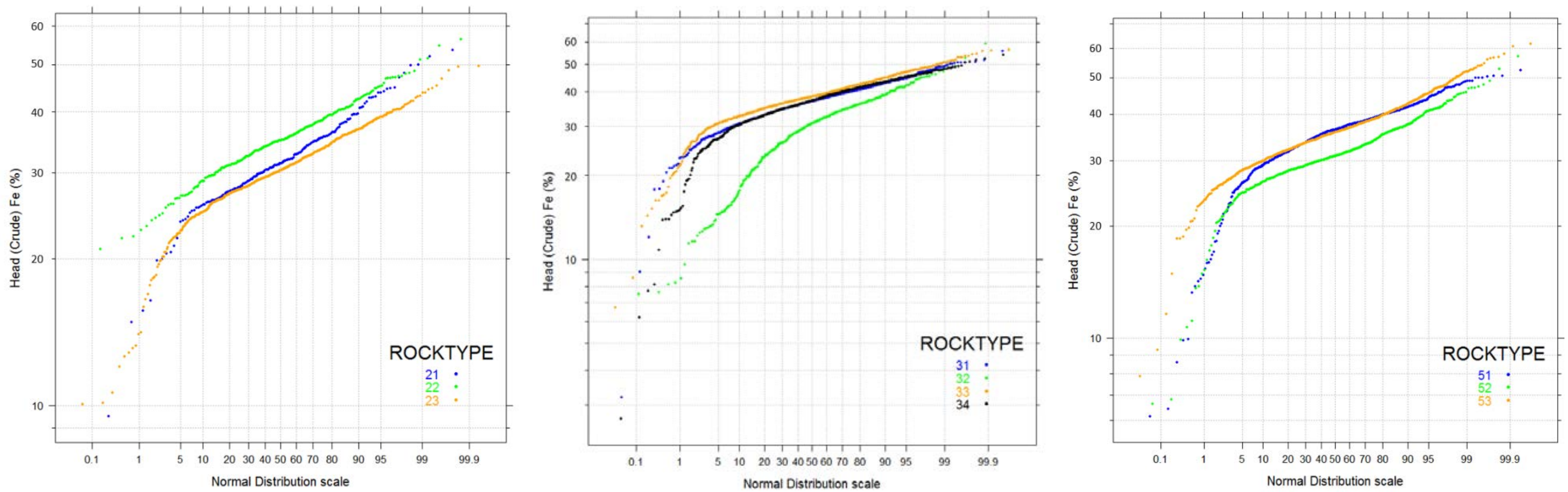
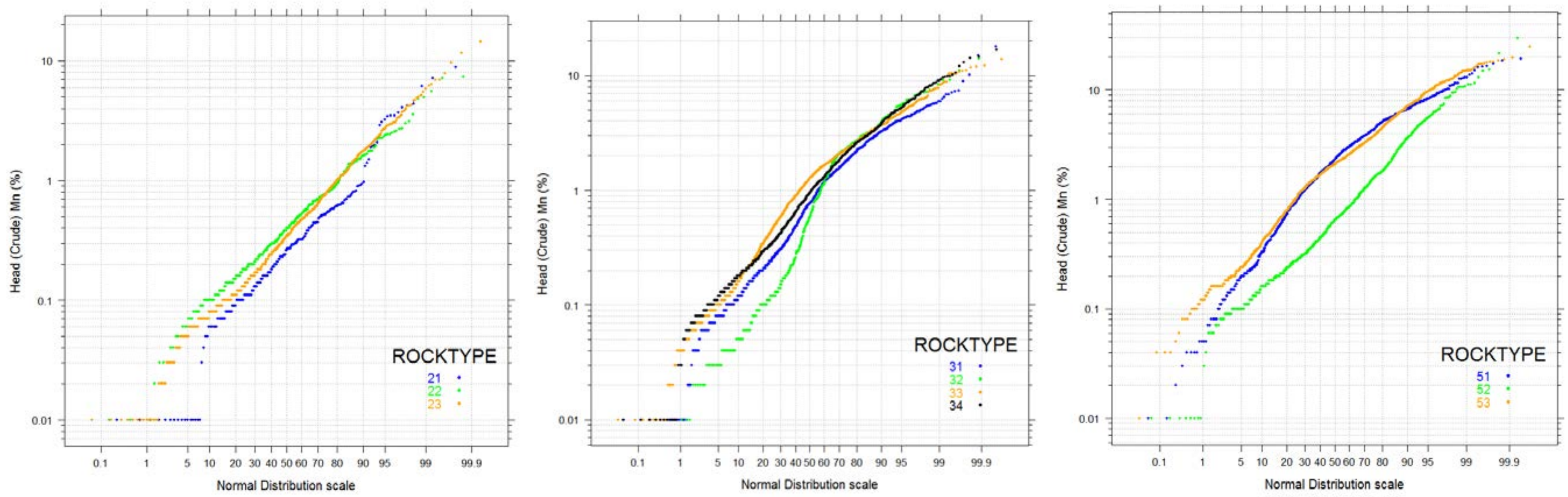


Figure 14.12: Cumulative Probability Graphs for Crude Mn% Raw Assays Inside the Upper (Left), Middle (Right) and Lower (Bottom) Members

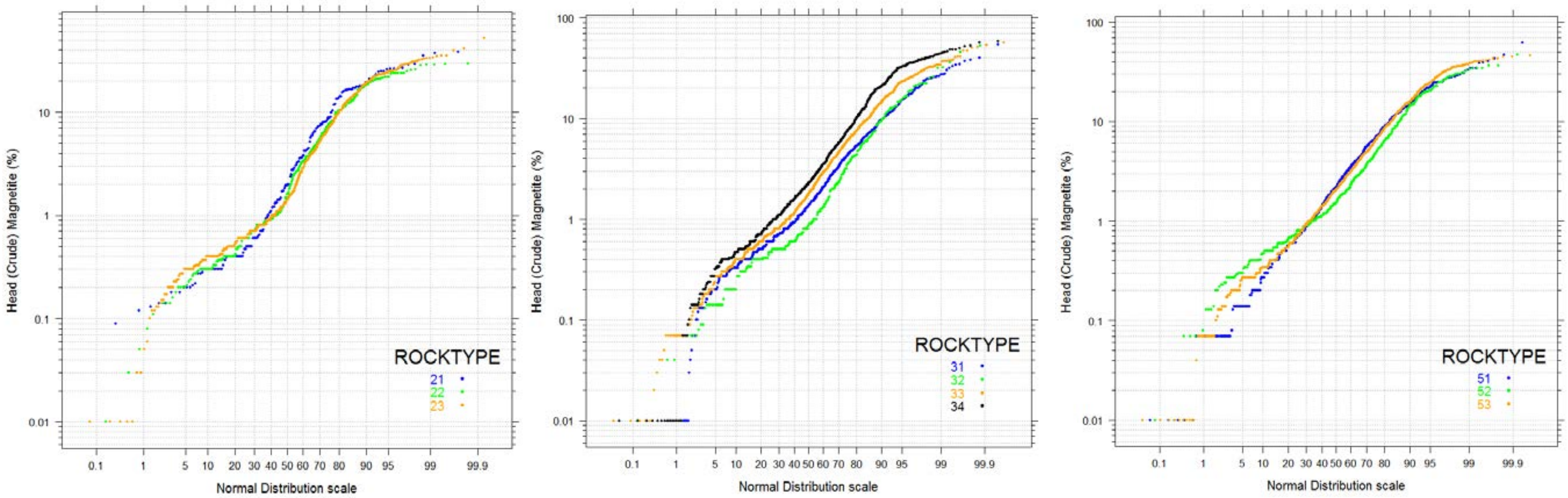


Finally, probability plots were also generated for crude Magnetite % for each rock code within the three (3) Members and are presented in Figure 14.13. The plots show a large spread of % magnetite with the Middle Member, with a potential sub-population present in rock code 33, 34 and 53. O'Leary (1972) describes magnetite increasing with depth in the Lower Member and the Middle Member, however, this relationship is not observed constantly across the deposit.

Spatial interrogation of crude Magnetite % reveals that magnetite is isolated to the western edge of the South Pit, and is generally present in rock codes 33, 34 and 53 (as described by O'Leary, 1974). As this concentration of high-magnetite reflects a separate spatial sub-domain, no capping will be applied to the crude Magnetite % attribute.

Probability plots were also examined for the concentrate attributes in the raw assays. Concentrate SiO_2% displayed potential outlier populations, and these will be addressed after compositing to evaluate if grade capping should be implemented. Fe% and Mn% concentrate grades did not display any significant outliers.

Figure 14.13: Cumulative Probability Graphs for Crude Magnetite % (Satmagan) Raw Assays Inside the Upper (Left), Middle (Right) and Lower (Bottom) Members



14.4.2 Compositing

As shown in Section 14.2.1, most assay lengths are between 3 and 12 m, reflecting quarter, half, and full bench heights (current bench heights are 40 ft in height). GMS considers a composite length of 6 m suitable to provide enough resolution within the block model for mine planning purposes.

Composites were subdivided by rock code with a maximum length of 6 m permitted, and end of interval composites less than 1 m were discarded. Statistical checks were made between the raw assays and the composites to ensure that the composites are representative of the original assays (i.e., total accumulations of length multiplied by crude Fe% were compared to ensure the loss of "metal" during compositing was minimal).

14.4.3 Exploratory Data Analysis

This section describes various statistical methods applied to the composite and blast hole databases to determine an appropriate grade estimation strategy, and to ensure that statistical populations in the database are well represented in the final block model.

Preliminary mine scheduling studies have shown that most of the future production tonnage will come from the Lower Member (units 51, 52 and 53), with the remaining tonnage provided by the Middle Member. Only minor tonnage will be sourced from the Upper Member. Therefore, the following section will focus mainly on the Middle and Lower Members of the stratigraphy.

Descriptive statistics of the 6.0 m composites generated for the crude attributes are summarized in Table 14.6, and the concentrate attributes are summarized in Table 14.7.

Table 14.6: Descriptive Statistics of Crude Attributes – 6 m Composites

Attribute	Stratigraphy	Rock Code	Mean	Median	St Dev	Variance	Coeff. of Var.	# of Obs.
Crude Fe %	Upper Member	21	31.39	31.24	6.92	47.91	0.22	264
		22	34.69	34.69	4.97	24.66	0.14	421
		23	31.20	30.92	4.70	22.09	0.15	1088
	Middle Member	31	36.61	36.80	4.62	21.38	0.13	1408
		32	30.42	31.79	8.30	68.85	0.27	555
		33	38.51	38.37	4.74	22.44	0.12	1920
		34	36.47	36.90	5.44	29.56	0.15	1389
	Middle Quartzite	41	24.53	26.53	10.50	110.32	0.43	853
	Lower Member	51	35.48	36.30	5.13	26.28	0.14	902
		52	31.13	30.72	4.61	21.24	0.15	916
		53	35.75	35.65	4.56	20.79	0.13	1836
Crude Mn %	Upper Member	21	0.55	0.25	1.08	1.16	1.97	256
		22	0.68	0.42	0.86	0.75	1.26	421
		23	0.77	0.36	1.21	1.46	1.58	1084
	Middle Member	31	1.28	0.78	1.42	2.03	1.11	1397
		32	1.75	1.11	1.98	3.92	1.13	551
		33	1.64	1.35	1.46	2.14	0.89	1917
		34	1.59	1.07	1.72	2.96	1.08	1389
	Middle Quartzite	41	2.82	1.94	2.65	7.04	0.94	853
	Lower Member	51	2.92	2.33	2.48	6.17	0.85	901
		52	1.32	0.64	1.97	3.88	1.49	913
		53	2.95	2.14	2.72	7.38	0.92	1835
	Upper Member	21	6.27	1.71	8.42	70.86	1.34	242
		22	6.11	1.99	7.68	58.97	1.26	383

Crude Magnetite % (Satmagan)		23	5.25	1.37	7.76	60.26	1.48	979
	Middle Member	31	3.76	1.50	5.53	30.55	1.47	1305
		32	3.77	1.39	6.05	36.61	1.61	510
		33	5.04	2.02	7.20	51.89	1.43	1800
		34	7.37	2.80	10.47	109.62	1.42	1288
	Middle Quartzite	41	4.62	1.45	8.96	80.22	1.94	680
	Lower Member	51	6.25	3.12	8.07	65.06	1.29	803
		52	4.85	1.79	7.02	49.32	1.45	745
		53	6.31	2.80	8.40	70.49	1.33	1472

Table 14.7: Descriptive Statistics of Concentrate Attributes – 6 m Composites

Attribute	Stratigraphy	Rock Code	Mean	Median	St Dev	Variance	Coeff. of Var.	# of Obs.
Weight Recovery %	Upper Member	21	23.87	23.70	13.29	176.54	0.56	262
		22	34.95	35.88	13.08	171.06	0.37	419
		23	22.38	20.87	12.22	149.29	0.55	1085
	Middle Member	31	44.28	45.29	10.75	115.63	0.24	1407
		32	30.68	29.06	14.07	198.00	0.46	551
		33	46.19	47.04	10.87	118.20	0.24	1911
		34	30.45	29.45	13.91	193.50	0.46	1385
	Middle Quartzite	41	13.30	9.20	12.36	152.86	0.93	834
	Lower Member	51	39.08	40.18	12.37	153.06	0.32	902
		52	27.68	27.61	9.72	94.58	0.35	915
		53	39.96	40.33	11.81	139.49	0.30	1829
Concentrate Fe %	Upper Member	21	65.86	67.03	3.26	10.66	0.05	254
		22	66.25	66.90	3.24	10.52	0.05	416
		23	66.54	66.91	8.64	74.59	0.13	1028
	Middle Member	31	65.95	66.50	2.37	5.62	0.04	1393
		32	64.36	65.42	4.55	20.67	0.07	532
		33	65.64	66.00	2.15	4.62	0.03	1905
		34	65.13	66.16	4.07	16.57	0.06	1359
	Middle Quartzite	41	56.28	61.27	13.70	187.67	0.24	702
	Lower Member	51	63.70	64.54	4.07	16.56	0.06	899
		52	65.36	66.33	3.71	13.77	0.06	899
		53	63.55	64.40	3.81	14.53	0.06	1823
Concentrate Mn %	Upper Member	21	0.61	0.29	1.23	1.51	2.00	254
		22	0.77	0.39	1.06	1.13	1.38	416
		23	0.87	0.36	1.54	2.36	1.76	1028

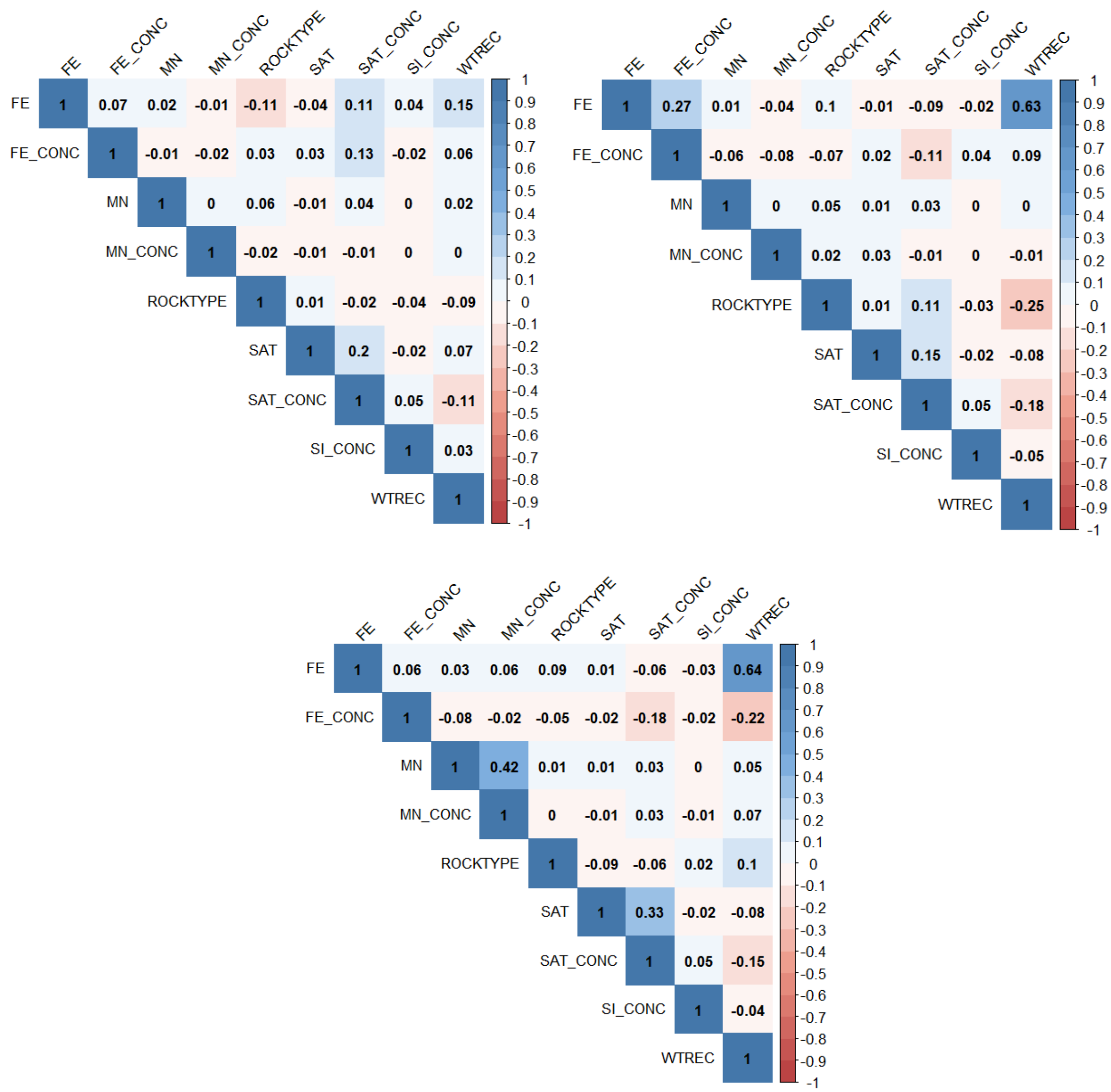
Attribute	Stratigraphy	Rock Code	Mean	Median	St Dev	Variance	Coeff. of Var.	# of Obs.
	Middle Member	31	1.28	0.72	1.58	2.51	1.24	1391
		32	2.16	1.06	2.73	7.45	1.26	530
		33	1.48	1.04	1.48	2.19	1.00	1905
		34	1.67	0.96	2.14	4.59	1.29	1358
	Middle Quartzite	41	5.24	2.99	5.98	35.74	1.14	687
	Lower Member	51	2.89	2.08	2.65	7.01	0.92	899
		52	1.45	0.61	2.49	6.20	1.72	898
		53	2.92	2.22	2.55	6.51	0.87	1822
Concentrate SiO_2 %	Upper Member	21	3.51	2.97	2.56	6.53	0.73	254
		22	3.37	2.89	3.25	10.57	0.97	415
		23	3.12	2.72	2.61	6.80	0.84	1023
	Middle Member	31	3.17	2.90	1.59	2.54	0.50	1392
		32	3.50	3.05	2.19	4.78	0.62	532
		33	3.38	3.11	1.63	2.66	0.48	1893
		34	3.36	2.84	2.59	6.70	0.77	1352
	Middle Quartzite	41	6.39	4.26	6.96	48.43	1.09	683
	Lower Member	51	3.91	3.18	3.42	11.72	0.88	895
		52	3.48	3.11	2.08	4.32	0.60	894
		53	3.73	3.29	2.80	7.84	0.75	1820
Concentrate Magnetite % (Satmagan)	Upper Member	21	10.21	2.70	15.80	249.60	1.55	102
		22	9.78	3.36	14.10	198.89	1.44	174
		23	9.03	1.80	18.02	324.63	2.00	396
	Middle Member	31	4.92	1.59	9.44	89.17	1.92	295
		32	2.93	0.85	5.03	25.33	1.72	216
		33	3.36	1.68	4.30	18.50	1.28	552
		34	7.76	4.19	10.12	102.37	1.30	504

Attribute	Stratigraphy	Rock Code	Mean	Median	St Dev	Variance	Coeff. of Var.	# of Obs.
	Middle Quartzite	41	5.83	3.43	7.28	52.99	1.25	230
	Lower Member	51	5.98	2.60	8.15	66.47	1.36	366
		52	3.89	2.00	5.75	33.02	1.48	317
		53	7.04	3.17	11.85	140.45	1.68	614

The composite statistics confirm the remarks made in Section 14.4.1. In addition, it appears that historical sampling practices have introduced a possible bias in relation to preferential sampling of rock codes 31, 33 and 34, which were considered higher quality mineralization compared to other rock codes. Rock codes 32, 51 and 52 are under-represented in the composites, however, this may be due to the thinner nature of these units compared to the major ore units. The Upper Member (rock code 21, 22, 23) does not contribute largely to the tonnage of remaining in-pit resources at Scully.

To investigate potential relationships between the crude and concentrate attributes in the composites, correlation plots were produced by grouping rock codes into each Member (Upper, Middle and Lower). Results are shown in Figure 14.14.

Figure 14.14: Correlation Plots of the Upper (Left), Middle (Right) and Lower (Bottom) Members

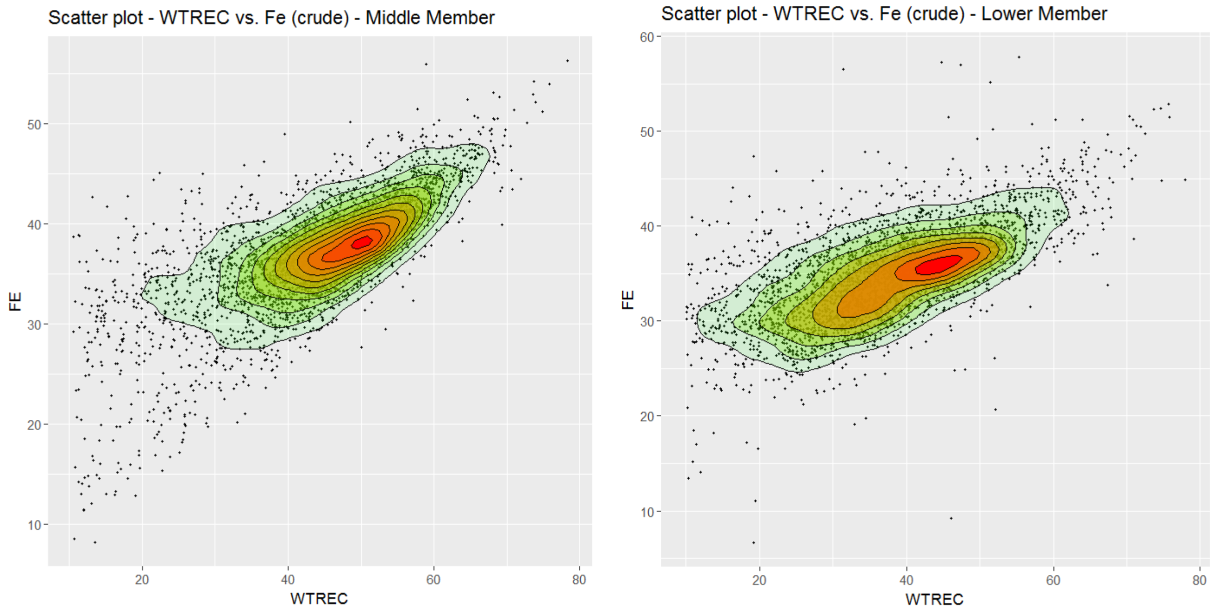


The key differences between the correlation plots are that there is only a very weak correlation between weight recovery (WTREC) and crude Fe% in the Upper Member, compared to a much stronger correlation for the Middle and Lower Member. We believe that this could be due to the different mineralogy and rock textures, with the Upper Member dominated by goethite, and banded (rather than massive) resulting in lower recoveries.

Weak correlations exist between Mn% and SAT% and their concentrate counterparts, indicating that magnetite and manganese levels have been maintained after shaker table treatment.

Scatter plots comparing WTREC and crude Fe% are presented in Figure 14.15 for the Middle and Lower Members, showing the strengthening correlation as crude Fe% increases.

Figure 14.15: Scatter Plots Comparing WTREC and Fe% (Crude) For the Middle Member (Left) and Lower Member (Right). Colours and Contours Reflect Point Density



14.4.4 <u>Blasthole vs. Composite Comparison – Q:Q Plots</u>

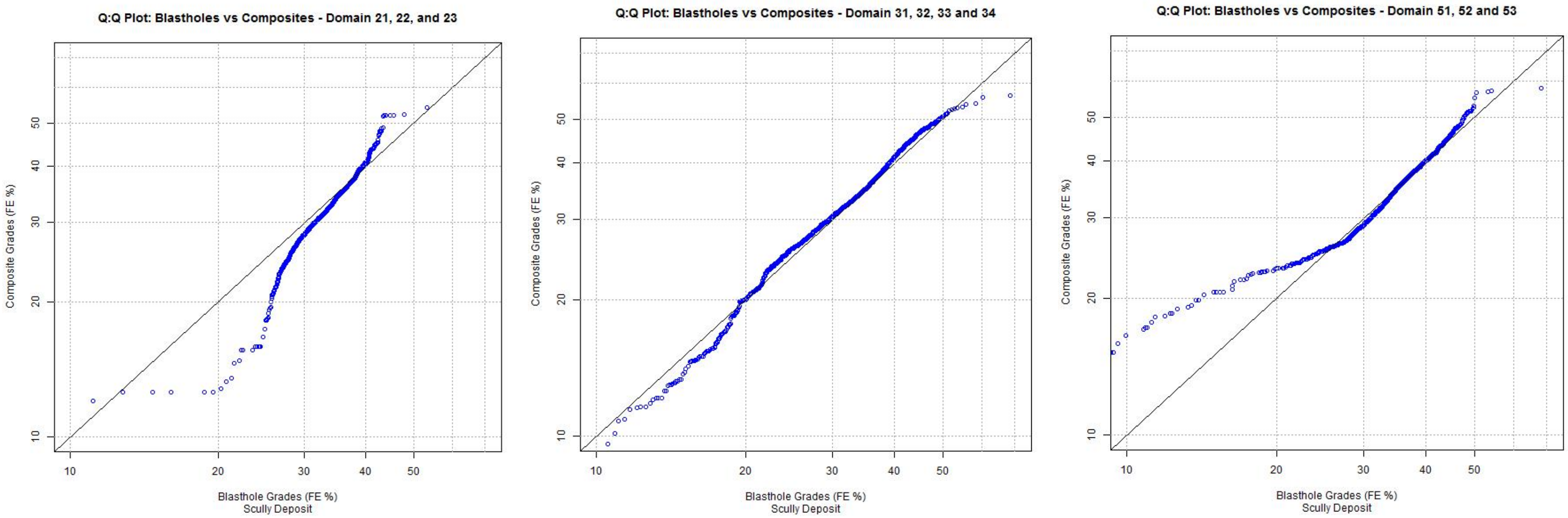
To enable the global comparison between the blast hole database and the exploration drilling database, blast holes were flagged using the wireframes constructed in Section 14.3.1. Q:Q plots were then constructed comparing the distributions of the two (2) datasets to confirm that the composites are globally representative of the blasthole database.

Q:Q plots are useful in comparing the *distributions* of two (2) datasets, and can identify any differences in bias (i.e., a shift in the mean), skewness (asymmetry of the distributions) or kurtosis (i.e., the “tightness” of the distribution in relation to any outliers). Two (2) identical distributions will follow the center diagonal line. The slope of the Q:Q plot curve reflects the difference in kurtosis of the two distributions.

As the wireframes were modelled based on the diamond drill holes only, there could be local variations in grade between the blast holes and drill holes, however, when considered as the three (3) main Members (Upper, Middle and Lower) these differences should be insignificant.

Figure 14.16 presents Q:Q plots for crude Fe% comparing blast holes and exploration drill holes for the 3 main Members (Upper, Middle and Lower).

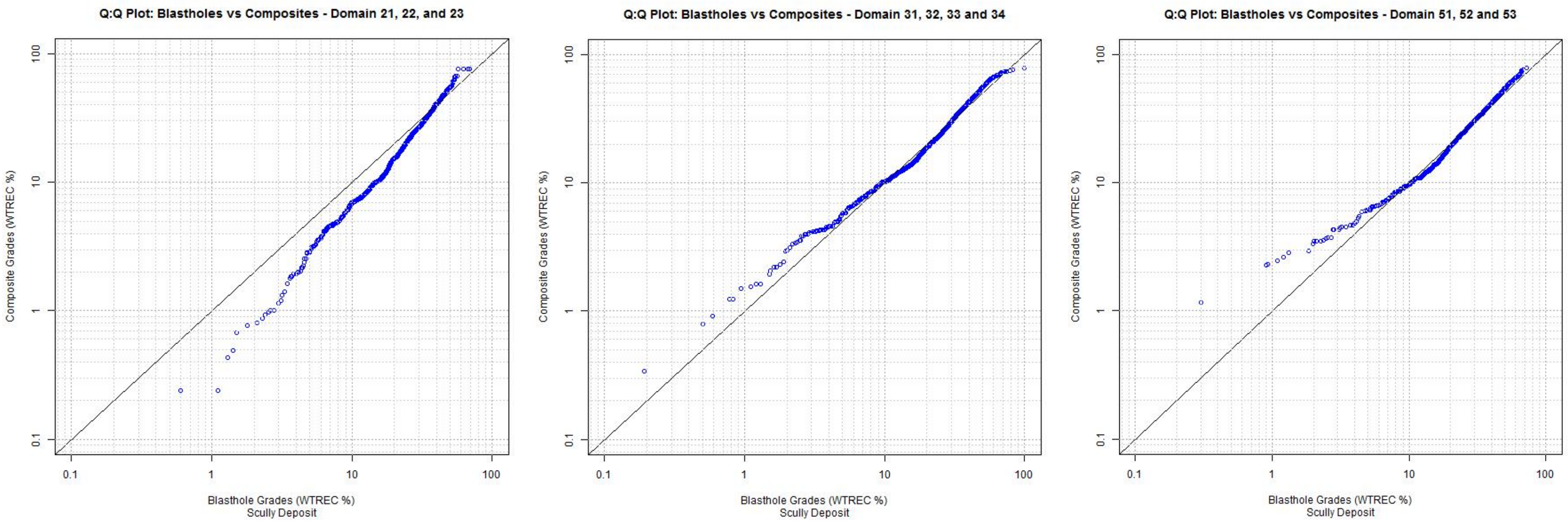
Figure 14.16: Q:Q Plots for Crude Fe % Comparing Blastholes (X Axis) and Exploration Drill Holes (Y-Axis) for the Upper (Left), Middle (Right) and Lower (Bottom) Members



The Q:Q plot comparisons show that the blasthole database shows lower variability for crude Fe % than the exploration drill holes for the Upper Member. For the Middle Member, the distributions are near identical, indicating that the blasthole and exploration drill hole database are very similar regarding their distributions of crude Fe%. Finally, the distributions of crude Fe% are similar for the Lower Member, apart from the blasthole database contains a population of samples assaying less than 25% Fe compared to the exploration drill holes. These samples could belong to the Middle Quartzite or the Basal Silicates (above and below of the Lower Member respectively) and are likely the result of localized differences between the geological wireframes, and the true geology.

A key attribute for determining the final tonnage of concentrate during mine planning will be the weight recovery attribute ("WTREC"). Therefore, it is important to analyse the distributions of the blast holes and exploration drill holes for this attribute to ensure there is no bias observed, and that WTREC can be confidently incorporated into the block model. Q:Q plots for WTREC are shown in Figure 14.17.

Figure 14.17: Q:Q Plots for Weight Recovery % (WTREC) Comparing Blastholes (X Axis) and Exploration Drill Holes (Y-Axis) for the Upper (Left), Middle (Right) and Lower (Bottom) Members



Q:Q plots of weight recovery % (WTREC) show that the blast holes and exploration drill holes show very similar distributions for the Middle and Lower Members. For the Upper Member, a bias is present where weight recoveries are generally higher in the blast holes than the exploration drill holes.

Q:Q plots were also reviewed for crude Mn %, and were found to be satisfactory, with the blasthole database a good representation of the exploration drilling database. Q:Q plots of concentrate grades were also reviewed, and no significant bias was found between the blast holes and exploration drilling for the Middle and Lower Members.

No blasthole comparisons were made for Magnetite %, as the blastholes were rejected due to data integrity concerns.

14.4.5 Grade Capping

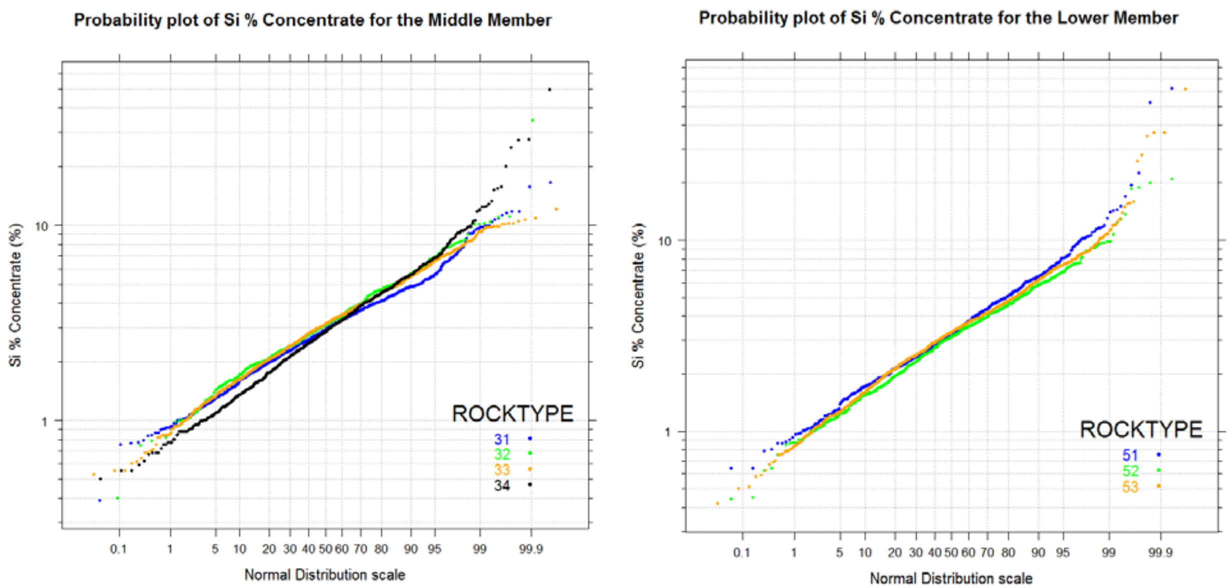
Section 14.2.1 concluded grape capping would be revisited after compositing, to evaluate the impact of compositing on the grade attributes. Probability plots of the composites were re-assessed, and grade capping was chosen and summarized in Table 14.8.

Table 14.8: Grade Capping Applied to Composites for Grade Estimation

Rock Code	Crude Fe %	Crude Mn %	Crude Magnetite %	Weight Recovery %	Fe % Con.	SiO_2 % Con.	Mn % Con.	Magnetite % Con.
21	-	-	-	-	-	-	-	-
22	-	-	-	-	-	10	-	-
23	-	-	-	-	-	-	-	-
31	-	6	-	-	-	-	-	-
32	-	-	-	-	-	15	-	-
33	-	-	-	-	-	-	-	-
34	-	-	-	-	-	20	-	-
51	-	-	-	-	-	20	-	-
52	-	-	-	-	-	-	-	-
53	-	-	-	-	-	20	-	-

SiO_2% concentrate grades show significant outliers in many rock codes. Most values for SiO_2% concentrate lie between 0% and 10%, however outliers occur up to 50% SiO_2. For this reason, SiO_2% concentrate grades were capped to between 10% and 20% based on probability plots of individual rock codes. These plots are presented in Figure 14.18.

Figure 14.18: Probability Plots of Sio$_2$% Concentrate by Rock Code for Outlier Identification



14.4.6 <u>Variography</u>

A 3D directional variography was conducted on the 6 m composites, grouped into their respective Member (Upper, Middle and Lower). The complex folding of stratigraphy at Scully means producing representative variograms is difficult without applying complex unfolding algorithms on the composites. An alternative strategy was adopted, where composites located in similar structural domains (structural domains were grouped based on orientation of bedding) were grouped, and then grouped again into their respective Member (Upper, Middle and Lower). This created variographic domains with sufficient data to derive representative experimental variograms.

Structural domains 1, 3 and 6 were grouped (Figure 14.19), as they share a common dip and strike of bedding (shallow dip to the south). Variographic domains for the Lower, Middle and Upper Members contained 641, 2225, 1375 composites respectively.

Directional semi-variograms were generated in Geovia GEMS™ for crude Fe%, Mn% and Magnetite % within the previously described variographic domain. Semi-variograms were produced every 5 degrees azimuth and 5 degrees dip increments. The spread angle was set to 30 degrees, with a bandwidth of 200 m. A lag distance of 75 m was applied. The manually-fitted variogram models included a nugget effect and two spherical structures.

The parameters of the semi-variograms modelled are summarized in Table 14.9. Table 14.10 presents the rotation angles defining the anisotropy for each combination of attribute and Member.

Table 14.9: Variogram Parameters for Crude Attributes in the Variographic Domain

Variable	Member	C_0	First Spherical Structure				Second Spherical Structure			
			Sill	Ranges (m)			Sill	Ranges (m)		
				Major	Semi-Major	Minor		Major	Semi-Major	Minor
Crude Fe%	Upper	0.12	0.16	130	98	43	0.28	425	320	141
	Middle	0.02	0.30	114	75	36	0.25	487	320	154
	Lower	0.15	0.16	129	79	26	0.21	504	306	102
Crude Mn%	Upper	0.34	0.34	97	81	29	0.96	540	447	164
	Middle	0.43	0.47	107	75	35	0.70	865	612	153
	Lower	0.56	1.62	126	126	55	0.58	435	435	191
Crude Magnetite %	Upper	7.6	19.9	169	169	79	29.2	370	370	172
	Middle	17.1	30.5	122	122	92	16.0	414	414	147
	Lower	7.0	24.0	93	87	93	9.4	481	450	137

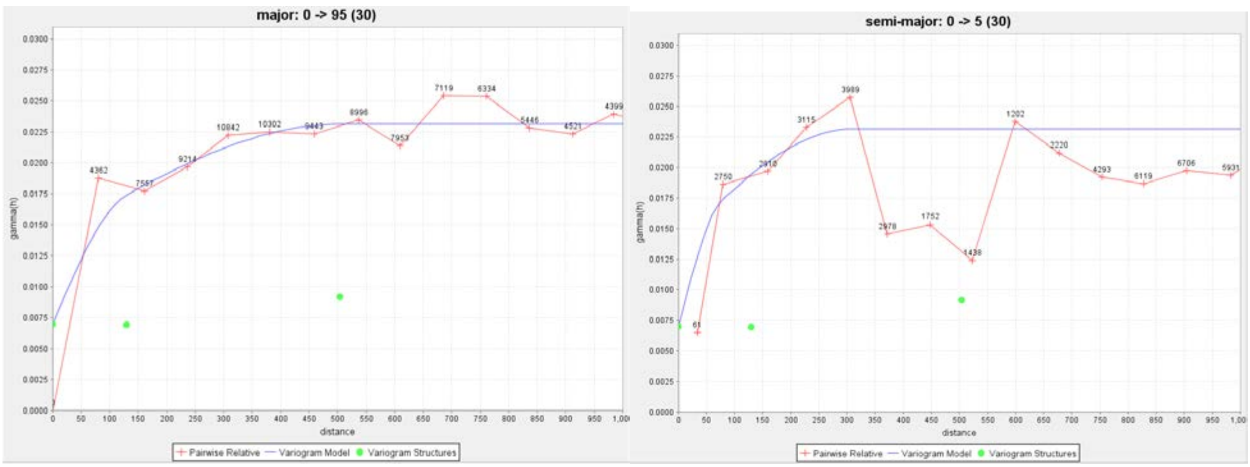
Table 14.10: Rotation Angles Defining the Major and Semi-Major Axes

Structural Domain	Rotation				
	Member	Major Azi	Major Plunge	Semi-Major Dip	Semi-Major Azi
Crude Fe %	Upper	080°	0°	-20°	170°
	Middle	100°	0°	-20°	190°
	Lower	095°	0°	0°	005°
Crude Mn %	Upper	105°	0°	-20°	195°
	Middle	100°	0°	-20°	190°
	Lower	105°	0°	-20°	195°
Crude Magnetite %	Upper	100°	0°	-20°	190°
	Middle	100°	0°	-20°	190°
	Lower	100°	0°	-20°	190°

Concentrate grades are strongly influenced by factors such as the performance of the recovery equipment (shaker tables), crushing sizes and other non-geological parameters. For this reason, GMS believes that undertaking variography on these attributes is unnecessary.

An example of the variogram model for crude Fe % for the Lower Member is given in Figure 14.19.

Figure 14.19: Example of Variogram Model for Crude Fe % for the Lower Member. Major Axis (Left) and Semi-Major Axis (Right)



14.5 Block Modelling

The 2021 block model for the Scully Deposit was created in Geovia® GEMS 6.7.4 environment.

14.5.1 Block Model Parameters

The drilling pattern, the thickness of the zones, the complexity of the geological model and the open pit mine planning considerations guided the choice of block dimensions. The block model parameters are summarized in Table 14.11.

Table 14.11: Block Model Settings (SC21_FINAL)

Axis	Origin[(1)] (m)	Block Size (m)	Number of Blocks	
X	917	20	352	Columns
Y	3654	20	240	Rows
Z	804	12	85	Levels

[1]. The origin is at the minimum X, minimum Y, and maximum Z. No rotation was applied

A series of attributes required during the block modelling development were incorporated into the block model project. Table 14.12 presents the list of attributes found in the block model SC21_FINAL in the folder "WA".

Table 14.12: List of Attributes Found in The Block Model SC21_FINAL

Origin of Attribute	Model Name (units)	Description
Geology and Structural Models	Rock Type	Lithology codification (11, 21, 22, 23, 31, 32, 33, 34, 41, 51, 52, 53, 61)
	STRUCT	Structural domain codification (100 to 1400)
	STROCK	Litho-structural domain codification (Rock Type, Structural Domain)
Grade Variables	DENS_WA (g/cm^3)	Weighted Average specific gravity
	FE_WA (%)	Weighted Average estimated grade for Crude Fe %
	MN_WA (%)	Weighted Average estimated grade for Crude Mn %
	SAT_WA (%)	Weighted Average estimated grade for Crude Magnetite %
	WTREC (%)	Weighted Average estimated grade for Weight Recovery
	FE_CONC_WA (%)	Weighted Average estimated grade for Concentrate Fe %
	SI_CONC_WA (%)	Weighted Average estimated grade for Concentrate Si %
	MN_CONC_WA (%)	Weighted Average estimated grade for Concentrate Mn %
	SAT_CONC_WA (%)	Weighted Average estimated grade for Concentrate Magnetite %
Mineral Resource Classification	CLASS	Iron recovery from Heavy Liquid Separation (HLS) composites, estimation by Ordinary Kriging
Kriging Performance Indicators	PASS_WA	Kriging Pass for Crude Fe % (1, 2, 3 ,4 or 5)
	SAMPLE_WA	Number of Samples used during estimation of Fe %
	SLOPE_WA	Slope of Regression for Fe %
Reporting	INPIT	Blocks used to report the mineral resource (inside pit optimisation)

14.5.2 Rock Type Model and Estimation Domains

Rock type models relied on a combination of geological and structural wireframes as presented in Section 14.3. Due to the folded nature of the deposit, and narrow width of some geological units (compared to the bench height), a block-percentage style model was adopted. This method will ensure a more representative grade is estimated within blocks shared by multiple geological units and is an alternative to sub-blocking. Once a block pertaining to numerous rock codes is estimated, the various grades and percentages within that block will be used to calculate a single tonnage-weighted average grade for each block.

Checks were made between the volumes of the wireframes, and the resulting volumes of the blocks to ensure that the rock codes within the block model were representative.

Estimation domains were created by concatenating the structural domains (1 - 14) with the geological rock codes of the iron formations (21, 22, 23, 31, 32, 33, 34, 41, 51, 52, 53) to produce individual estimation domains (attribute named STROCK in the block model).

Views of the Rock Type Domain attribute are illustrated in Figure 14.20 and Figure 14.21.

Figure 14.20: Cross-Section 1677mE Showing the Dominant Rock Type - Looking East

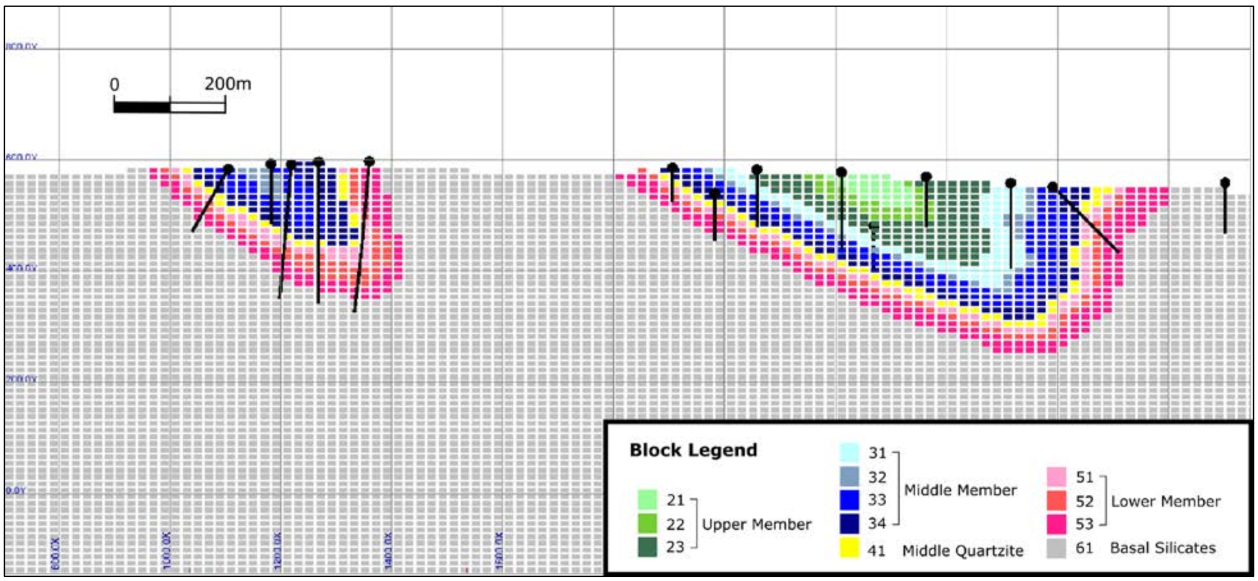
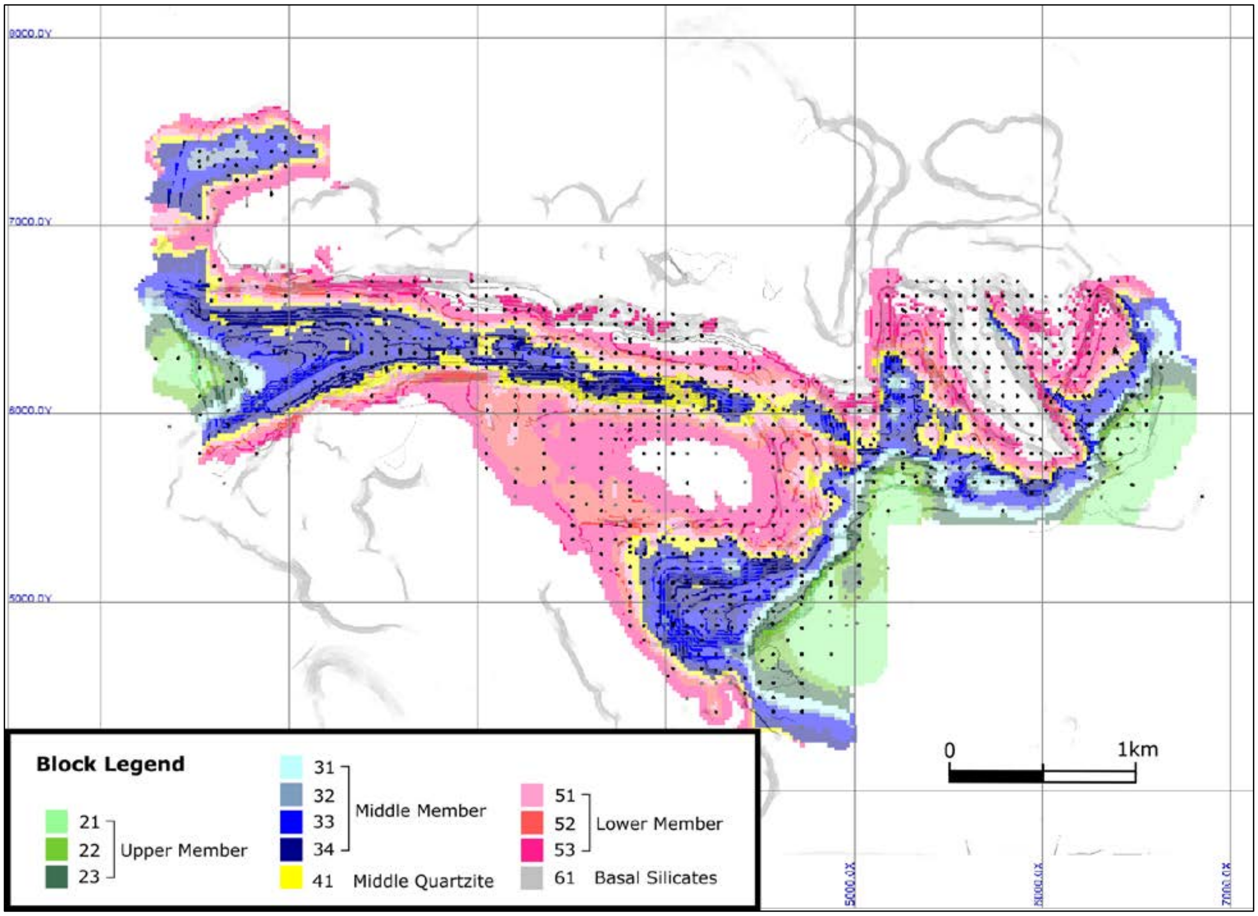


Figure 14.21: Plan View of the Base of the Current Pit Showing the Rock Type Attribute (Overburden Blocks are Hidden)



14.5.3 <u>Density Model</u>

As no existing bulk density measurements were available for the Scully Deposit, GMS used a regression formula derived from typical Sokoman Formation rocks in the Labrador Trough iron ore district to assign bulk densities to the block model. These values were then validated by laboratory measurements taken as part of the re-sampling program described in Section 11.

The regression curve for bulk density is based on the crude Fe % values in the block model and an adjustment factor (0.9) to account for the more friable nature of the mineralization at Scully. The equation is presented below:

$$Bulk\ Density = (0.0002Fe^2 + 0.0157Fe + 2.758) \times 0.9$$

Any resulting density values less than 2.7 g/cc were adjusted upwards to a value of 2.7 g/cc. Blocks pertaining to overburden (rock code 11) were assigned a density of 2.2 g/c, and blocks pertaining to the Middle Quartzite and the Basal Silicates (rock codes 41 and 61 respectively) were assigned a rock code of 2.7 g/cc.

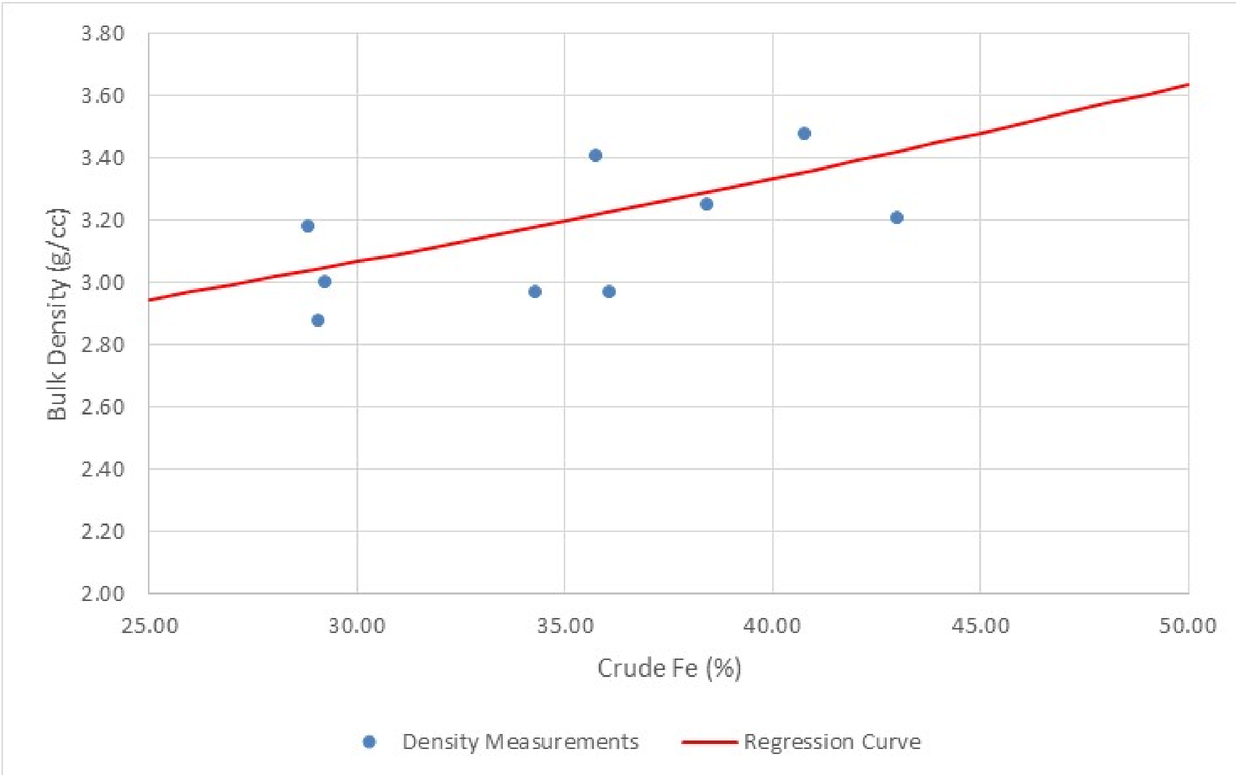
Density values presented in Table 14.13 are average values for each rock type within the optimized pit shell using a cut-off of 20% Fe.

Table 14.13: Resulting Density Values by Rock Code

Member	Rock Code	Density (g/cm^3)
Overburden	11	2.20
Upper Member	21	3.08
	22	3.16
	23	3.12
Middle Member	31	3.19
	32	3.15
	33	3.25
	34	3.16
Middle Quartzite	41	2.70
Lower Member	51	3.11
	52	3.12
	53	3.19
Basal Silicates	61	2.70

The regression curve was subsequently compared to limited laboratory measurements undertaken as part of the re-sampling program. The results are shown in Figure 14.22, and GMS concludes that the density regression equation used in the block model is an acceptable representation of the laboratory measurements.

Figure 14.22: Bulk Density Measurements from Re-Assay Program Vs. Crude Fe (%) Compared with Regression Curve Applied in Block Model



14.5.4 <u>Estimation Methodology</u>

Grade estimates were carried out using a combination of Ordinary Kriging (OK) and Inverse Distance Square (ID2). A four-pass estimation methodology was adopted for the following attributes: crude Fe (%), crude Mn (%), crude SAT (%), weight recovery (%), concentrate Fe (%), concentrate Mn (%), concentrate SiO_2 (%), and concentrate SAT (%).

Geological boundaries were considered as hard boundaries during grade estimation. Structural domains were considered as soft boundaries, to allow composites (within the same rock code) to be shared between the different fold limbs of the Scully Deposit. Geovia® GEMS 6.7.4 software was used to produce the grade estimates.

Ordinary Kriging (OK) was used in the estimation of crude Fe%, Mn% and SAT% for the Upper, Middle and Lower Members (rock codes 21, 22, 23, 31, 32, 33, 34, 51, 52 and 53). The kriging strategy is summarized below:

- **First Pass**: A minimum of 9 and a maximum of 18 composites within the Pass 1 search ellipse ranges, using a combination of the exploration drill hole composites and the blast hole composites. Blast holes were used in the estimation of Fe% and Mn%, and were not used to estimate SAT% (due to their poor representation of exploration drill hole grades).

- **Second Pass**: A minimum of six (6) and a maximum of 18 composites within the Pass 2 search ellipse ranges. Only blocks which were not estimated during the first pass could be estimated during the second pass. Only exploration drill hole composites were used for this pass.

- **Third Pass**: A minimum of three (3) and a maximum of 10 composites within the Pass 3 search ellipse ranges. Only blocks which were not estimated during the first and second pass could be estimated during the third pass. Only exploration drill hole composites were used for this pass

- **Fourth Pass**: A minimum of one (1) and a maximum of 10 composites within the Pass 4 search ellipse ranges. Only blocks which were not estimated during the first, second or third pass could be estimated during the fourth pass. Only exploration drill hole composites were used for this pass.

Inverse Distance Squared (ID2) approach used to estimate weight recovery % (WTREC) and concentrate grades (Fe%, Mn%, SiO_2 and SAT%) within all rock codes apart from the Overburden (11) and Basal Silicates (61). The estimation strategy for these attributes is summarized below:

- **Single Pass**: A minimum of one (1) and a maximum of 18 composites, using the Pass 4 search ellipse ranges. Only exploration drill hole composites were used in the estimation of these attributes.

Finally, any remaining un-estimated blocks were assigned the average grade of the rock type for each grade variable (Pass 5).

Table 14.14 details search ellipse orientations by structural domain (structural domains were converted to 100 – 1400 to enable the concatenation of the estimation domains). Ranges and orientations of the ellipses were determined from the orientation of structural domains (fold limbs), and the ranges observed in the variograms. An example is shown in Figure 14.23, and search ellipses ranges are shown in Table 14.15.

Grade capping was applied for concentrate SiO_2 and crude Mn % attributes, as summarized in Table 14.8. No high-grade transition limits, or high-grade thresholds were applied during estimation.

Figure 14.23: Section 3197mE - Example of Pass 1 Search Ellipse Orientations and Structural Domains – Looking East

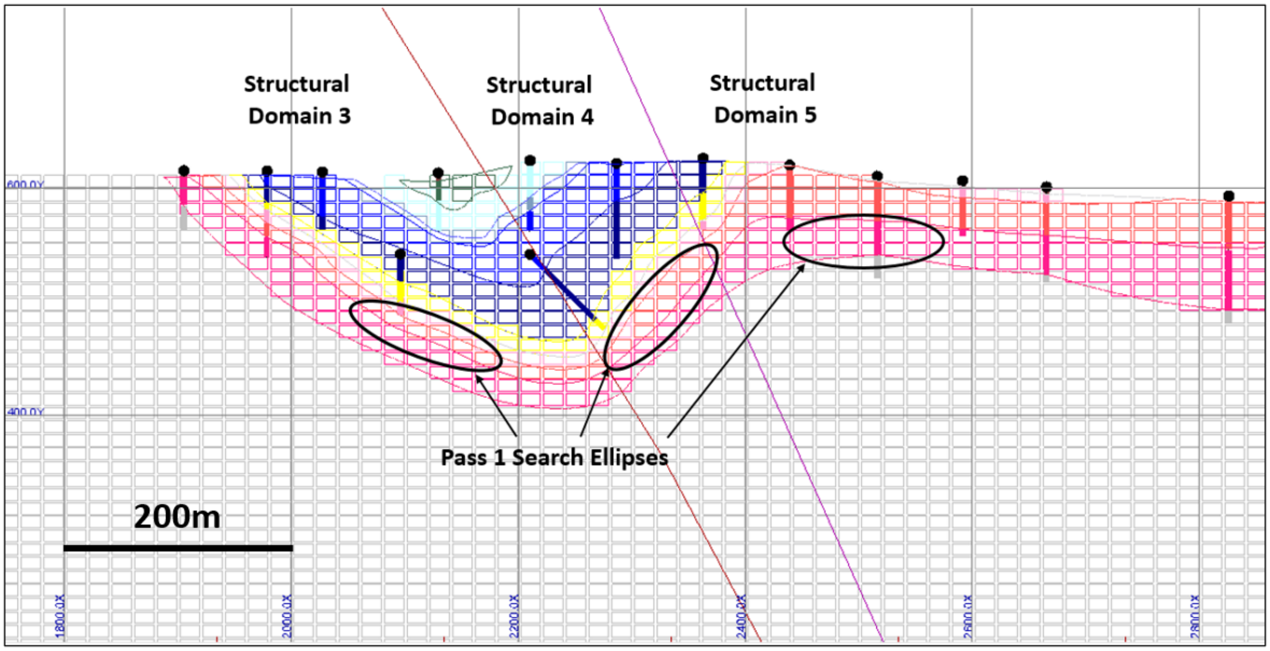


Table 14.14: Anisotropy Search Ellipsoid Orientations Presented by Structural Domain

Structural Domain	Rotation (GEMS ZXZ in Degrees)		
	Z	X	Z
100	0	30	0
200	0	65	0
300	-5	20	0
400	-10	-50	0
500	-10	0	0
600	-10	27	0
700	-10	-30	0
800	90	40	0
900	95	-25	0
1000	100	0	0
1100	115	50	0
1200	105	20	0
1300	110	-25	0
1400	90	0	0

Table 14.15: Search Ellipse Ranges by Pass

PASS	Search Ellipse Ranges (Meters)		
	X	Y	Z
1	100	75	20
2	200	150	40
3	300	200	60
4	400	250	80

14.5.5 Tonnage-Weighting of Estimated Grades

As a “percentage” type block model was used during estimation, tonnage-weighted average grades for each estimated attribute were calculated for each block. The calculation used the various rock types present in each block and their respective grades and tonnages to calculate a final grade for each 20 m x 20 m x 12 m block. These attributes will be used for final Mineral Resource reporting.

14.6 Mineral Resource Classification

The resource classification definitions used for this report are those published by the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM"). The "*CIM Definition Standards on Mineral Resources and Mineral Reserves*", prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM council on May 10, 2014, provides standards for the classification of Mineral Resources and Mineral Reserve estimates into various categories. The category to which a Resource or Reserve estimate is assigned depends on the level of confidence in the geological information available on the mineral deposit, the quality and quantity of data available, the level of detail of the technical and economic information which has been generated about the deposit and the interpretation of that data and information. Under CIM Definition Standards:

A *"Measured Mineral Resource"* is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of modifying factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resources. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

An *"Indicated Mineral Resource"* is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of modifying factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probably Mineral Reserve.

An *"Inferred Mineral Resource"* has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration, Confidence in the estimate is insufficient to allow the meaningful application of technical economic parameter or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

For the Scully Deposit, the classification of interpolated blocks was undertaken by considering the following criteria:

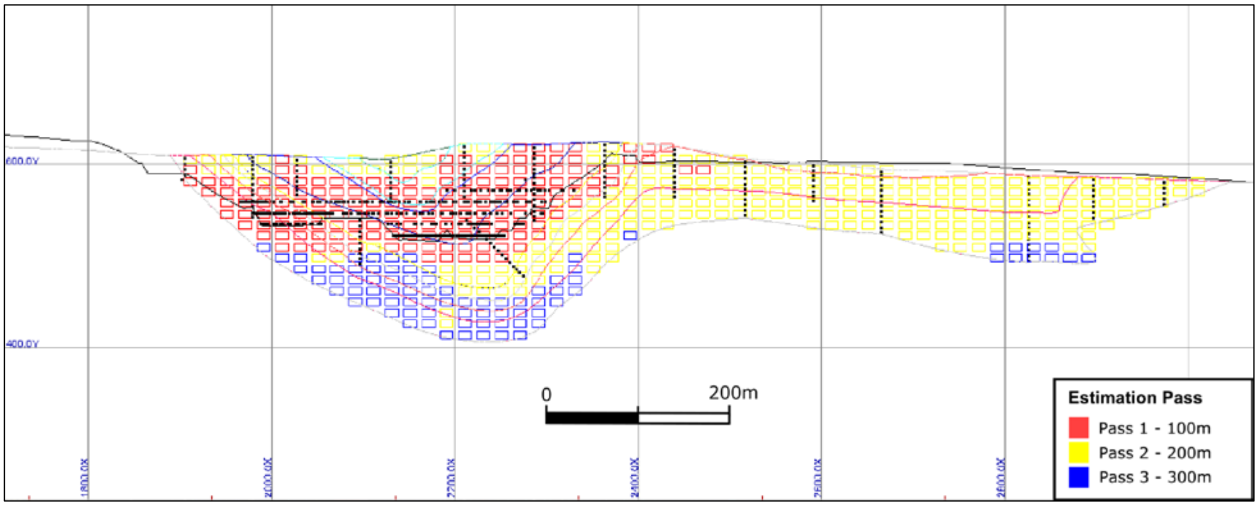
- Quality and reliability of drilling and sampling data.
- Distance between sample points (drilling density).
- Confidence in the geological interpretation.
- Continuity of the geological structures and the continuity of the grade within these structures.
- Variogram models and their related ranges (first and second structures).
- Statistics of the data population.
- Quality of assay data.

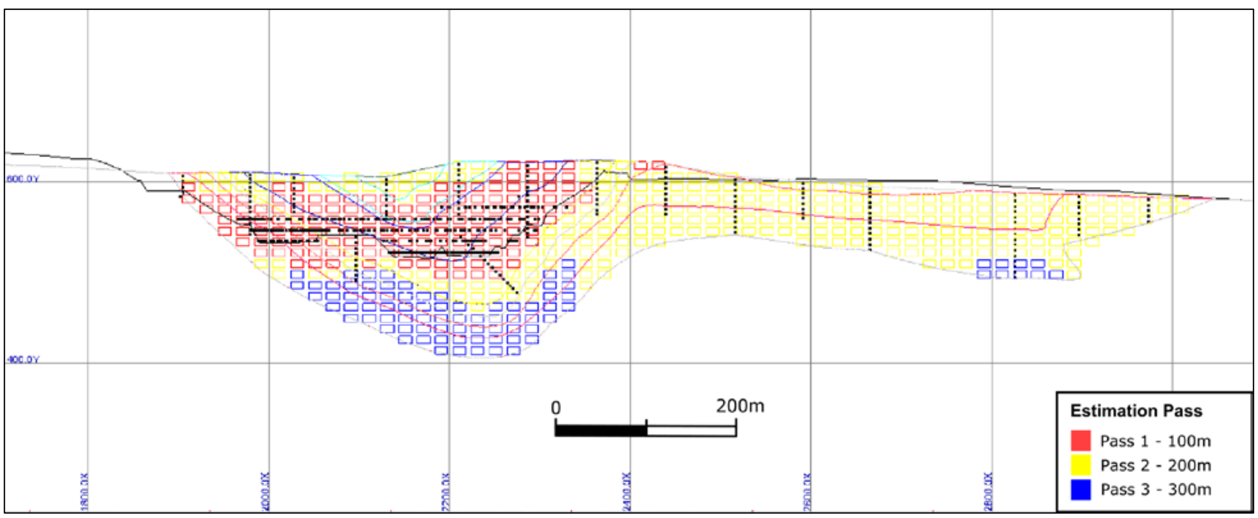
A preliminary Mineral Resource classification was produced utilizing estimation pass, where Pass 1 represents Measured Resources, Pass 2 represents Indicated Resources, and Pass 3 represents Inferred Resources. Pass 4 was reserved for unclassified mineral inventory. This method considers sample spacings, extrapolation of grades and the variogram ranges, and is a commonly applied method to produce an appropriate Mineral Resource classification.

The preliminary classification resulted in generally cohesive zones of Measured and Indicated Mineral Resources, however, several areas showed an interspersion of classifications, which creates an irregular and mismatched appearance (Figure 14.24, top image).

GMS developed a smoothing algorithm to produce more cohesive zones of Mineral Resource classifications. For each block, an average of the pass attribute from the six (6) surrounding blocks was calculated and is shown in Figure 14.24 (bottom image). The result was a cleaner, more continuous Mineral Resource classification.

Figure 14.24: Section 3197mE Showing Preliminary Mineral Resource Classification (Top) and Smoothed Mineral Resource Classification (Bottom) – Looking East.





14.7 Block Model Estimation Validation

Every step of the block modelling process, including assay and composite database, topography, drill hole location, down-hole survey, geology interpretation, geological coding, block model development and resource estimation and classification, was reviewed to ensure fair representation of the available data in the Scully resource model.

More specific validations were completed on the Scully block model including visual checks, global validations (such as Q:Q plots, comparative statistics) and local validations (swath plots and blasthole reconciliation).

14.7.1 Visual Validation

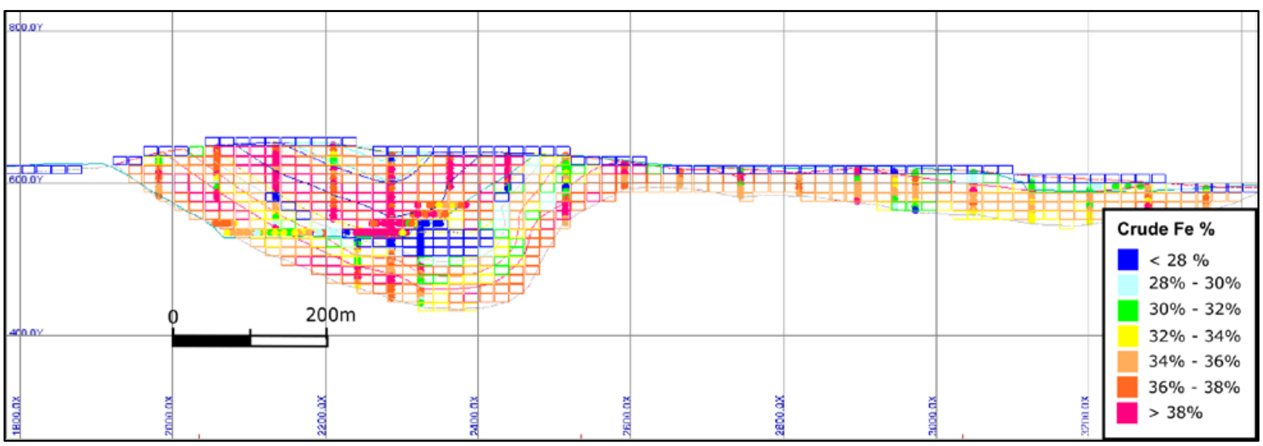
The visual checks consisted of visualization of slices of the block model, mineralized zones, and exploration drill hole and blast hole databases. The slicing was performed vertically on 76 m intervals and horizontally on 12 m intervals (bench by bench). The composites were visually compared with the different model attributes (rock types, grades) along the strike length of the deposit.

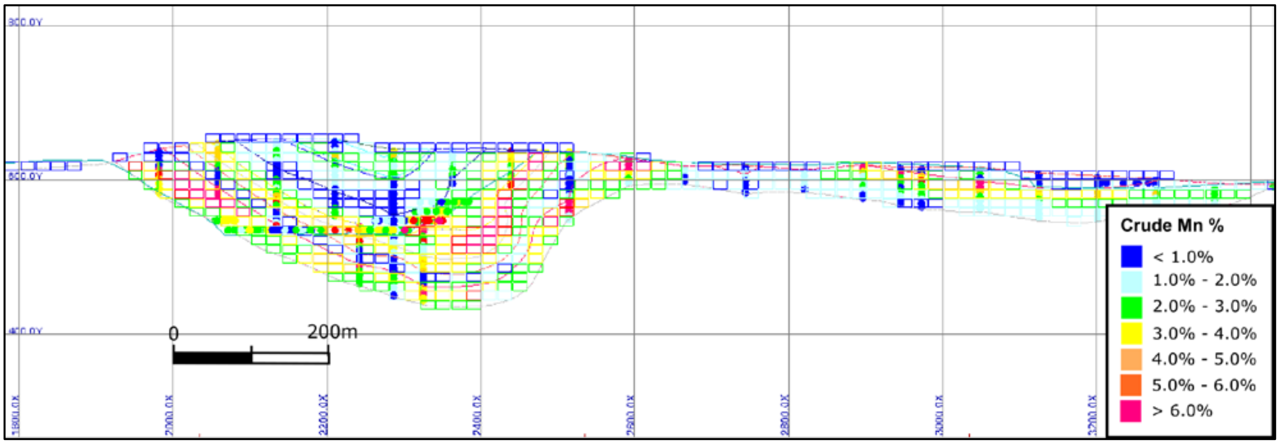
The ordinary kriged attributes Crude Fe%, Mn% and SAT% were found to be a good visual representation of the drill hole composites and are shown in Figure 14.25.

14.7.2 Statistical Validation

A statistical comparison between drill hole composites used in the interpolation and block grades was performed to evaluate if samples used in the estimation are globally represented in the block model. The inclusion of blast holes into the first estimation pass complicated the reconciliation of grade between the drill hole composites and the block model grades. Declustering weights based on crude Fe% were applied to the composites in an attempt for overcome the clustering of data caused by the blast holes.

Figure 14.25: Section 3653 - Block Grades Vs. Composite Grades for Crude Fe% (top), Mn% (Middle) and SAT% (Bottom) – Looking East




0
200m
Crude Mn %
< 1.0%
1.0% - 2.0%
2.0% - 3.0%
3.0% - 4.0%
4.0% - 5.0%
5.0% - 6.0%
> 6.0%

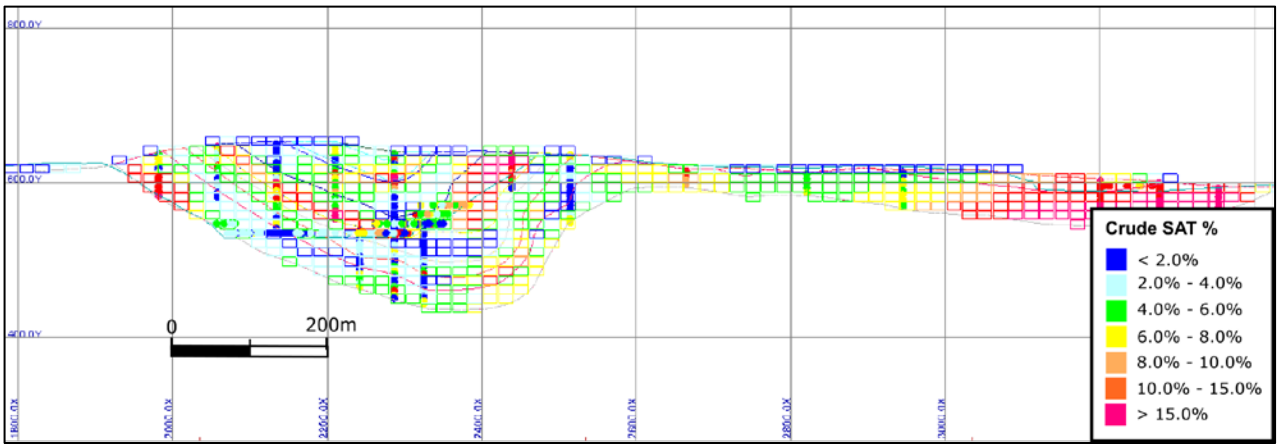

0
200m
Crude SAT %
< 2.0%
2.0% - 4.0%
4.0% - 6.0%
6.0% - 8.0%
8.0% - 10.0%
10.0% - 15.0%
> 15.0%

Table 14.16: Global Statistical Comparison Between Composites and Blocks for Measured and Indicated Blocks

Rock Code	Composites				Blocks				% Change in Grade			Reduction in Coeff. of Variation			% of Block Model (M+I)
	Fe%	Mn%	SAT%	No. Obs	Fe%	Mn%	SAT%	No. Obs	Fe%	Mn%	SAT%	Fe	Mn	SAT	
21	31.57	0.54	6.27	1265	31.95	0.45	5.21	5957	1.2%	-19.4%	-20.3%	-19%	-5%	-15%	3%
22	34.70	0.60	6.02	3190	34.65	0.62	4.70	8384	-0.2%	2.3%	-28.2%	-27%	-64%	0%	4%
23	31.74	0.70	5.16	6721	31.88	0.71	4.41	22230	0.4%	0.5%	-16.9%	-31%	-37%	-7%	10%
31	36.00	1.22	3.75	10940	35.88	1.23	2.95	25707	-0.4%	1.4%	-27.1%	-46%	-45%	-23%	11%
32	32.17	1.86	3.72	6170	32.20	1.95	3.43	15167	0.1%	4.8%	-8.4%	-31%	-25%	-31%	7%
33	37.92	1.62	5.02	19048	37.77	1.61	3.75	36270	-0.4%	-0.4%	-33.7%	-35%	-32%	-15%	16%
34	36.58	1.53	7.36	12502	36.26	1.58	6.79	31343	-0.9%	2.8%	-8.5%	-35%	-50%	-18%	13%
51	34.51	2.89	6.22	5788	34.22	3.02	7.25	21639	-0.8%	4.2%	14.3%	-50%	-33%	-26%	9%
52	31.92	1.43	4.86	6233	31.87	1.53	4.72	23810	-0.2%	6.2%	-2.9%	-41%	-31%	-19%	10%
53	35.73	3.06	6.31	10488	35.74	3.30	5.94	42092	0.0%	7.4%	-6.1%	-46%	-28%	-25%	18%

Global reproduction of the mean composite grades was achieved for crude Fe% and Mn%; however, SAT% was more problematic to reconcile. This is likely due to the influence of very high-magnetite composites in areas categorized as Inferred (included in the comparison shown in Table 14.16) skewing the statistics in the blocks compared to the composites.

Concentrate grades were also compared, and global block grades were found to be within 10% of the composite grades for each rock code.

14.7.3 Quantile : Quantile Plots

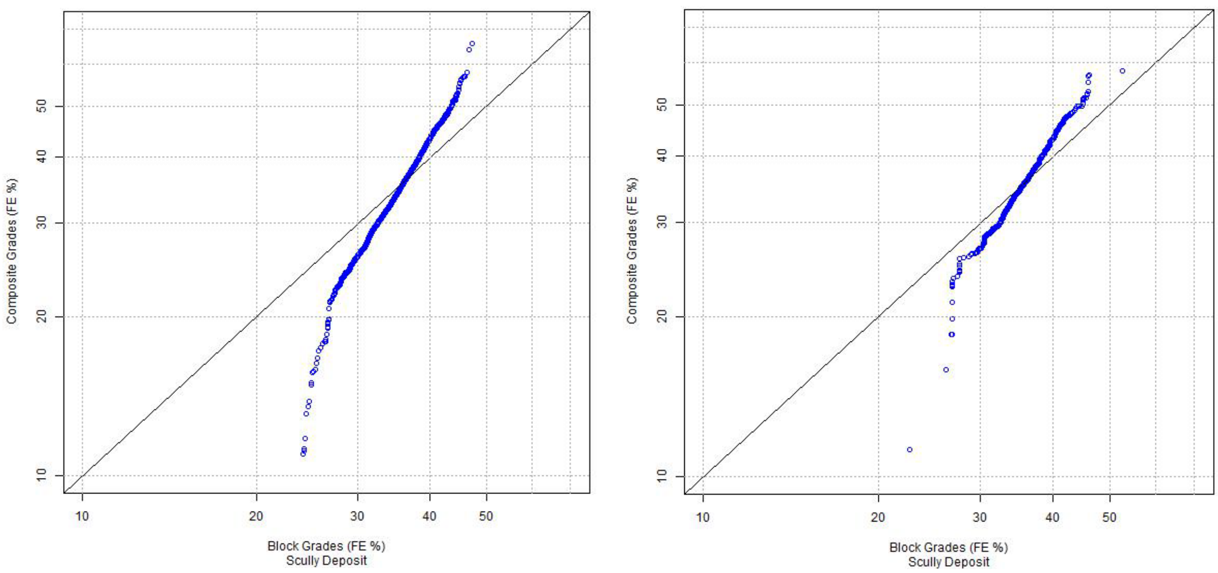
In addition to descriptive statistics, Q:Q plots were generated to assess the *distribution* of crude and concentrate grades of composites against blocks on a rock code by rock code basis. These plots are useful in assessing the degree of smoothing (conditional bias) observed during the grade estimation process and can identify any significant over/under estimation of grades.

Measured Resources were compared separately to Indicated Mineral Resources for crude Fe% and Mn % due to the inclusion of blast hole composites into the estimation of Measured Mineral Resources. For the remaining attributes, Measured and Indicated Mineral Resources were grouped to produce the Q:Q plots.

Examples of Q:Q plots for rock code 53 (the most significant contributor to future production) are shown in Figure 14.26, Figure 14.27 and Figure 14.28.

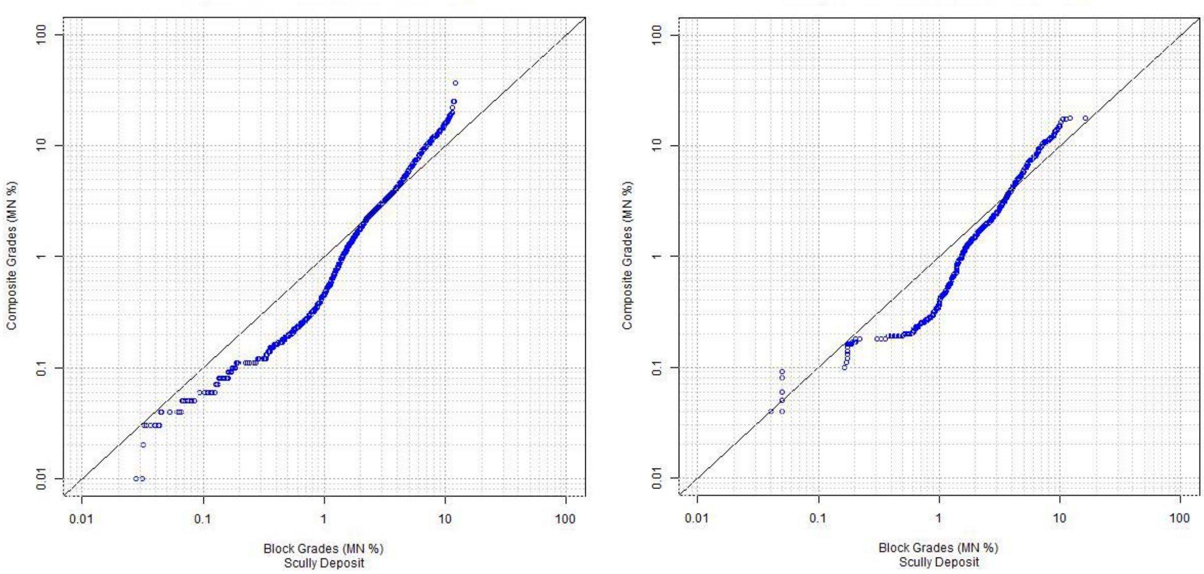
Regarding crude Fe% grades, smoothing is present (and expected) in the Measured Resource due to the dense spacing of the blast holes and the comparatively large search ellipse and variogram ranges resulting in elevated levels of conditional bias. For Indicated and Inferred Mineral Resources, smoothing is observed to a lesser extent, as the spatial configuration of composites to blocks is more typical of the distances observed in the variogram models.

Figure 14.26: Q:Q Plot Comparing Crude Fe % Block Grades (X-Axis) and Composite Grades (Y-Axis) for Measured Resources (Left) and Indicated + Inferred Resources (Right) for Rock Code 53



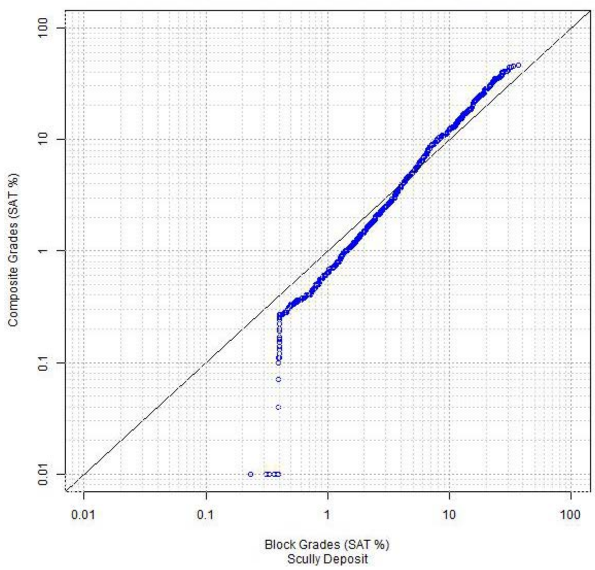
Regarding Mn%, smoothing is within acceptable limits, with good reproduction of grades at each end of the distributions. A small positive bias exists in the blocks at lower crude Mn% grades.

Figure 14.27: Q:Q Plot Comparing Crude Mn% Block Grades (X-Axis) and Composite Grades (Y-Axis) for Measured Resources (Left) and Indicated + Inferred Resources (Right) for Rock Code 53



Regarding crude SAT%, minimal conditional bias is observed, with excellent reproduction of the composite distribution for rock code 53. No significant shift in mean grades were observed for any of the crude grades.

Figure 14.28: Q:Q Plot Comparing Crude SAT% Block Grades (X-Axis) and Composite Grades (Y-Axis) for Combined Measured and Indicated Resources for Rock Code 53



Q:Q plots for concentrate grades were also compiled by rock code, and showed varying degrees of smoothing due to the single-pass methodology adopted.

14.7.4 Local Statistical Validation – Swath Plots

The swath plot method is considered a local validation, which works as a visual mean to compare estimated block grades against composite grades within a 3D moving window. In addition, it can identify possible bias in the interpolation (i.e., over/under estimation of grades).

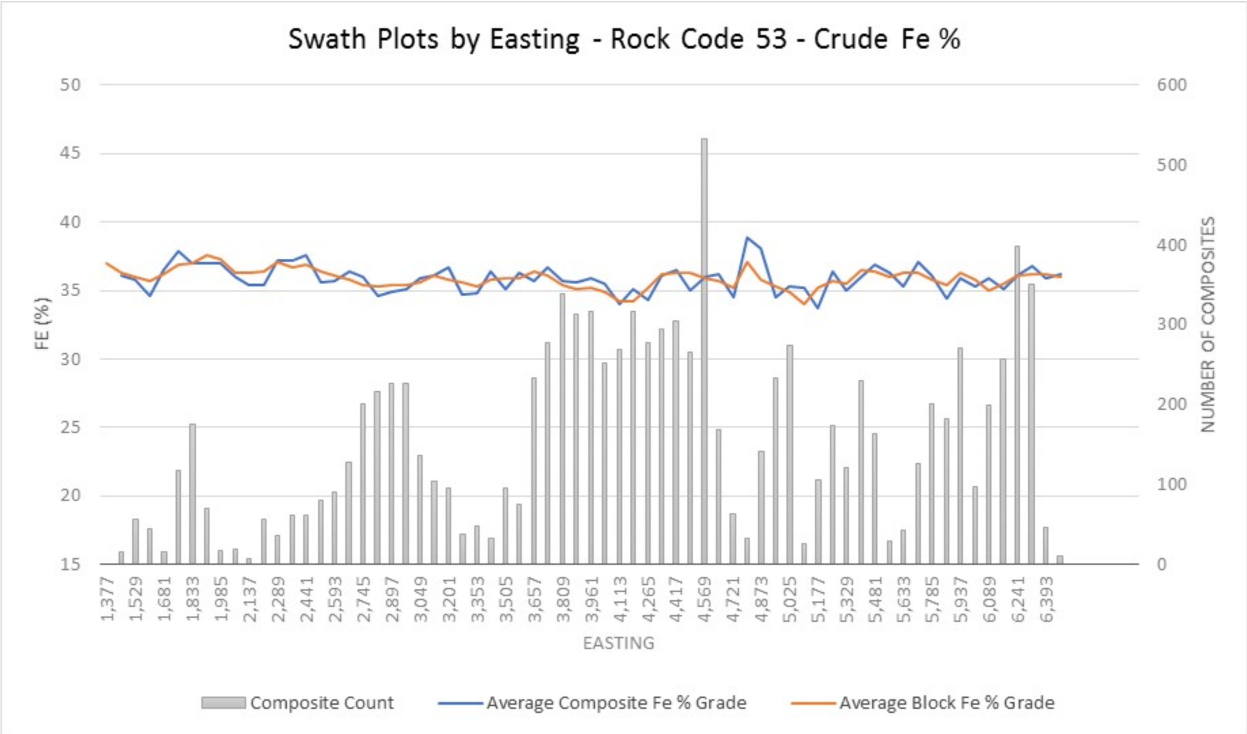
Swath plots were generated for all rock codes of the Upper, Middle and Lower Member, at increments of 76 m (Easting) for all crude and concentrate grades. Only blocks within Measured or Indicated classifications were considered.

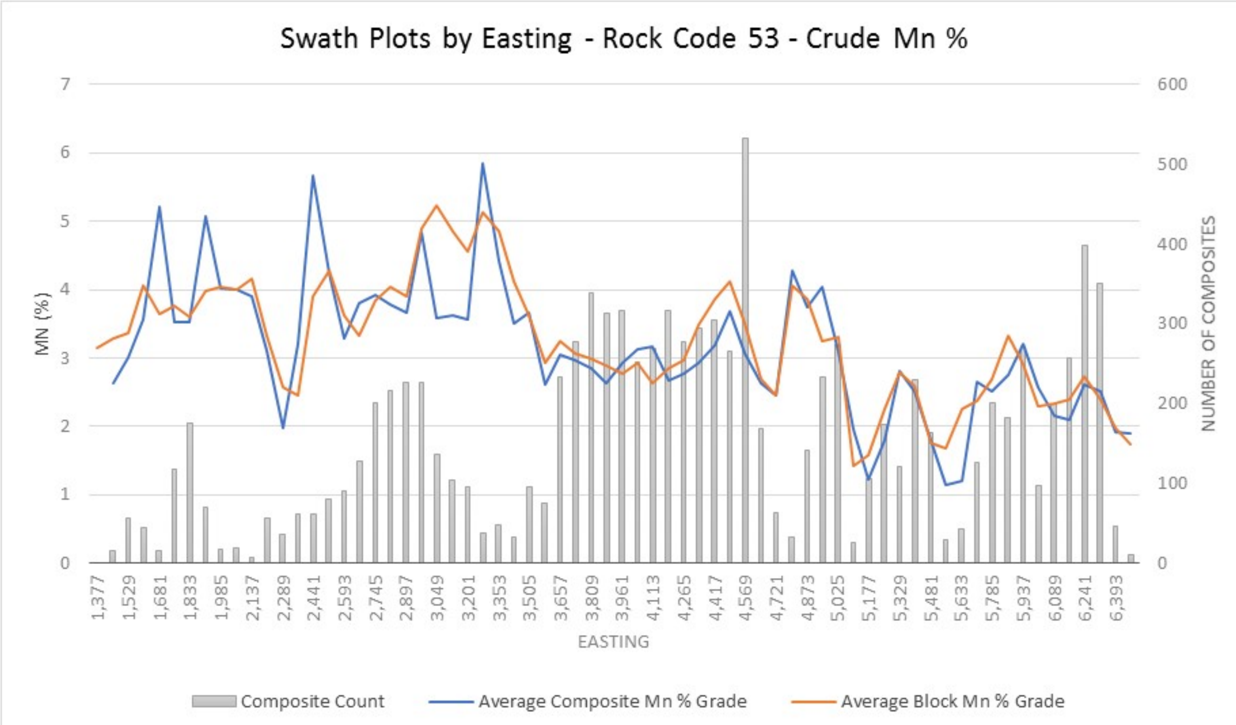
Peaks and lows in estimated grades should generally follow peaks and lows in composites grades in well informed areas of the block model, whereas less informed areas can occasionally show some discrepancies between the grades.

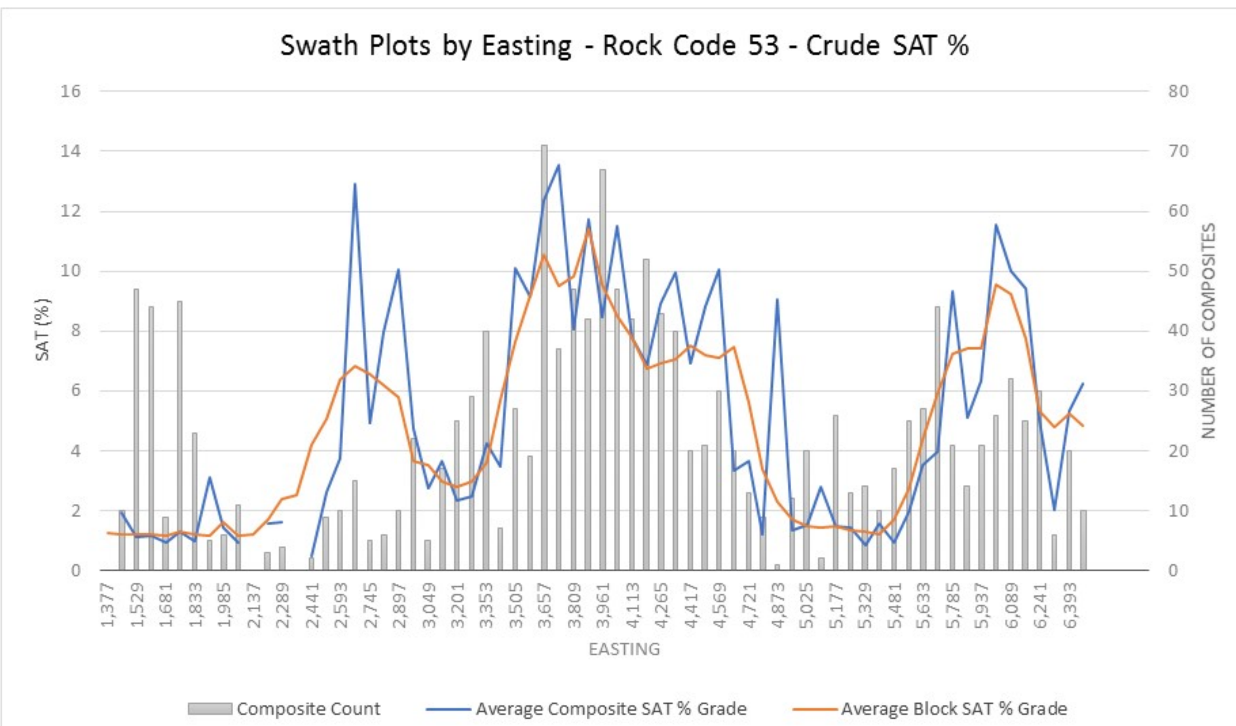
Figure 14.29 presents the results of the swath plots for the crude grades for rock code 53. Local reproduction of grade is considered acceptable, and no significant bias is observed in any of the crude grades.

Swath plots were also produced for concentrate grades, and no issues were identified.

Figure 14.29: Swath Plots by Easting for Crude Fe% (Top), Mn% (Middle) and SAT% (Bottom) for Measured and Indicated Mineral Resources Combined




Swath Plots by Easting - Rock Code 53 - Crude Mn %
MN (%)
NUMBER OF COMPOSITES
EASTING
Composite Count
Average Composite Mn % Grade
Average Block Mn % Grade


Swath Plots by Easting - Rock Code 53 - Crude SAT %
SAT (%)
NUMBER OF COMPOSITES
EASTING
Composite Count
Average Composite SAT % Grade
Average Block SAT % Grade

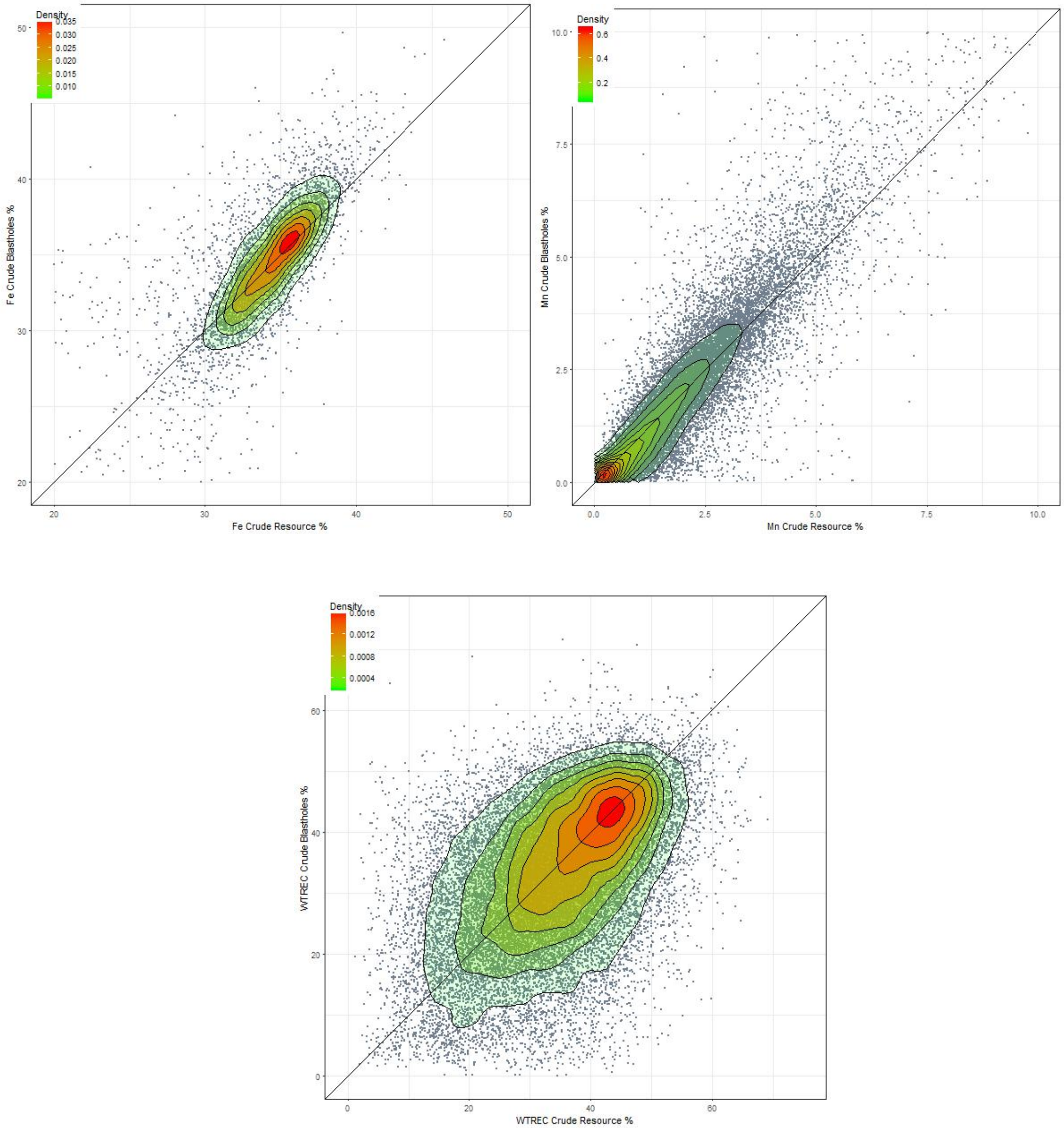
14.7.5 Blastholes Vs. Exploration Drill Holes Reconciliation

To assess the application of blastholes during the estimation process, GMS compared the average grade of blastholes within each block against the interpolated grade using only the exploration drill holes. This method is considered a reconciliation exercise between the two datasets (the blastholes representing "true grade" of the block, and the interpolated grades from exploration drill holes representing the "predicted" grade). The following steps were followed:

- Calculate the average grade of the blastholes for attributes Fe, Mn and WTREC that are spatially contained within a given block (point to block transfer).
- Interpolate attributes Fe, Mn and WTREC using only the exploration drill holes on a rock code by rock code basis. Calculate weighted average grades for each block using the rock code percentages.
- Produce scatter plots comparing the blasthole grades and the interpolated grades.

The results are shown in Figure 14.30.

Figure 14.30: Interpolated Block Grades from Exploration Drill Holes (X-Axis) Vs. Average Blasthole Grades Within a Given Block (Y-Axis). Fe (Left), Mn (Right) and Weight Recovery (Bottom) with Point Density Contours.



This validation was designed to assess how accurately the exploration drilling could predict the crude grades of the blastholes. Fe and Mn% blasthole grades were reconciled with a relatively minor range of error, however, weight recovery proved more problematic to reconcile. Although no significant bias was observed, the large range of error for WTREC could be due to local inaccuracies of geological interpretation, or non-geological factors such as shaker table performance, grind size, etc.

14.7.6 Mining Reconciliation

A comparison between the tonnes and grade in the short-term block model and the 2018 MRE block model has been made on a monthly basis since mid-2019 when mining operations began. In addition, these metrics are compared to tonnages and grades derived from the dispatch system, the mill feed and finally the two concentrate streams (wet and dry) that exit the plant.

Concerning the Mineral Resource, the most significant comparison is between the long-range block model (MRE) and the short-range block model (Grade Control Model). The data has been simplified by GMS into half-year timeframes, to address any variability associated with monthly stockpile fluctuations.

Table 14.17 shows a summary of reconciliation data for the 2019 and 2020 provided to GMS from Tacora, comparing the 2018 MRE with the short-range block model. Apart from the second half of 2019 which was affected by overburden-related effects, GMS believes that the results fall within acceptable limits, and the long-range block model performs well.

Table 14.17 Reconciliation of Crude Grades - 2019 and 2020

	GMS 2018				Short Term Model (Grade Control)				Difference %			
Time Period	Ore Tonnes	Crude Fe%	Crude Mn%	Crude Mag%	Ore Tonnes	Crude Fe%	Crude Mn%	Crude Mag%	Ore Tonnes	Crude Fe%	Crude Mn%	Crude Mag%
H2 2019*	2,956	34.6	5.2	3.1	4,230	35.0	3.6	2.9	+43%	+1%	-30%	-6%
H1 2020	4,793	35.4	5.5	3.3	4,515	36.3	5.2	3.2	-6%	+2%	-6%	-1%
H2 2020**	4,036	34.7	5.0	3.0	3,483	34.5	4.5	3.2	-13%	-1%	-10%	+7%

**The first few months of production were mainly surface blocks in the 2018 MRE where grades were diluted with the overburden and converted to waste in the reserve. The short-term model reported significant gains during this period as overburden was stripped and material beneath was ore.*

***H2 2020 does not include the month of December.*

14.8 Constrained Mineral Resources

14.8.1 Optimization Parameters

A Lerchs-Grossman open-pit shell was produced using Whittle and the same optimization parameters as used for reserves with the exception that Measured, Indicated and Inferred Mineral Resource categories were included in the optimization process, and a concentrate price CFR client of USD 93.5/dmt was used (14% higher than the reserve price assumption). Key infrastructure (rail loop, processing plant) and lakes were considered by incorporating a mining exclusion zone into the pit optimization process. The end-of-year 2020 mining topography was used as the mining depletion surface. A 10% WTREC lower cut-off was used during the optimisation process, which represents a marginal break-even cash flow cut-off for the operation as shown in Table 14.18 below.

Table 14.18: Marginal Break-even Cash Flow Calculations for the Mineral Resource

	(USD / t of Concentrate)
Revenue CFR Client	93.50
Net Revenue – Pointe Noire	75.65
Net Revenue – Scully	54.01
Recoveries	**%**
Laboratory Weight Recovery (Wilfley Table)	10
Wet Weight Recovery	9.7
Total Weight Recovery	8.9
	USD / t of ore
Marginal Revenue – 10% WREC	4.81
	Marginal Cash Cost
	(USD/t ore)
Mining	-
Processing	3.82
G&A Costs	0.27
Sustaining Capital	0.27
Closure	-
Rehandling	0.46

Total	**4.82**

Note: Net Revenue Pointe Noire include 1% Loss of Concentrate

Additional manual manipulation was undertaken to remove isolated partial blocks that do not meet REEE (Reasonable Prospects of Eventual Economic Extraction), and the final blocks reported in the mineral resource are tagged in the INPIT attribute in the block model.

14.8.2 Open-Pit Constrained Mineral Resource

GMS calculated the open-pit constrained tonnages and grades using both a Fe% lower cut-off and a WTREC% lower cut-off, and found the differences to be immaterial between the two methods. As the WTREC% (wet weight recovery %) is the principal controlling economic factor for the mining operation, WTREC% was used as a lower cut-off to report the official Mineral Resource Tabulation.

The Measured and Indicated Mineral Resource for the Scully Deposit is estimated 721.9 Mt at an average grade of 34.7% Fe and Inferred Mineral Resource to 263.4 Mt at an average grade of 34.1% Fe using a lower cut-off of 10% WTREC. Table 14.19 presents the resource estimation tabulation by resource category.

GMS emphasizes that the WTREC and concentrate grades quoted in Table 14.19 are only an indication of the recoveries and concentrate grades derived from Wet Shaking Table (Wilfley Table) test work. No modifying factors have been applied to the weight recoveries nor concentrate grades presented in this Mineral Resource to account for the plant performance or the scaling up of the recovery process.

Table 14.19: 2021 Mineral Resource Estimate for the Scully Deposit at a 10% WTREC Lower Cut-Off

Classification	Tonnage (dry)	Fe	Mn	SAT	WTREC	Fe Conc.	SiO_2 Conc.	Mn Conc.	Sat Conc.
	Mt	%	%	%	%	%	%	%	%
Measured	195.6	35.1	2.3	5.9	36.6	64.6	3.7	2.0	7.1
Indicated	526.3	34.5	2.4	6.1	35.5	63.9	3.9	2.5	8.2
Total M&I	**721.9**	**34.7**	**2.4**	**6.0**	**35.8**	**64.1**	**3.8**	**2.4**	**7.9**
Inferred	263.4	34.1	2.1	6.2	34.0	64.2	3.9	2.1	9.1

Notes on Mineral Resources:

13. *The Mineral Resources were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards for Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council May 10th, 2014.*
14. *The independent and qualified persons for the 2021 Scully resource estimate, as defined by NI 43-101, are Mr. Réjean Sirois, P. Eng., and Mr. James Purchase, P.Geo from G Mining Services Inc. The effective date of the estimate is February 15th, 2021.*
15. *The Mineral Resources are reported at a lower cut-off grade of 10% WTREC, within rock codes 22, 31, 32, 33, 34, 51, 52 and 53 only.*
16. *The Mineral Resources have been depleted using a mining surface representing the end-of-year 2020.*
17. *The Mineral Resources are reported within an optimized Whittle shell using a long-term iron price of USD 93.5/dmt conc. with an exchange rate of 1.25 USD/CAD. Measured, Indicated, and Inferred categories are considered. Manual manipulation was undertaken to remove isolated partial blocks that do not meet REEE (Reasonable Prospects of Eventual Economic Extraction).*
18. *The Mineral Resources are reported inclusive of the Mineral Reserves.*
19. *"SAT" stands for Satmagan or Saturation Magnetization Analyzer, an instrument which estimates magnetite content in samples.*
20. *"Conc." stands for concentrate grades obtained from shaker table test work.*
21. *"WTREC" stands for Wilfley shaking table recoveries (wet process) under laboratory conditions.*
22. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Mineral Reserves.*
23. *The number of metric tonnes was rounded to the nearest hundred thousand. Any discrepancies in the totals are due to rounding effects.*
24. *The weight recoveries and concentrate grades stated in the table above are derived from the Wilfley shaking table process. These values are only indicative of potential concentrate recovery and quality under laboratory conditions, and no modifying factors have been applied to compensate for the scaling up of the recovery process to industrial levels.*

The conceptual open pit shell used to constrain the Mineral Resource is shown in Figure 14.31.

Figure 14.31: 3D View Showing Mineral Resource Blocks Located Inside the Whittle Shell – Coloured by crude Fe %

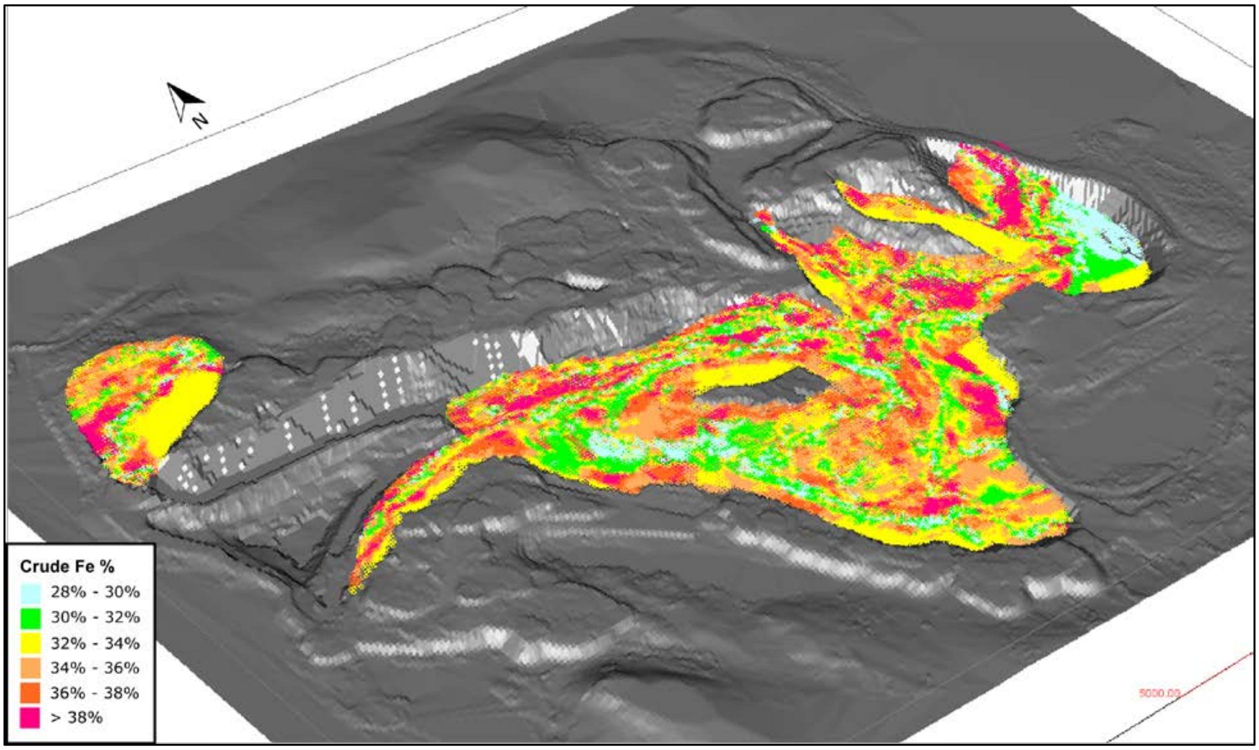


Table 14.20 presents the Mineral Resource tabulation using a 20% Fe lower cut-off for comparison purposes. Reporting the Mineral Resource using a 20% Fe cut-off grade results in an immaterial difference in tonnage in the Measured and Indicated Category (0.2% increase) and Inferred Category (0.1% increase) when compared to Mineral Resources using a 10% WTREC cut-off value.

Table 14.20: 2021 Mineral Resource Estimate for the Scully Deposit at a 20% Fe Lower Cut-Off

Classification	Tonnage (dry)	Fe	Mn	SAT	WTREC	Fe Conc.	SiO_2 Conc.	Mn Conc.	Sat Conc.
	Mt	%	%	%	%	%	%	%	%
Measured	195.5	35.2	2.3	5.9	36.5	64.6	3.7	2.0	7.1
Indicated	528.1	34.5	2.4	6.1	35.4	63.9	3.9	2.5	8.2
Total M&I	**723.6**	**34.7**	**2.4**	**6.0**	**35.8**	**64.1**	**3.8**	**2.4**	**7.9**
Inferred	263.7	34.1	2.1	6.1	34.0	64.2	3.9	2.1	9.1

14.8.3 Grade Sensitivity Analysis – Measured and Indicated Category

The sensitivity of the block model estimates to changes in the WTREC% and crude Fe% cut-off grade is illustrated in Table 14.21 and Table 14.22 respectively for open-pit constrained Measured and Indicated Mineral Resources. Grade tonnage curves are presented in Figure 14.32 and Figure 14.33.

Table 14.21: Grade and Tonnage Sensitivity to WTREC% Cut-off Grade (Measured and Indicated Resources)

Cut-Off Grade WTREC %	Tonnage (dry)	Fe	Mn	SAT	WTREC	Fe Conc.	SiO_2 Conc	Mn Conc.	SAT Conc.
%	Mt	%	%	%	%	%	%	%	%
10	721.9	34.7	2.4	6.0	35.8	64.1	3.8	2.4	7.9
14	719.6	34.7	2.4	6.0	35.9	64.1	3.8	2.4	7.9
18	712.2	34.7	2.4	6.0	36.1	64.1	3.8	2.4	7.8
22	692.2	34.8	2.4	5.9	36.5	64.1	3.8	2.4	7.8
26	644.8	34.9	2.5	5.9	37.4	64.1	3.8	2.4	7.8
30	552.3	35.2	2.6	5.8	39.0	64.0	3.8	2.5	7.7
34	427.6	35.6	2.8	5.6	41.0	63.9	3.8	2.6	7.5
38	288.5	36.1	3.0	5.1	43.4	63.7	3.8	2.8	7.0

Figure 14.32: Grade-Tonnage Curve for In-Pit Measured and Indicated Resources using a WTREC% Lower Cut-off

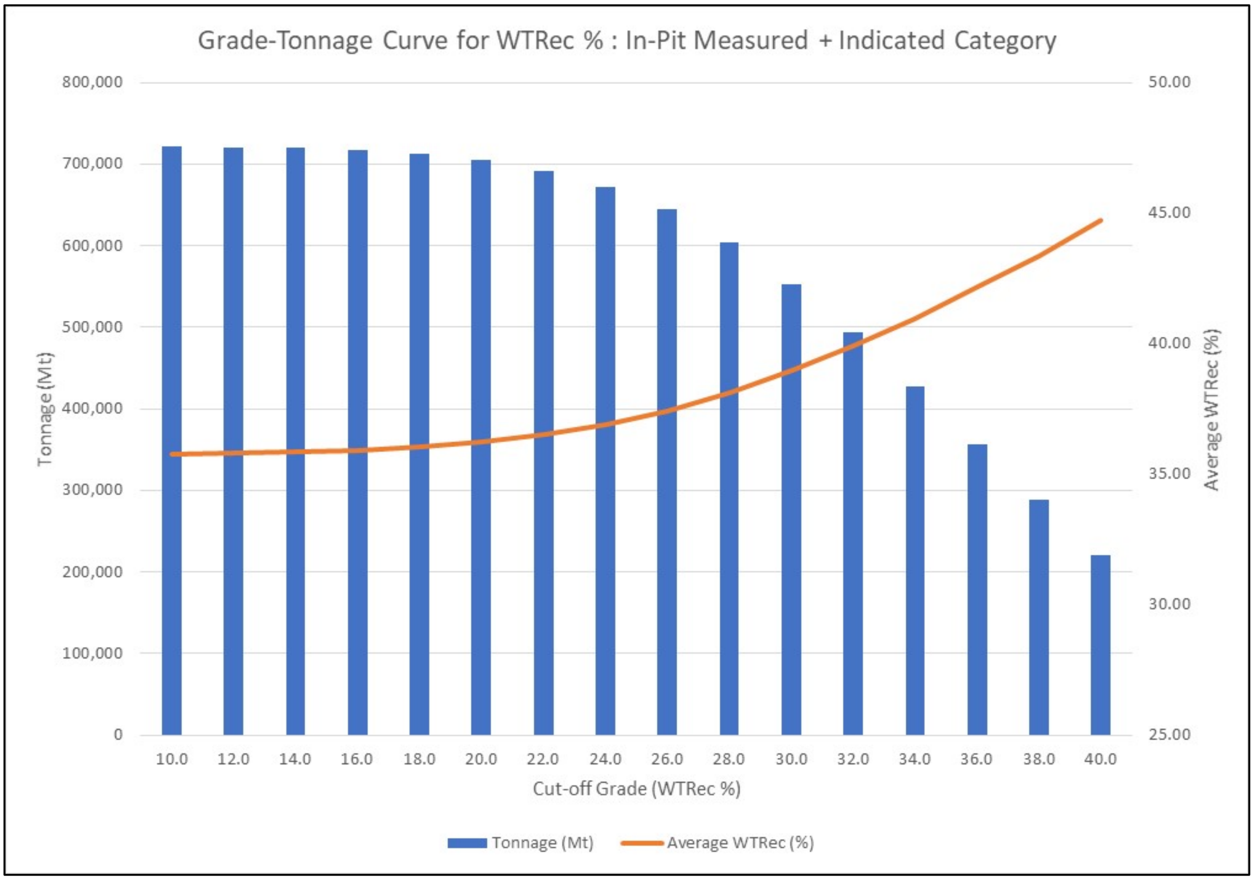
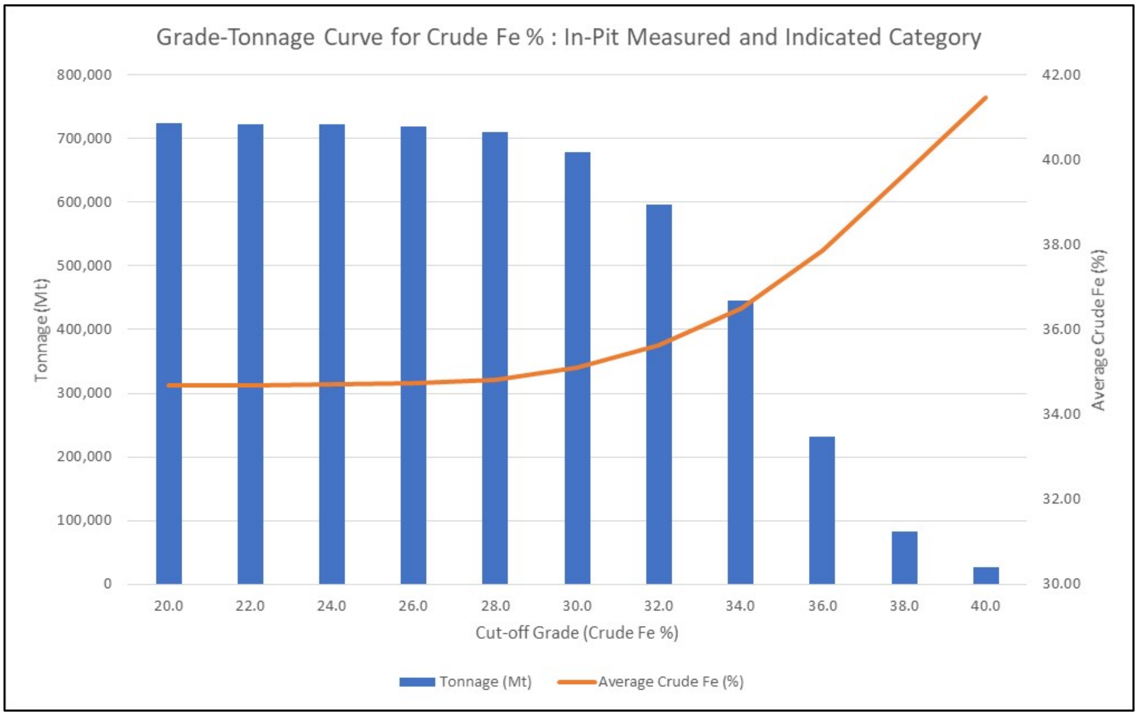


Table 14.22: Grade and Tonnage Sensitivity to Crude Fe% Cut-Off Grade (Measured and Indicated Resources)

Cut-Off Grade Fe	Tonnage (dry)	Fe	Mn	SAT	WTREC	Fe Conc.	SiO_2 Conc	Mn Conc.	SAT Conc.
%	Mt	%	%	%	%	%	%	%	%
20%	723.6	34.7	2.4	6.0	35.7	64.1	3.8	2.4	7.6
24%	721.9	34.7	2.4	6.0	35.7	64.1	3.8	2.4	7.5
28%	711.1	34.8	2.4	6.0	35.9	64.2	3.8	2.4	7.5
32%	597.2	35.6	2.5	5.7	37.0	64.2	3.8	2.4	7.3
36%	231.8	37.9	2.6	4.4	40.0	64.3	3.7	2.4	5.7
40%	26.0	41.5	2.2	3.6	41.8	65.0	3.4	1.8	4.7

Figure 14.33: Grade-Tonnage Curve for In-Pit Measured and Indicated Resources using a Fe% Lower Cut-off



It can be observed that only marginal increases in tonnage are observed below 18% WTREC or 28% Fe.

14.8.4 Grade Sensitivity Analysis – Inferred Category

The sensitivity of the block model estimates to changes in the WTREC% and crude Fe% cut-off grade is illustrated in Table 14.23 and Table 14.24 respectively for open-pit constrained Inferred Mineral Resources. Grade tonnage curves are presented in Figure 14.34 and Figure 14.35. Similarly, minimal tonnage increases are observed below 18% WTREC or 28% Fe.

Table 14.23: Grade and Tonnage Sensitivity to WTREC% Cut-Off Grade (Inferred Category)

Cut-Off Grade WTREC %	Tonnage (dry)	Fe	Mn	SAT	WTREC	Fe Conc.	SiO_2 Conc	Mn Conc.	SAT Conc.
%	Mt	%	%	%	%	%	%	%	%
10	263.4	34.1	2.1	6.1	34.0	64.2	3.9	2.1	9.1
14	262.3	34.1	2.1	6.2	34.1	64.2	3.9	2.1	9.1
18	258.8	34.1	2.1	6.1	34.4	64.3	3.9	2.1	9.1
22	246.9	34.3	2.1	6.1	35.0	64.3	3.9	2.1	9.2
26	223.6	34.5	2.2	6.1	36.2	64.4	3.9	2.1	9.3
30	182.5	34.8	2.2	6.0	38.0	64.4	3.9	2.2	9.2
34	129.2	35.4	2.3	5.9	40.4	64.2	3.9	2.3	8.5
38	78.6	36.4	2.5	5.0	43.3	64.0	3.8	2.5	6.7

Figure 14.34: Grade-Tonnage Curve for In-Pit Inferred Category using a WTREC% Lower Cut-off

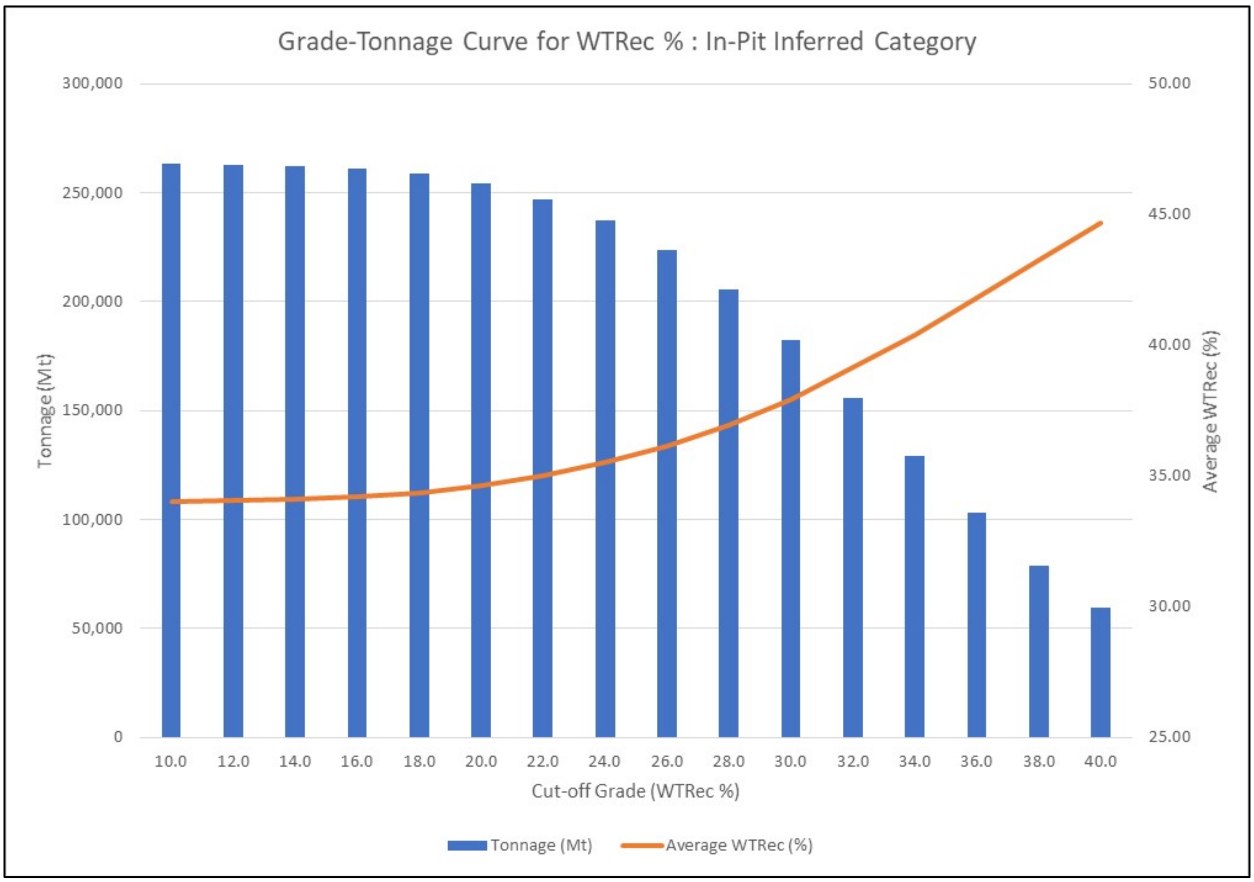
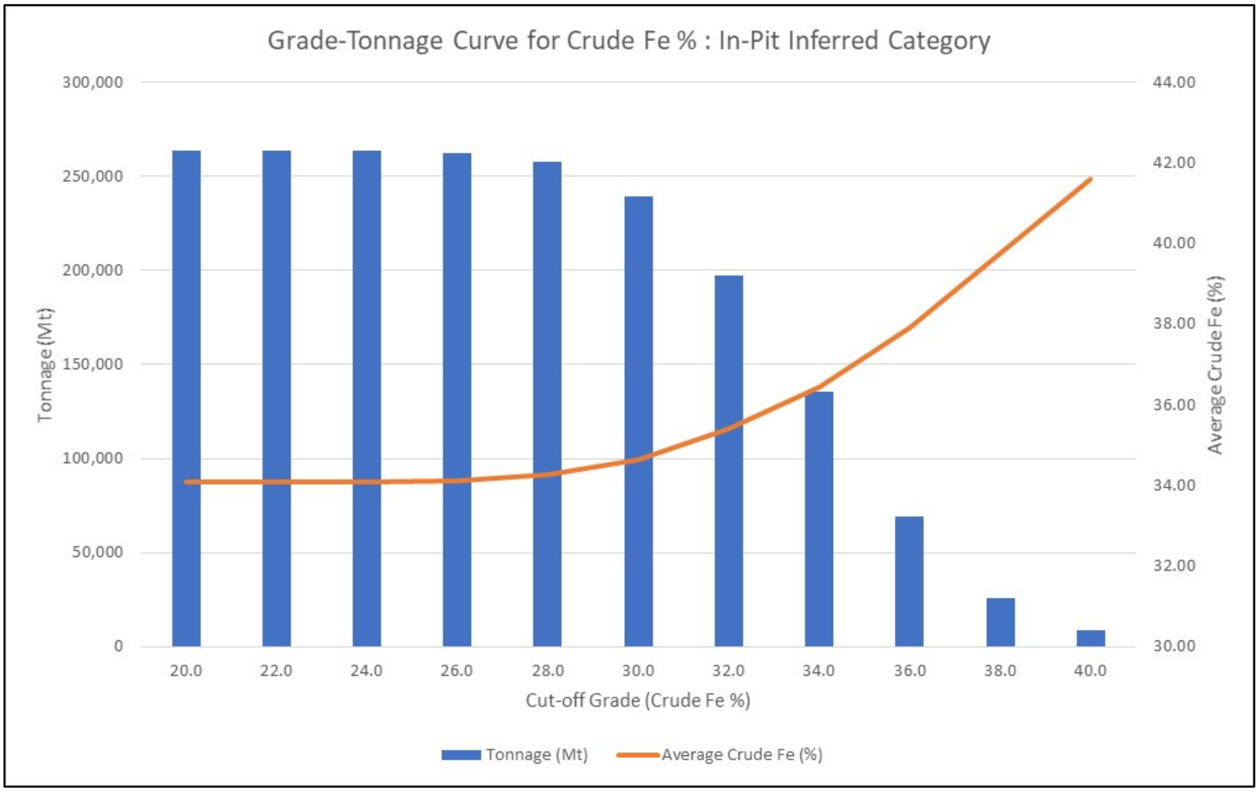


Table 14.24: Grade and Tonnage Sensitivity to Crude Fe% Cut-Off Grade (Inferred Category)

Cut-Off Grade Fe	Tonnage (dry)	Fe	Mn	SAT	WTREC	Fe Conc.	SiO_2 Conc	Mn Conc.	SAT Conc.
%	Mt	%	%	%	%	%	%	%	%
20%	263.7	34.1	2.1	6.1	34.0	64.2	3.9	2.1	9.1
24%	263.4	34.1	2.1	6.1	34.0	64.2	3.9	2.1	9.1
28%	257.5	34.3	2.1	6.1	34.2	64.3	3.9	2.1	9.0
32%	197.0	35.4	2.2	5.6	35.8	64.3	3.8	2.2	8.0
36%	68.9	37.9	2.4	5.5	39.6	64.3	3.7	2.3	7.1
40%	8.6	41.6	1.5	4.1	43.9	65.2	3.8	1.4	6.7

Figure 14.35: Grade-Tonnage Curve for In-Pit Inferred Category using a Fe% Lower Cut-off



14.9 Comparison with 2018 Mineral Resource

Table 14.25 compares the tonnages and grades of the previous 2018 MRE with the 2021 MRE (subject of this report). A net loss of 12.5 Mt was observed in the Measured and Indicated category, mainly due to mining depletion and minor effects associated with the new pit optimisation. In addition, there has been a net gain of 26.4 Mt in the Inferred category mainly due to the increase in iron price used for the optimisations (USD 93.5 in 2021 vs. USD 79 in 2018) and other economic parameters, which has resulted in a deeper pit shell.

Table 14.25: Comparison Between the 2018 and 2021 Scully Mineral Resource Estimates

Classification	Tonnage (dry)	Fe	Mn	SAT	WTREC	Fe Conc.	SiO_2 Conc.	Mn Conc.	Sat Conc.
	Mt	%	%	%	%	%	%	%	%
2018 MRE[1]									
Measured	213.6	35.1	2.3	5.6	36.6	64.5	3.6	2.0	6.8
Indicated	520.8	34.3	2.4	5.8	35.4	63.5	3.8	2.5	7.7
Total M&I	**734.4**	**34.6**	**2.4**	**5.7**	**35.7**	**63.8**	**3.8**	**2.4**	**7.4**
Inferred	237.0	34.1	2.1	5.8	34.6	64.0	3.8	2.1	8.9
2021 MRE[2]									
Measured	195.6	35.1	2.3	5.9	36.6	64.6	3.7	2.0	7.1
Indicated	526.3	34.5	2.4	6.1	35.5	63.9	3.9	2.5	8.2
Total M&I	**721.9**	**34.7**	**2.4**	**6.0**	**35.8**	**64.1**	**3.8**	**2.4**	**7.9**
Inferred	263.4	34.1	2.1	6.2	34.0	64.2	3.9	2.1	9.1

1. *Reported using a 20% Fe lower cut-off*
2. *Reported using a 10% WTREC lower cut-off*

To better understand the effect of the various modifications to the Mineral Resource since 2018, waterfall charts were prepared (Figure 14.36 and Figure 14.37). A slight gain in tonnage (1.1%) was observed in the Measured and Indicated blocks due to the corrections made to the surface blocks and other minor modifications; however, the main changes were related to mining depletion since the resumption of operations in 2019 and the new pit optimisation. Tonnage in the Inferred Category increased by 13% due to the deeper pit shell encompassing mainly Inferred blocks. Moving from a Fe% based cut-off to a WTREC% cut-off had a negligible effect on the tonnages.

The updated pit optimisation parameters use a new weight recovery calculation which is based on actual plant performance data since 2019 (rather than assumptions made in 2018), and a new mining exclusion perimeter has been imposed which limits the footprint of the pit mainly in the eastern portion of the deposit. A higher iron ore price has been selected to report the Mineral Resource, reflecting the increases observed in the market since 2018.

Figure 14.36: Waterfall Chart Showing Effects of the Modifications to the Mineral Resource since 2018 MRE for Measured and Indicated Categories

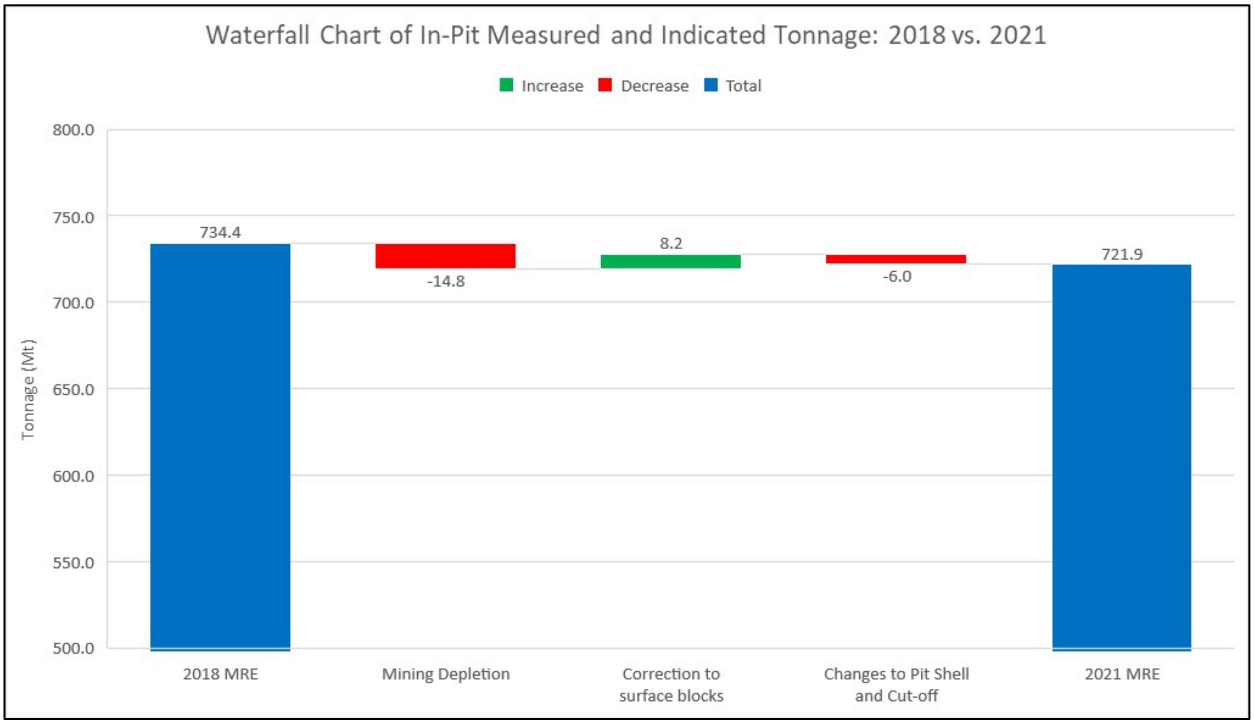
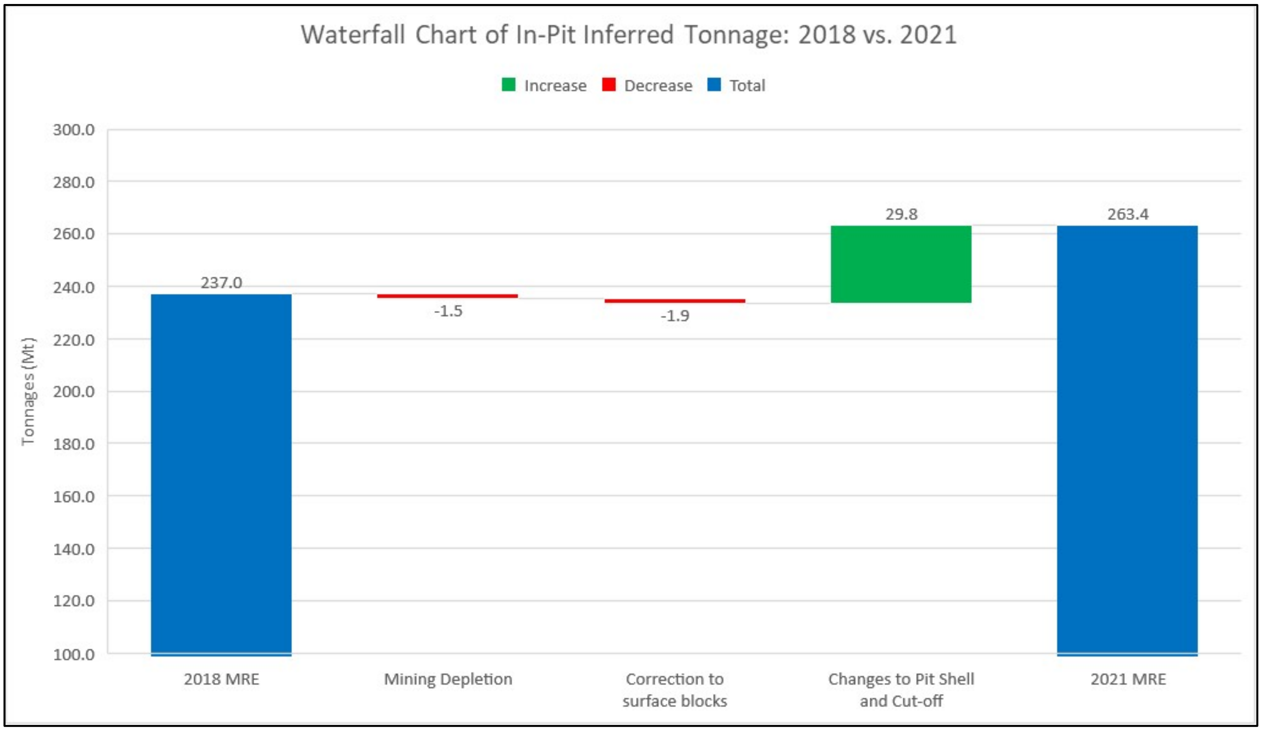


Figure 14.37: Waterfall Chart Showing Effects of the Modifications to the Mineral Resource since 2018 MRE for the Inferred Category



15 MINERAL RESERVE ESTIMATES

15.1 Summary

The Mineral Reserve for the Tacora Mine is estimated at 478.9 Mt at an average grade of 34.89% Fe and 2.62% Mn as summarized in Table 15.1. The Mineral Reserve estimate was prepared by GMS. The resource block model was also generated by G Mining Services Inc. ("GMS"). The mine design and Mineral Reserve Estimate have been completed to a level appropriate for feasibility studies. The Mineral Reserve Estimate stated herein is consistent with the CIM definitions and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated ("M&I") Mineral Resources and do not include any Inferred Mineral Resources. The Inferred Mineral Resources contained within the mine design are treated as waste.

Table 15.1: Tacora Mine Mineral Reserve Estimate (Effective Date of January 1st, 2021)

Mineral Reserves by Category		Proven	Probable	Stockpile	Proven & Probable
Crude Ore Tonnage	k dmt	**341,439**	**136,508**	**997**	**478,943**
Crude Iron Grade	% Fe	**34.85**	**34.97**	**38.41**	**34.89**
Crude Manganese Grade	% Mn	**2.72**	**2.35**	**5.31**	**2.62**
Concentrate Tonnage	k dmt	**113,577**	**45,478**	**369**	**159,425**
Concentrate Iron Grade	% Fe	**65.60**	**65.60**	**65.30**	**65.60**
Concentrate Manganese Grade	% Mn	**1.63**	**1.53**	**5.92**	**1.61**
Concentrate Silica Grade	% SiO_2	**3.06**	**3.22**	**3.00**	**3.11**
Total Weight Recovery	%	**33.26**	**33.32**	**37.02**	**33.29**
Total Fe Recovery	%	**62.62**	**62.49**	**62.94**	**62.59**

Notes on Mineral Reserves:

9. *The Mineral Reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards for Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council May 10th, 2014.*
10. *Mineral Reserves based on December 2020 depletion surface merged with an updated Lidar dated September 2017.*
11. *Mineral Reserves are estimated at a minimum of 20% Lab weight recovery for all sub-units except sub-unit 52 which is 30%. In addition, sub-unit 34 must have a ratio of weight recovery to iron of at least 1.*
12. *Mineral Reserves are estimated using a long-term iron price reference price (Platt's 62%) of USD 70/dmt and an exchange rate of 1.25 CAD/USD. An Fe concentrate price adjustment of USD 12/dmt was added as an iron grade premium net of a USD 5/dmt marketing charge.*
13. *Bulk density of ore is variable but averages 3.20 t/m^3.*
14. *The average strip ratio is 0.82:1.*
15. *The Mineral Reserve includes a 5.2% mining dilution and a 97% ore recovery.*
16. *The number of metric tonnes was rounded to the nearest thousand. Any discrepancies in the totals are due to rounding*

effects; rounding followed the recommendations in NI 43-101.

15.2 Resource Block Model

The resource block model was prepared by GMS in September 2017 where each lithological unit was modelled with precision. The block model framework information is presented in Table 15.2. Average block properties were calculated based on the weighting of the various units and a majority rock type was assigned (Table 15.3). In addition to the modelled iron grade, other interpolated attributes include manganese, magnetite, shaker (Wilfley) table weight recovery, shaker table iron grade, shaker table silica grade, shaker table manganese grade and shaker table magnetite grade. The shaker table results are used to predict plant performance and plant concentrate grades with metallurgical recovery equations provided by Soutex in Section 13.

Table 15.2: Block Model Framework

Model Setting	Value
X Origin	917
Y Origin	3,654
Z Origin	804
Block Size in X Direction	20
Block Size in Y Direction	20
Block Size in Z Direction	12
Number of Blocks in X Direction	350
Number of Blocks in Y Direction	240
Number of Blocks in Z Direction	85

Table 15.3: Weighted Average Block Model Attributes

Model Attributes	Model Abbreviation
Weighted Average Density (t/m^3)	DENS_WA
Weighted Average Iron Grade (%)	FE_WA
Weighted Average Manganese Grade (%)	MN_WA
Weighted Average Magnetite Content (%)	SAT_WA
Weighted Average Shaker Table Weight Recovery (%)	WTREC_WA
Weighted Average Shaker Table Iron (%)	FE_CONC_WA
Weighted Average Shaker Table Manganese (%)	MN_CONC_WA
Weighted Average Shaker Table Silica (%)	SI_CONC_WA

Weighted Average Magnetite (%)	SAT_CONC_WA
Resource Category	PASS_FINAL

The Wabush Iron-Formation comprises five members as presented in Figure 15.1 totaling more than 300 m in thickness through the center of the Scully Property. Four sub-units (21,23,41,61) have no iron content of economic interest. These sub-units, along with the overlying overburden, are treated as waste units.

Figure 15.1: Wabush Iron Formation Stratigraphic Column

Block Model Code	***Geological Formation***	***O/W***
Unit 11	**Overburden**	**Waste**
Sub-unit 21	**Upper Member**	**Waste**
Sub-unit 22		**Ore**
Sub-unit 23		**Waste**
Sub-unit 31	**Middle Member**	**Ore**
Sub-unit 32		**Ore**
Sub-unit 33		**Ore**
Sub-unit 34		**Ore**
Unit 41	**Middle Quartzite**	**Waste**
Sub-unit 51	**Lower Member**	**Ore**
Sub-unit 52		**Ore**
Sub-unit 53		**Ore**
Unit 61	**Basal Silicates**	**Waste**

The middle sub-unit of the Upper Member (sub-unit 22), the Middle Member and the Lower Member are potential iron ore mostly in its oxide form as hematite (Fe_2O_3) and to a lesser extent as magnetite (Fe_3O_4). The main gangue mineral in the iron formation is quartz or silica.

The weight recoveries in the ore units are variable and have been measured with a Wilfley table. To reflect the actual plant performance, recovery equations have been defined to estimate the final dry concentrate grades and weight recovery. These equations are presented in Section 13-Mineral Processing and Metallurgical Testing.

The objective is to produce a concentrate above 65.6% iron and a manganese grade and a silicate grade of respectively less than 2% and 3% respectively.

The magnetite content of the ore is tracked in the scheduling to limit the magnetite content to less than 8%. Ore with high magnetite content is generally harder and therefore reduces the concentrator throughput.

15.3 Pit Optimization

Open pit optimization was conducted to determine the optimal economic shape of the open pit to guide the pit design process. This task was undertaken using the Whittle software which is based on the Lerchs-Grossmann algorithm. The method works on a block model of the ore body, and progressively constructs lists of related blocks that should, or should not, be mined. The method uses the values of the blocks to define a pit outline that has the highest possible total economic value, subject to the required pit slopes defined as structure arcs in the software. This section describes all the parameters used to calculate block values in Whittle.

For this Feasibility Study, Measured and Indicated Mineral Resource blocks were considered for optimization purposes. However, sensitivities were run using the complete resource by including the Inferred Mineral Resource blocks.

The pit optimization process was prevented from encroaching into existing nearby lakes by enforcing a hard boundary constraint. A 120 m buffer was maintained with Little Wabush Lake on the east side and a 100 m buffer with Long Lake on the west side. The rail right of way east of East Pit is respected by preventing a pushback of the pit wall.

15.3.1 Pit Slope Geotechnical Assessment

Recommendations from the 2018 Feasibility Study were used as an input to the pit optimization and design process.

15.3.1.1 Rock Mass Characterization

The rock mass can generally be described as moderately strong (UCS in the range of 40 to 70 MPa) and of fair quality (GSI in the range of 30 to 60). It appears that no laboratory rock testing was carried out to confirm the intact rock strengths. The Piteau (2003) report indicates that field strength estimates were made on cores collected during the drilling of hydrogeological wells in 2002. However, those values were unavailable for review during the preparation of this Report. Those values should be reviewed to confirm the assumption used herein.

The rock mass is characterized by the deep penetrative weathering and oxidation of some iron bearing units, causing the degradation of the hematite minerals into limonite (waste) and, consequently, the reduction of the intact rock strength.

Large (site-scale) geological structures of the Wabush Deposit consist of folds, contact zones (which may exhibit poor rock mass quality or limonitic alteration) and bedding. A large sub-vertical, north-south trending fault separates the East and East-East Pit areas from the West, West Pit Extension and South Pit areas (i.e., at about 17000E). Based on differences in static water levels on either side of the fault, it is suggested that this fault acts as a groundwater barrier.

Bedding is dominant throughout the Wabush Deposit, which tends to follow the folding anticlines and synclines, and is a major control of the pit slope configuration. In addition to bedding, there are two major joint sets, labelled bedding joints and cross-joints. These joint sets provide release planes for planar failure along the bedding planes or raveling of rock blocks from the bench face. The cross-joints strike approximately orthogonal to bedding and dip steeply to sub-vertically, whereas the bedding joints strike approximately parallel to bedding and dip approximately normal to bedding.

15.3.1.2 Previous Mining Experience at Wabush

At the time of mine closure, five open pits were in operation: West Pit Extension, West Pit, South Pit, East Pit West, and East Pit East. The West Pit extension is the deepest pit, with a depth of 105 m below the Duley Lake level, located to the West.

No deep-seated rock mass failure was reported to have occurred on the slopes that can be up to 133 m high. However, three multi-bench slope failures were reported where bedding was undercut by the excavation; they were controlled and followed kinematically.

Kinematic control observations:

- Where bedding was steeply dipping (i.e., > 45°) out of the slope, it was observed that undercutting of bedding planes results in excessive back-break and the creation of unstable undercut blocks was observed.
- Where bedding was dipping less than about 30° to 35° out of the slope, it was observed that undercutting of bedding planes did not lead to widespread back-break of the bench faces. However,

undercutting slopes with bedding in this range of dip resulted in a planar failure where the un-benched slope with face at 65° was 35 m high. Slope water pressure on the basal failure plane was a potential contributor to the instability, (Golder 2012).

Most of the footwall benches and many of the hanging wall benches are now filled by debris and provide limited rockfall catchment area (Figure 15.2). It is inferred that the poor bench quality observed on site is related to operational procedures. Implementing improved procedures could yield better bench quality.

Figure 15.2: West Pit North Slope

15.3.1.3 Stability Evaluations

For preliminary analysis, a minimum offset of 50 m from the toe of a waste dump to the crest of a hanging wall slope with a height of 170 m and overall pit slope angles of 52° was kept. The results of limit equilibrium 2D analysis indicate that the waste dumps can be located at 50 m of the pit crest with no significant effect on pit slope stability. This analysis has not been optimized.

Rock pillars will be located between the lakes and the Boot Pit (to the west) and the East Pit East (to the east). Slope stability analysis of the pit shells provided by GMS yielded factors of safety ≥ 1.3 for assumed reasonable slope water conditions. Precedent experience at the operation indicates that active dewatering will be required for production.

Bi-linear failures were evaluated for moderately to steeply dipping bedding. Factors of safety above 1.3 were achieved for an assumed reasonable slope water condition that may require active dewatering to achieve.

Planar failure was evaluated for shallow dipping bedding between 25° and 35° with a bedding friction angle of 30° and a cohesion of 15 kN based on back analysis of stable and unstable slopes (Golder 2012). Analyses showed that:

- Where bedding dips less than 25°, an inter-ramp angle of 46° can be used for a slope up to 120 m high, for the cohesion that was assumed from the back-analysis and a range of slope water levels in a tension crack and along the basal failure plane.
- Where bedding dips between 25° and 35°, the bench face angle can undercut bedding for a 12 m high (single) bench. However, it becomes marginally stable for a double 24 m bench, particularly if the phreatic surface is assumed to be elevated, daylighting on the slope face.

15.3.1.4 Slope Design Criteria

There do not appear to be any deep-seated or overall stability concerns with the slopes exposed to date. However, the lack of catch-berms in most areas and the potential for bench scale instability that could be a hazard to operations personnel, indicate that modifications to the design and operating practices should be explored and evaluated.

Slope designs are based on the dip of the bedding relative to the wall orientation.

- The maximum bench face angle will be 65° except on footwall with steeper dips up to 70°.
- Catch-bench widths are based on Newfoundland and Labrador legislation (not less than 8 m) and the modified Ritchie formula (Ryan and Prior 2000) with additional width added for backbreak.
- For shallow dipping beds that are significantly flatter than the friction angle (Domain 1a), the achievable bench face angle is assumed limited to 65° due to assumed crest loss on bedding and observed slope conditions.
- For bedding in the range of the friction angle of the beds (25° to 44°), the inter-ramp angle shall range between 30° and 32° with single benching (12 m). In Domain 1b, the achieved bench face angles are limited to 65° due to back-break on bedding at the crest. For Domain 2 steeper bedding, the achieved bench face angle is limited to 50° due to back-break on bedding at the crest.

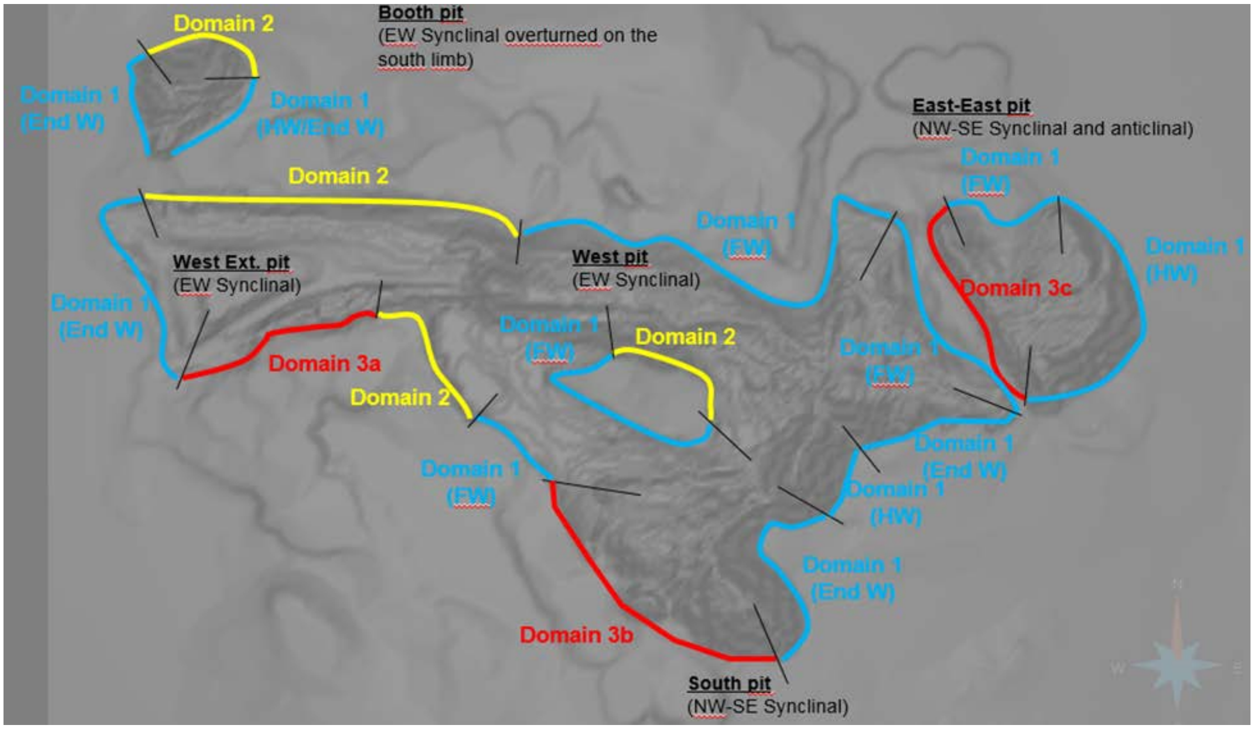
- In Domain 1b, the IRA is limited to 30° to control potential multi-bench planar failure on bedding triggered by the overcoming of the cohesion component of the bedding plane strength with increased slope height and exacerbated by slope water.
- In Domain 2, the IRA is limited to 32° to control potential multi-bench planar failure on bedding. This domain allows for uncertainty in the actual strength the bedding plane. It is like Domain 1b only for slightly steeper bedding.
- In Domain 3, the bedding is dipping steeper than its strength, and bench face angles are equal to the dip of the bed. Based on slope observations, achieved bench face angles can be steeper than 65°, up to 70°, where this conforms to bedding.
- End walls and hanging walls are assumed to behave like Domain 1a, based on slope observations.

The pit slope design criteria by slope domain is summarized in Table 15.4 and graphically in Figure 15.3.

Table 15.4: Pit Slope Design Criteria

Geotechnical Rock Slope Profiles	Bench Face Angle (deg.)	Bench Height (m)	Bench Width (m)	Inter-Ramp Angle (deg.)
Domain 1 & End Walls	65	24	12.0	46.0
Domain 2	50	12	9.1	32.0
Domain 3 Bedding Dip 45º	45	24	10.0	35.2
Domain 3 Bedding Dip 50º	50	24	10.0	38.5
Domain 3 Bedding Dip 55º	55	24	10.0	41.8
Domain 3 Bedding Dip 60º	60	24	10.0	45.2
Domain 3 Bedding Dip 65º	65	24	11.0	47.2
Domain 3 Bedding Dip 70º	70	24	11.0	50.6

Figure 15.3: Final Pit Slope Geotechnical Domains



Source: Golder Associates Ltd.

15.3.2 <u>Mining Dilution and Ore Loss</u>

A mining dilution assessment was performed by evaluating the number of waste contacts for ore blocks. The block contacts are then used to estimate a dilution skin around ore blocks representing the expected dilution during mining. The dilution skin consists of 1.5 m of material in a north-south direction (across strike) and 1.5 m in an east-west direction (along strike). The dilution is therefore specific to the geometry of the ore body and the number of contacts between ore and waste.

For each mineralized block in the resource model, a diluted grade and density are calculated by taking into account the grade and density of the surrounding blocks.

For the pit optimization in Whittle, final design and calculations of Mineral Reserves and the production schedule, the diluted block model was used. This dilution factor included in the Mineral Reserves was estimated at 5.2% and a 3% ore loss factor was applied.

15.3.3 <u>Pit Optimization Parameters</u>

A summary of the pit optimization parameters is presented in Table 15.5. The parameters correspond to the long-term operating revenues and costs as reflected in the final production schedules and economic evaluation. A reference iron ore price (Platt's 62% Fe CFR China) of USD 70/t of concentrate and an exchange rate of CAD/USD 1.25. A net price adjustment of USD 12/t was applied (USD 17/t for iron premium less USD 5/t for marketing).

The plant weight recovery is based on actual plant performance. The plant weight recovery is adjusted for a target grade of 65.6% iron in concentrate. Subsequently, developed weight recovery and concentrate grade relationships were established and used to develop the final production schedule.

The total logistics cost, including rail transport from mine to Pointe-Noire port, port handling and ocean freight to the clients, is estimated at USD 32.75/t. All royalties are estimated at USD 5.06/t. These costs are deducted to estimate an FOB Scully site concentrate price of USD 44.03/t.

Unit reference mining costs are used for a "reference mining block" located near the pit crest or surface and are incremented with depth which corresponds to the additional cycle time and resulting incremental hauling cost. The reference waste mining cost is estimated at USD 2.10/t with an incremental depth factor of USD 0.02/t per 12 m bench. The reference ore mining cost is slightly different for the various pit sectors due to the distance to the primary crusher. The reference ore mining cost ranges from USD 2.25/t to USD 2.34/t.

The overall slope angles utilized in Whittle are based on the inter-ramp angles recommended from Golder Associate's pit slope study with provisions for ramps and geotechnical berms. The overall slope angles range from 28 to 42 degrees.

Table 15.5: Pit Optimization Parameters

Pit Optimization Parameters - Mineral Reserves		
Mining Dilution (included in block model)	%	0
Mining Recovery	%	97
Weight Recovery (average)	%	33.29
Revenues		
Concentrate Fe Grade	% Fe	65.60
Concentrate Moisture Content	%	2.00
Reference Price (IODEX 62% FE CFR China)	USD/dmt con.	70.00
65.3% Fe Premium	USD/dmt con.	15.00
Other Premium	USD/dmt con.	2.00
Cargill Marketing	USD/dmt con.	-5.00
Revenue (CFR Client)	**USD/dmt con.**	**82.00**
Freight	USD/dmt con.	-17.07
Net Revenue (FOB Pointe Noire)	USD/dmt con.	64.93
Additional Cost for Concentrate Losses (1%)	USD/dmt con.	-0.16
Rail Transportation (mine to Sept-Iles Port)	USD/wmt con.	-9.85
Railcar Maintenance	USD/wmt con.	-0.27
Port SFPPN	USD/wmt con.	-5.56
Royalties - MFC	USD/dmt con.	-4.22
Royalties - Others	USD/dmt con.	-0.84
Net Revenue (FOB Scully)	**USD/dmt con.**	44.03
Exchange Rate	CAD/USD	1.25
Net Revenue (FOB Scully)	CAD/dmt con.	55.04
Ore Based Costs		
Processing Cost	USD/dmt ore	3.82
G&A Costs	USD/dmt ore	0.74
Tailings Sustaining Capital	USD/dmt ore	0.27
Rehabilitation and Closure Cost	USD/dmt ore	0.20
Total Ore Based Cost	**USD/dmt ore**	5.03
Mining Costs & Parameters		
Incremental Bench Cost	USD/t/12 m	0.020
Reference Elevation	RL	600
Waste		
All Pit	USD/dmt waste	2.10
Ore		
Boot Pit	USD/dmt ore	2.34
West Pit Extension	USD/dmt ore	2.29
West Pit, South Pit, East Pit West and East Pit East	USD/dmt ore	2.25

15.3.4 Open Pit Optimization Results

The Whittle nested shell results are presented in Table 15.7 using only the Measured and Indicated ("M&I") resource and a minimum Lab Weight Recovery of 20%. Table 15.8 presents the Whittle resources including the Inferred ("MII"). The MII optimization results are generated for resource reporting purposes at a minimum Lab Weight Recovery of 10%. The nested shells are generated by using revenue factors to scale up and down from the base case selling price.

The minimum Lab Weight Recovery of 10% for MII Resources was established on the basis that a stockpiled marginal ore would generate sufficient concentrate to cover its marginal cash cost. The minimum Lab Weight Recovery of 20% for M&I Resource was selected to generate sufficient concentrate to cover the full cash cost per tonne of ore required to produce this concentrate. Detailed calculations are presented in Table 15.6.

Table 15.6: Detailed Calculations for Minimum Lab Weight Recovery

	Concentrate (USD/t)	Concentrate (USD/t)
Revenue CFR Client	93.50	82.00
Net Revenue – Pointe Noire	75.65	64.28
Net Revenue – Scully	54.01	43.54
Recoveries	**%**	**%**
Laboratory Weight Recovery (Wilfley Table)	10	20
Wet Weight Recovery	9.7	19.4
Total Weight Recovery	8.9	17.8
	Ore (USD/t)	**Ore (USD/t)**
Revenue – 10% WREC	4.81	-
Revenue – 20% WREC	-	7.75
	Marginal Cash Cost	**Full Cash Cost**
	(USD/t ore)	USD/t ore)
Mining	-	2.13
Processing	3.82	3.82
G&A Costs	0.27	0.74
Sustaining Capital	0.27	0.70
Closure	-	0.20
Rehandling	0.46	-
Total	**4.82**	**7.59**

Note: Net Revenue Pointe Noire include 1% Loss of Concentrate

Table 15.7: M&I Whittle Shell Results

Pit Shell	Rev. Factor	Iron Price	Total (kt)	Ore (kt)	Strip Ratio	Fe %	Wt Rec %	Conc. (kt)
1	0.30	16.15	2,104	1,847	0.14	40.14	49.55	915
2	0.32	17.23	4,943	4,357	0.13	39.02	47.35	2,063
3	0.34	18.30	10,951	9,577	0.14	37.92	45.15	4,324
4	0.36	19.38	25,121	21,957	0.14	37.11	42.62	9,359
5	0.38	20.46	53,180	44,678	0.19	36.62	40.77	18,217
6	0.40	21.53	95,371	78,572	0.21	36.10	39.28	30,867
7	0.42	22.61	171,770	137,860	0.25	35.70	37.82	52,133
8	0.44	23.69	270,759	211,976	0.28	35.34	36.70	77,789
9	0.46	24.76	357,288	271,150	0.32	35.25	36.03	97,688
10	0.48	25.84	441,256	321,344	0.37	35.19	35.59	114,378
11	0.50	26.92	521,110	363,524	0.43	35.16	35.27	128,199
12	0.52	27.99	623,119	415,662	0.50	35.13	34.82	144,741
13	0.54	29.07	688,238	450,438	0.53	35.11	34.49	155,348
14	0.56	30.15	734,474	473,081	0.55	35.06	34.26	162,098
15	0.58	31.22	784,352	494,709	0.59	35.02	34.06	168,521
16	0.60	32.30	806,779	504,674	0.60	35.00	33.96	171,372
17	0.62	33.38	853,293	522,373	0.63	34.96	33.80	176,555
18	0.64	34.45	894,732	536,815	0.67	34.94	33.67	180,769
19	0.66	35.53	917,579	544,423	0.69	34.93	33.60	182,949
20	0.68	36.61	938,848	550,547	0.71	34.93	33.56	184,745
21	0.70	37.68	951,793	554,525	0.72	34.92	33.52	185,856
22	0.72	38.76	980,153	562,645	0.74	34.90	33.43	188,099
23	0.74	39.84	989,892	564,918	0.75	34.89	33.42	188,771
24	0.76	40.91	995,964	566,565	0.76	34.89	33.40	189,212
25	0.78	41.99	1,017,133	570,596	0.78	34.88	33.38	190,447
26	0.80	43.07	1,027,808	572,657	0.79	34.87	33.36	191,063
27	0.82	44.14	1,035,574	574,114	0.80	34.87	33.35	191,492
28	0.84	45.22	1,046,597	576,027	0.82	34.86	33.34	192,062
29	0.86	46.30	1,048,850	576,496	0.82	34.86	33.34	192,188
30	0.88	47.37	1,053,609	577,362	0.82	34.86	33.33	192,430
31	0.90	48.45	1,057,998	578,055	0.83	34.86	33.32	192,635
32	0.92	49.53	1,062,300	578,706	0.84	34.86	33.32	192,828
33	0.94	50.61	1,066,334	579,317	0.84	34.86	33.32	193,005
34	0.96	51.68	1,072,525	580,148	0.85	34.86	33.31	193,256
35	0.98	52.76	1,075,972	580,670	0.85	34.86	33.31	193,400
36	1.00	53.84	1,077,284	580,849	0.85	34.86	33.31	193,452
37	1.02	54.91	1,078,090	580,946	0.86	34.86	33.30	193,482
38	1.04	55.99	1,078,640	581,061	0.86	34.86	33.30	193,507
39	1.06	57.07	1,079,652	581,192	0.86	34.86	33.30	193,544
40	1.08	58.14	1,080,342	581,268	0.86	34.86	33.30	193,568
41	1.10	59.22	1,080,842	581,330	0.86	34.86	33.30	193,584
42	1.12	60.30	1,081,227	581,371	0.86	34.86	33.30	193,597
43	1.14	61.37	1,081,998	581,451	0.86	34.86	33.30	193,621
44	1.16	62.45	1,082,250	581,479	0.86	34.86	33.30	193,629
45	1.18	63.53	1,084,750	581,706	0.86	34.86	33.30	193,701

**Concentrate calculated at 65.6% Fe concentrate grade*

Table 15.8: MII Whittle Shell Results

Pit Shell	Rev. Factor	Iron Price	Total (kt)	Ore (kt)	Strip Ratio	Fe %	Wt Rec %	Conc. (kt)
1	0.30	16.15	41,597	35,509	0.17	36.94	41.74	14,821
2	0.32	17.23	99,175	80,649	0.23	36.26	39.69	32,008
3	0.34	18.30	199,997	158,873	0.26	35.73	37.85	60,125
4	0.36	19.38	389,782	294,695	0.32	35.31	36.37	107,170
5	0.38	20.46	521,623	382,530	0.36	35.17	35.61	136,230
6	0.40	21.53	695,901	488,888	0.42	35.01	34.84	170,308
7	0.42	22.61	828,462	567,827	0.46	34.93	34.28	194,632
8	0.44	23.69	930,535	626,771	0.48	34.88	33.84	212,117
9	0.46	24.76	1,068,011	699,442	0.53	34.74	33.32	233,045
10	0.48	25.84	1,148,963	741,810	0.55	34.67	33.00	244,784
11	0.50	26.92	1,229,123	782,602	0.57	34.60	32.67	255,672
12	0.52	27.99	1,264,755	799,925	0.58	34.58	32.53	260,183
13	0.54	29.07	1,317,240	817,788	0.61	34.53	32.43	265,210
14	0.56	30.15	1,332,709	825,619	0.61	34.52	32.35	267,084
15	0.58	31.22	1,372,682	840,729	0.63	34.50	32.22	270,843
16	0.60	32.30	1,383,822	845,614	0.64	34.49	32.16	271,978
17	0.62	33.38	1,397,247	850,013	0.64	34.48	32.12	273,065
18	0.64	34.45	1,404,377	852,904	0.65	34.48	32.09	273,711
19	0.66	35.53	1,412,778	855,441	0.65	34.47	32.07	274,321
20	0.68	36.61	1,417,649	857,395	0.65	34.47	32.04	274,732
21	0.70	37.68	1,424,969	859,154	0.66	34.46	32.03	275,166
22	0.72	38.76	1,428,545	860,514	0.66	34.46	32.01	275,442
23	0.74	39.84	1,431,325	861,461	0.66	34.46	32.00	275,634
24	0.76	40.91	1,432,669	862,100	0.66	34.46	31.99	275,748
25	0.78	41.99	1,435,879	862,876	0.66	34.46	31.98	275,920
26	0.80	43.07	1,441,884	863,835	0.67	34.45	31.97	276,177
27	0.82	44.14	1,444,073	864,380	0.67	34.45	31.96	276,289
28	0.84	45.22	1,445,746	864,785	0.67	34.45	31.96	276,373
29	0.86	46.30	1,450,502	865,471	0.68	34.45	31.95	276,554
30	0.88	47.37	1,455,727	866,100	0.68	34.45	31.95	276,733
31	0.90	48.45	1,456,829	866,341	0.68	34.45	31.95	276,780
32	0.92	49.53	1,458,020	866,594	0.68	34.45	31.94	276,832
33	0.94	50.61	1,459,040	866,881	0.68	34.45	31.94	276,881
34	0.96	51.68	1,459,111	867,024	0.68	34.45	31.94	276,896
35	0.98	52.76	1,461,158	867,342	0.68	34.45	31.93	276,967
36	1.00	53.84	1,463,130	867,712	0.69	34.45	31.93	277,039
37	1.02	54.91	1,465,901	868,019	0.69	34.45	31.93	277,118
38	1.04	55.99	1,465,968	868,091	0.69	34.45	31.92	277,127
39	1.06	57.07	1,466,540	868,186	0.69	34.45	31.92	277,145
40	1.08	58.14	1,467,357	868,258	0.69	34.45	31.92	277,165
41	1.10	59.22	1,468,138	868,317	0.69	34.45	31.92	277,184
42	1.12	60.30	1,468,556	868,361	0.69	34.45	31.92	277,195
43	1.14	61.37	1,470,685	868,580	0.69	34.45	31.92	277,248
44	1.16	62.45	1,471,250	868,648	0.69	34.45	31.92	277,263
45	1.18	63.53	1,471,985	868,758	0.69	34.45	31.92	277,283

**Concentrate calculated at 65.6% Fe concentrate grade*

Pit by pit results are generated with the nested shells based on three mining approaches to estimate the net present value ("NPV") of operating cash flow discounted at 8%. A schematic of these approaches is presented in Figure 15.4.

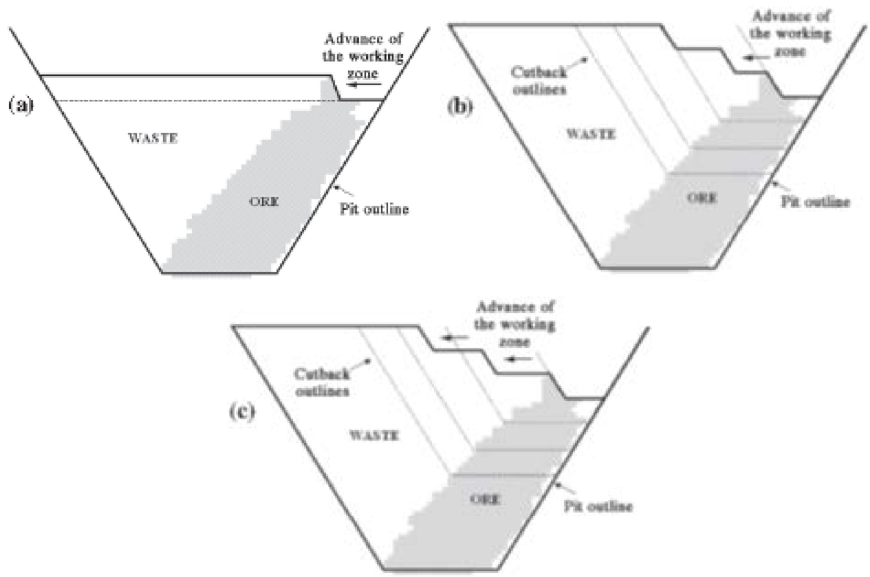
The worst case is based on mining a given pit shell bench by bench from the top down. It is referred to as the "worst case" as it produces the lowest NPV; however, it has the advantage of almost always being practical.

The best case is based on mining each nested shell one by one up to the final shell. This is referred to as the "best case" because it produces the highest NPV. It is almost never practical but provides a theoretical maximum value to aim for through practical phasing approaches. The difference with the worst case gives an indication of the value to be generated by phasing the pit.

The specified case is generated by imposing a more practical phasing approach and provides a more realistic target and phasing evaluation for the pit. In the present case, a two-phase approach was evaluated. The pushback chooser module in Whittle evaluates which combination of nested shells can produce the highest NPV.

Pit shell 17 is selected from the M&I optimizations as the optimum final pit shell which corresponds to a USD 33.38/t pit shell (Revenue Factor 0.62). This shell selection aims to maximize the discounted operating cash flow of the project.

Figure 15.4: Schematic Representation of Three Mining Schedule Configurations:
(a) Worst-Case Mining Schedule (b) Best-Case Mining Schedule (c) Intermediate Mining Schedule



Source: Whittle

Table 15.9: M&I Pit by Pit Results

Pit Shell	Best Case Disc. @ 8% (M USD)	Specified Disc. @ 8% (M USD)	Worst Case Disc. @ 8% (M USD)	Total Tonnage (kt)	Ore Tonnage (kt)	Strip Ratio (W:O)	Fe Grade %	Weight Rec %	Conc. (kt)
1	25	25	25	2,104	1,642	0.28	40.02	49.50	813
2	54	54	54	4,943	3,826	0.29	39.02	47.41	1,814
3	110	110	110	10,951	8,604	0.27	37.86	45.02	3,874
4	227	229	227	25,121	20,182	0.24	37.04	42.42	8,560
5	420	420	419	53,180	42,079	0.26	36.53	40.46	17,027
6	657	655	650	95,371	74,550	0.28	35.99	39.01	29,081
7	978	973	957	171,770	131,569	0.31	35.61	37.58	49,448
8	1,253	1,242	1,207	270,759	200,458	0.35	35.30	36.57	73,303
9	1,412	1,394	1,345	357,288	255,825	0.40	35.23	35.93	91,910
10	1,516	1,493	1,429	441,256	303,229	0.46	35.17	35.50	107,652
11	1,585	1,556	1,479	521,110	343,914	0.52	35.15	35.17	120,970
12	1,650	1,609	1,513	623,119	395,432	0.58	35.11	34.69	137,182
13	1,683	1,635	1,523	688,238	428,038	0.61	35.08	34.38	147,169
14	1,700	1,648	1,527	734,474	449,158	0.64	35.04	34.18	153,524
15	1,714	1,656	1,530	784,352	469,663	0.67	35.01	33.99	159,661
16	1,720	1,658	1,529	806,779	478,917	0.68	34.99	33.90	162,357
17	1,729	1,660	1,526	853,293	495,525	0.72	34.95	33.76	167,272
18	1,735	1,658	1,522	894,732	509,039	0.76	34.93	33.65	171,268
19	1,738	1,656	1,516	917,579	516,094	0.78	34.92	33.58	173,303
20	1,740	1,654	1,507	938,848	521,911	0.80	34.92	33.53	175,022
21	1,741	1,652	1,501	951,793	525,811	0.81	34.91	33.49	176,109
22	1,743	1,650	1,494	980,153	533,890	0.84	34.89	33.40	178,340
23	1,744	1,649	1,488	989,892	536,113	0.85	34.88	33.39	178,999
24	1,744	1,647	1,485	995,964	537,733	0.85	34.88	33.37	179,435
25	1,745	1,643	1,470	1,017,133	541,590	0.88	34.87	33.35	180,618
26	1,745	1,642	1,465	1,027,808	543,591	0.89	34.86	33.34	181,216
27	1,746	1,641	1,460	1,035,574	544,933	0.90	34.86	33.33	181,608
28	1,746	1,639	1,454	1,046,597	546,770	0.91	34.85	33.31	182,155
29	1,746	1,639	1,453	1,048,850	547,224	0.92	34.85	33.31	182,275
30	1,746	1,639	1,451	1,053,609	548,066	0.92	34.85	33.30	182,511
31	1,746	1,638	1,449	1,057,998	548,717	0.93	34.85	33.30	182,703
32	1,746	1,637	1,445	1,062,300	549,321	0.93	34.85	33.29	182,879
33	1,746	1,636	1,442	1,066,334	549,886	0.94	34.85	33.29	183,043
34	1,746	1,636	1,440	1,072,525	550,716	0.95	34.85	33.28	183,294
35	1,746	1,635	1,437	1,075,972	551,225	0.95	34.85	33.28	183,435
36	1,746	1,635	1,437	1,077,284	551,404	0.95	34.85	33.28	183,488
37	1,746	1,635	1,437	1,078,090	551,501	0.95	34.85	33.28	183,517
38	1,746	1,635	1,436	1,078,640	551,617	0.96	34.85	33.27	183,542
39	1,746	1,634	1,435	1,079,652	551,734	0.96	34.85	33.27	183,576
40	1,746	1,634	1,434	1,080,342	551,811	0.96	34.85	33.27	183,599
41	1,746	1,634	1,434	1,080,842	551,872	0.96	34.85	33.27	183,616
42	1,746	1,634	1,433	1,081,227	551,914	0.96	34.85	33.27	183,628
43	1,746	1,634	1,432	1,081,998	551,993	0.96	34.85	33.27	183,652
44	1,746	1,634	1,432	1,082,250	552,008	0.96	34.85	33.27	183,657
45	1,746	1,633	1,431	1,084,750	552,222	0.96	34.85	33.27	183,725

Table 15.10: MII Pit by Pit Results

Pit Shell	Best Case Disc. @ 8% (M USD)	Specified Disc. @ 8% (M USD)	Worst Case Disc. @ 8% (M USD)	Total Tonnage (kt)	Ore Tonnage (kt)	Strip Ratio (W:O)	Fe Grade %	Weight Rec %	Conc. (kt)
1	482	482	482	41,597	34,049	0.22	36.77	41.09	13,989
2	947	943	940	99,175	78,338	0.27	36.09	39.12	30,642
3	1,522	1,512	1,488	199,997	157,011	0.27	35.51	37.19	58,394
4	2,103	2,085	2,013	389,782	291,386	0.34	35.10	35.67	103,949
5	2,314	2,279	2,185	521,623	375,180	0.39	35.02	35.02	131,382
6	2,472	2,424	2,298	695,901	474,983	0.47	34.91	34.38	163,280
7	2,547	2,484	2,323	828,462	549,517	0.51	34.86	33.88	186,169
8	2,586	2,512	2,321	930,535	604,226	0.54	34.83	33.49	202,384
9	2,620	2,532	2,310	1,068,011	675,187	0.58	34.68	32.96	222,526
10	2,634	2,540	2,310	1,148,963	714,499	0.61	34.62	32.68	233,500
11	2,645	2,542	2,306	1,229,123	752,173	0.63	34.56	32.39	243,657
12	2,648	2,542	2,302	1,264,755	767,384	0.65	34.54	32.28	247,712
13	2,651	2,540	2,296	1,317,240	782,934	0.68	34.51	32.21	252,214
14	2,652	2,538	2,292	1,332,709	789,160	0.69	34.49	32.16	253,785
15	2,655	2,535	2,284	1,372,682	803,304	0.71	34.48	32.04	257,354
16	2,655	2,534	2,279	1,383,822	807,108	0.71	34.48	32.00	258,302
17	2,656	2,531	2,275	1,397,247	810,468	0.72	34.47	31.98	259,201
18	2,656	2,530	2,271	1,404,377	812,522	0.73	34.46	31.96	259,705
19	2,656	2,528	2,267	1,412,778	814,446	0.73	34.46	31.95	260,212
20	2,656	2,527	2,265	1,417,649	815,825	0.74	34.46	31.94	260,538
21	2,656	2,526	2,261	1,424,969	817,140	0.74	34.45	31.93	260,896
22	2,656	2,525	2,259	1,428,545	817,957	0.75	34.45	31.92	261,092
23	2,656	2,524	2,258	1,431,325	818,542	0.75	34.45	31.91	261,235
24	2,656	2,524	2,257	1,432,669	818,926	0.75	34.45	31.91	261,315
25	2,656	2,523	2,256	1,435,879	819,471	0.75	34.45	31.91	261,457
26	2,656	2,522	2,250	1,441,884	820,211	0.76	34.45	31.90	261,687
27	2,656	2,522	2,249	1,444,073	820,613	0.76	34.45	31.90	261,784
28	2,656	2,521	2,247	1,445,746	820,948	0.76	34.45	31.90	261,859
29	2,657	2,520	2,245	1,450,502	821,427	0.77	34.45	31.90	262,006
30	2,657	2,519	2,241	1,455,727	821,909	0.77	34.45	31.90	262,152
31	2,657	2,519	2,241	1,456,829	822,050	0.77	34.45	31.89	262,190
32	2,657	2,519	2,240	1,458,020	822,198	0.77	34.45	31.89	262,229
33	2,657	2,519	2,239	1,459,040	822,425	0.77	34.44	31.89	262,273
34	2,657	2,519	2,239	1,459,111	822,453	0.77	34.44	31.89	262,277
35	2,657	2,518	2,238	1,461,158	822,675	0.78	34.44	31.89	262,338
36	2,657	2,518	2,236	1,463,130	822,931	0.78	34.44	31.89	262,395
37	2,657	2,517	2,234	1,465,901	823,148	0.78	34.44	31.89	262,463
38	2,657	2,517	2,233	1,465,968	823,177	0.78	34.44	31.88	262,467
39	2,657	2,517	2,233	1,466,540	823,232	0.78	34.44	31.88	262,479
40	2,657	2,517	2,232	1,467,357	823,304	0.78	34.44	31.88	262,500
41	2,657	2,517	2,232	1,468,138	823,362	0.78	34.44	31.88	262,517
42	2,657	2,517	2,232	1,468,556	823,407	0.78	34.44	31.88	262,528
43	2,656	2,516	2,231	1,470,685	823,626	0.79	34.44	31.88	262,582
44	2,656	2,516	2,230	1,471,250	823,693	0.79	34.44	31.88	262,597
45	2,656	2,516	2,230	1,471,985	823,803	0.79	34.44	31.88	262,617

Figure 15.5: M&I Pit by Pit Graph

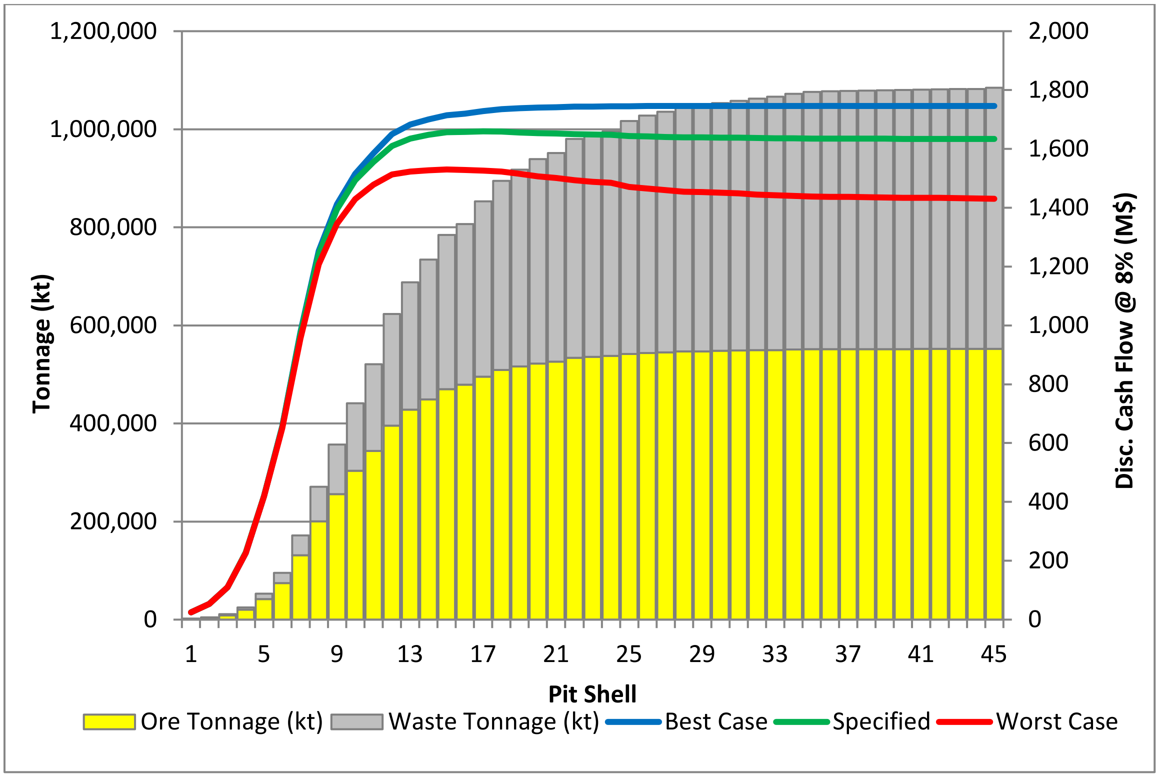
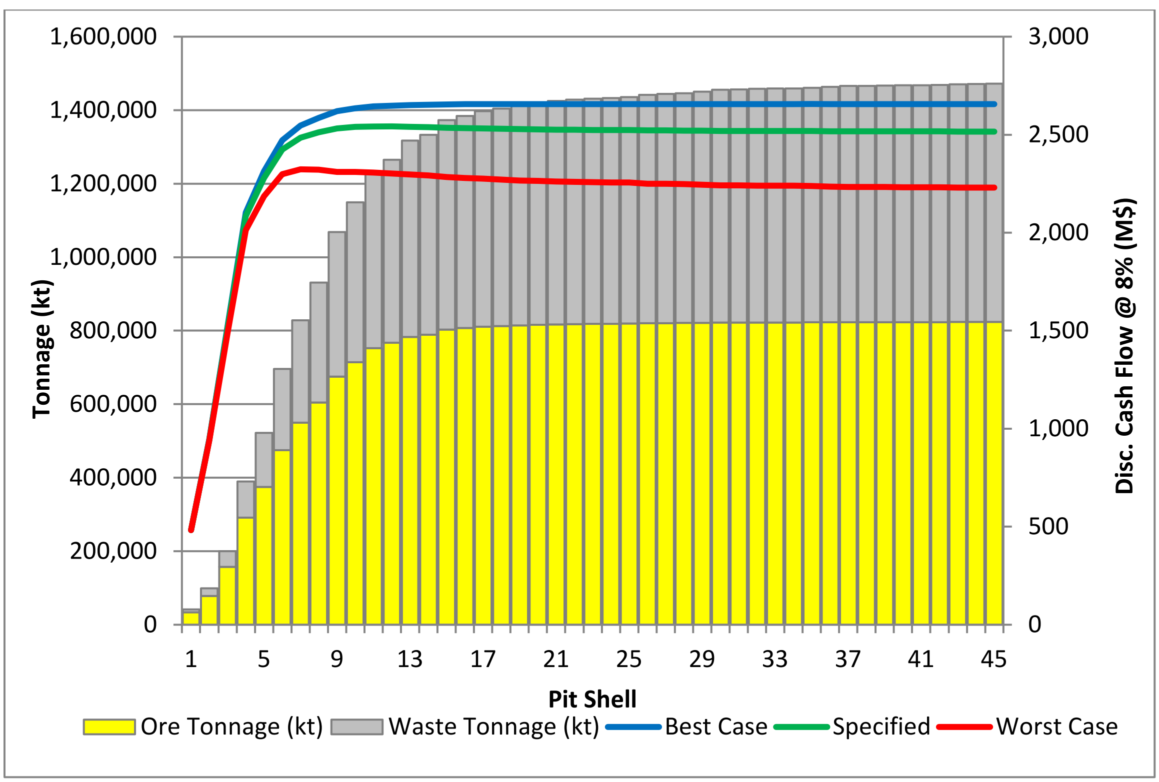


Figure 15.6: MII Pit by Pit Graph



15.4 Mine Design

15.4.1 Slope Design Criteria

The final pit was designed using a single bench configuration for Domain 3 wall and double benching configuration for other domains to a final height of 24 m. The pit slope profile is based on recommendations by Golder as presented in Section 15.3.1. The slope profile is based on recommended batter angles with catch bench width recommendations for an inter-ramp angle ranging from 32 to 46 degrees. A 20 m geotechnical berm is introduced every 120 m or four double-benches where ramp segments do not pass in the slope to reduce the vertical stack height.

Most of the open pit surface has already been stripped from past operations but some overburden tonnage remains to be mined. Some portions of the waste rock dumps are mined to allow for the pushbacks and is reported as overburden.

15.4.2 Ramp Design Criteria

The ramps and haul roads are designed for the largest equipment being 240-ton (211 tonne) class haul trucks with an operating width of 7.3 m. For double lane traffic, industry best-practice is to design a haul road of at least 3.5 times the width of the largest vehicle, in this case, at least 25 m. Ramp gradients are designed at 10%.

A shoulder barrier or safety berm on the outside edge will be constructed of crushed rock to a height equal to the rolling radius of the largest tire using the ramp. The rolling radius of the truck tire is 1.8 m. These shoulder barriers are required wherever a drop-off greater than 3 m exists and will be designed at 1.1H:1V. A ditch planned on the highwall will capture run-off from the pit wall surface and manage snow removal. The ditch will be 3.7 m wide on the highwall side and 2.7 m wide on the berm side; to facilitate drainage of the roadway, a 2% cross slope on the ramp is planned.

The double lane ramp width is 36.6 m wide (Figure 15.7) and the single lane ramp is 19.7 m wide (Figure 15.8). Single lane ramps are introduced in the pit bottom when the benches start narrowing and when the mining rates will be significantly reduced.

Figure 15.7: Double Lane Ramp Profile Design

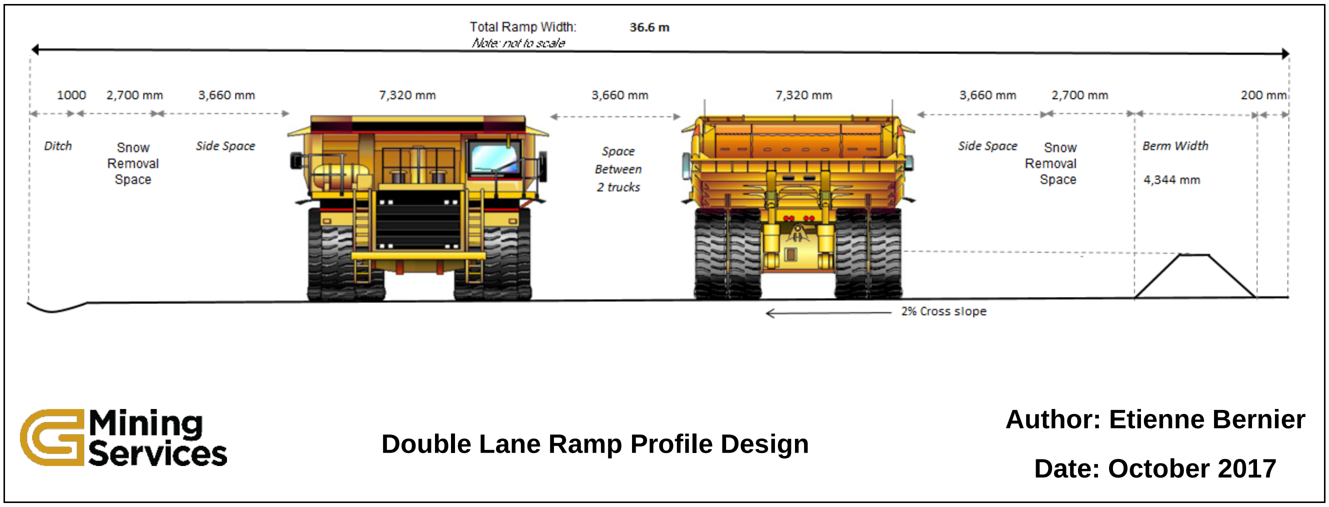


Figure 15.8: Single Lane Ramp Profile Design

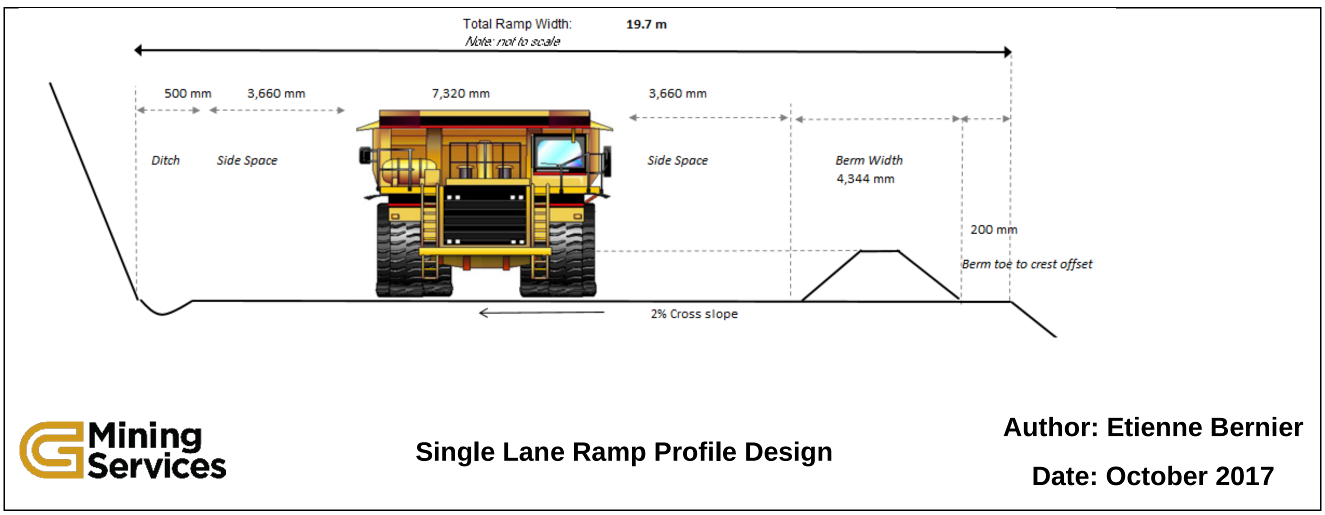
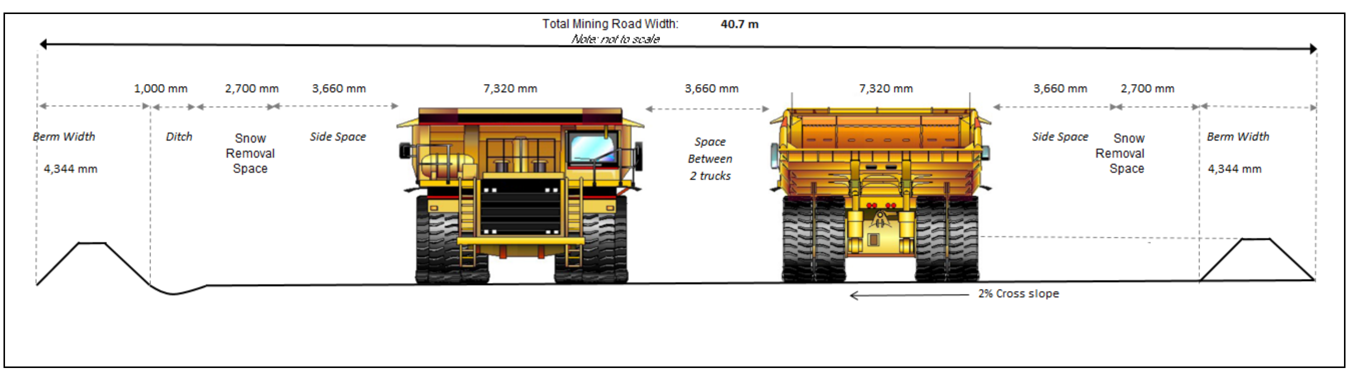


Figure 15.9: Mine Haulage Roads



Mining Services	**Mine Haulage Roads**	**Author: Etienne Bernier** **Date: October 2017**

15.4.3 Open Pit Mine Design Results

The starting status map is presented in Figure 15.10 and the final pit design in Figure 15.11. The final pit is divided into pit sectors according to historical designations. These sectors are subdivided as separate mining phases for subsequent scheduling.

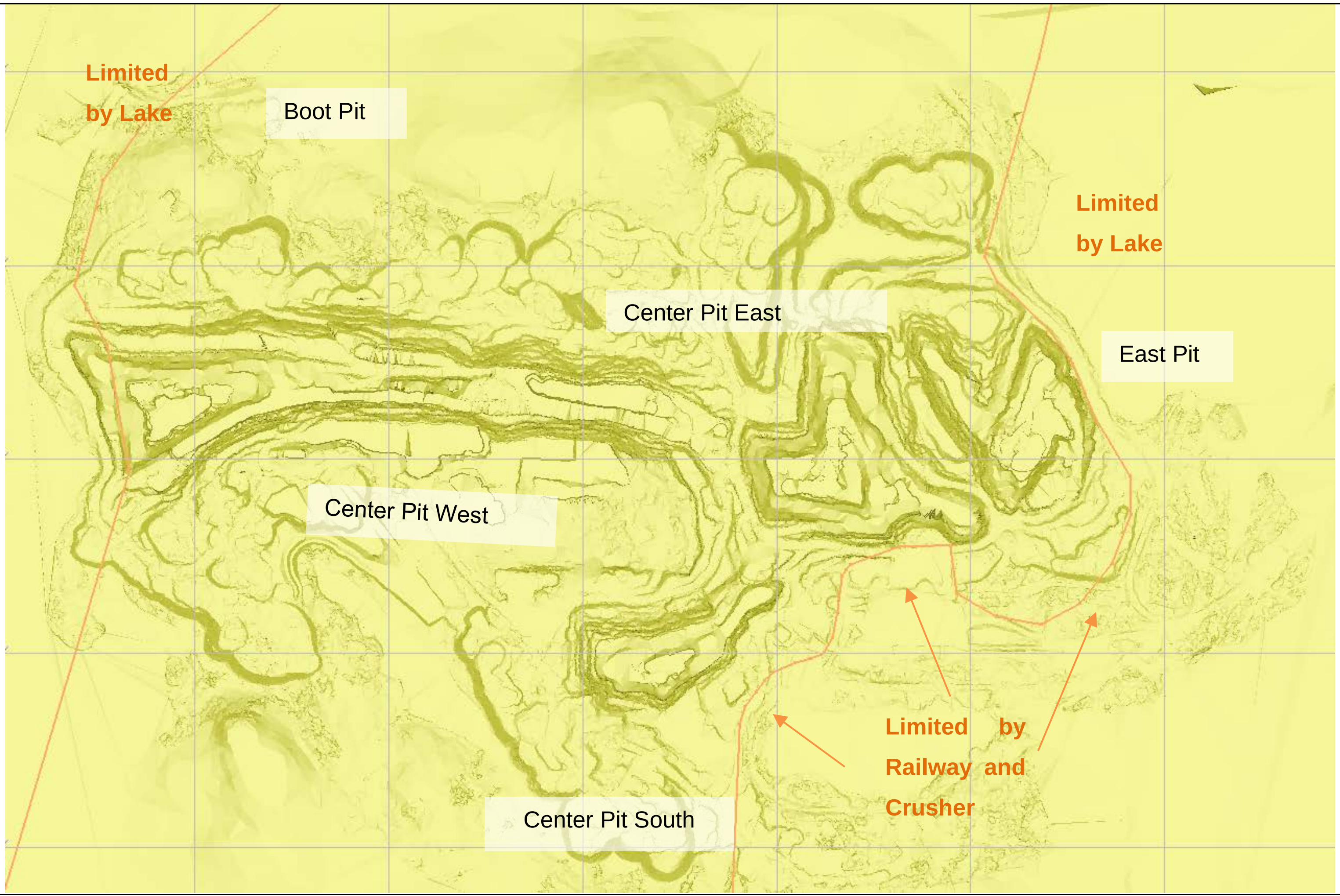
The pit sectors include the following:

- Boot Pit
- Center Pit West
- Center Pit South
- Center Pit East
- East Pit

The final pit layout was designed with the current pit status map taken into consideration with respect to ramp entrances and road networks in order to keep the primary crusher accessible as well as access to waste dump locations. The pushbacks to existing pits allowed for a mining width of 80 m with some narrower areas at 60 m.

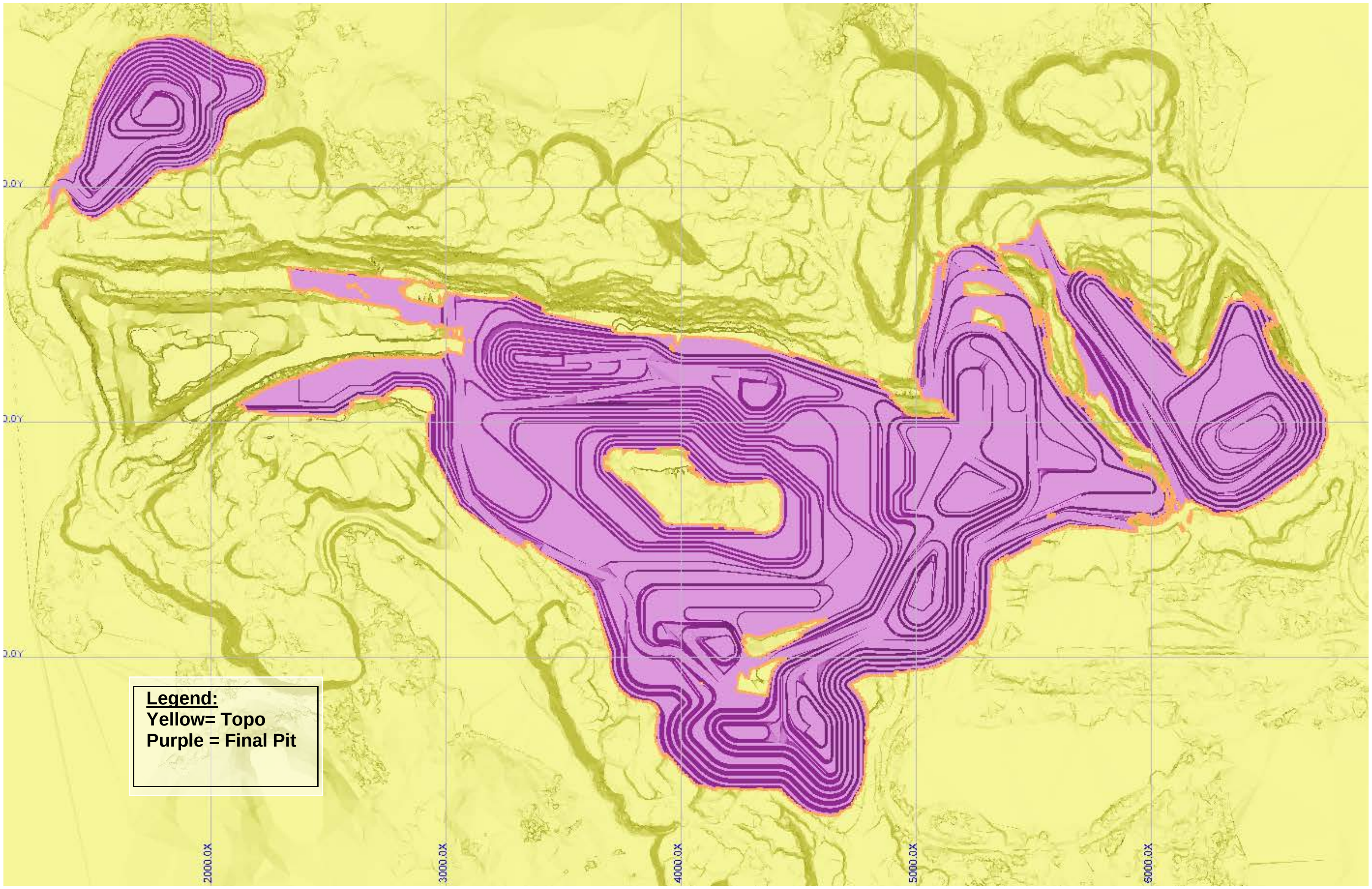
The final pit design closely follows the guiding Whittle shell as presented in Figure 15.12.

Figure 15.10: Tacora Mine Status Map & Lidar



Mining Services	**Tacora Mine Status Map & Lidar and Pit Limit**	**Author: Antoine Champagne** **Date: March 2021**

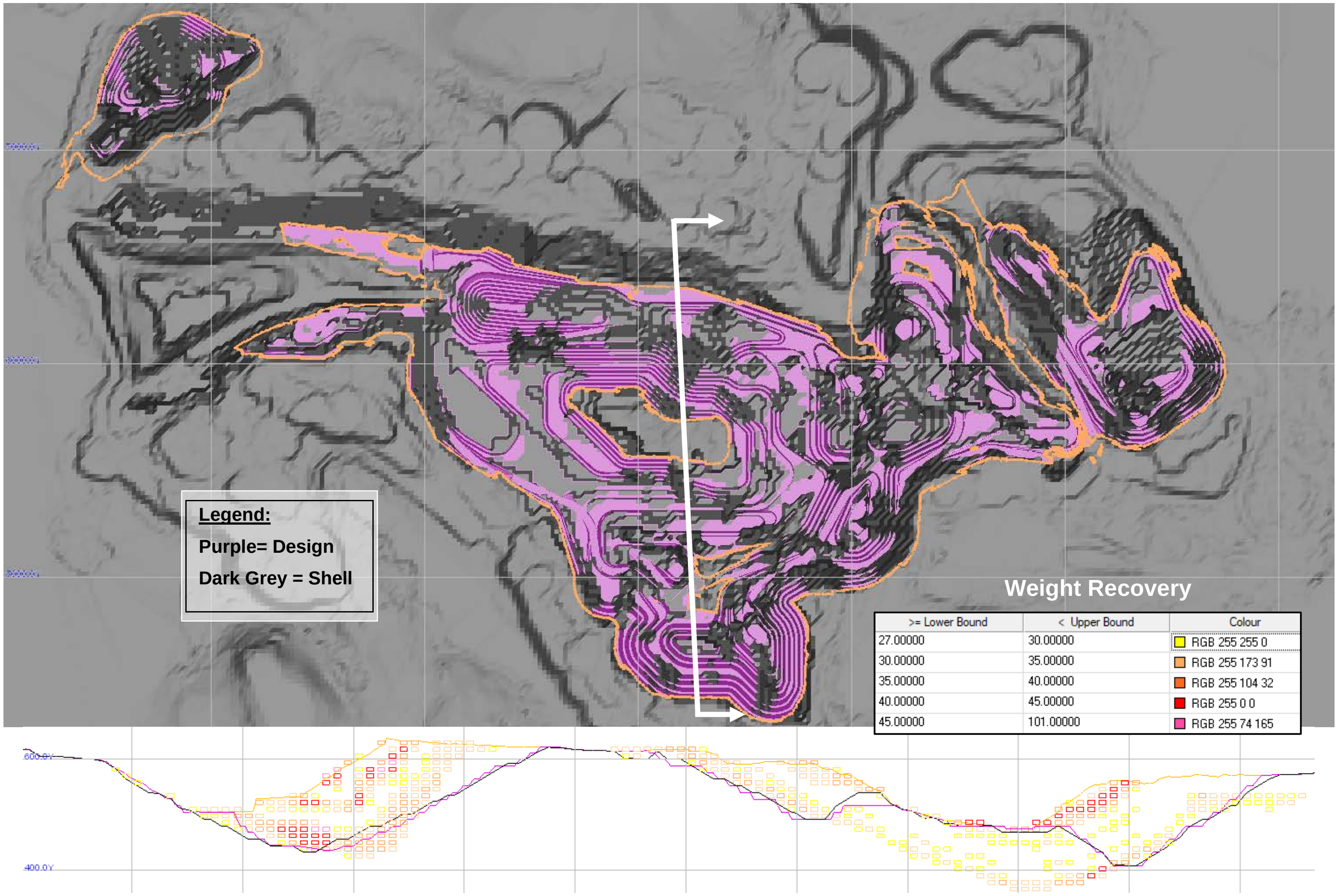
Figure 15.11: Tacora Mine Final Pit Design



Tacora Mine Final Pit Design

Author: Antoine Champagne
Date: March 2021

Figure 15.12: Final Pit Design vs. Whittle Shell



Mining Services

Final Pit Design vs. Whittle Shell

Author: Antoine Champagne

Date: March 2021

15.5 Mineral Reserve Statement

The Mineral Reserve and stripping estimates are based on the final pit design. The Proven and Probable Mineral Reserves total 478.9 Mt at an average grade of 34.89% Fe and 2.62% Mn (Table 15.11). The total tonnage to be mined is estimated at 868.8 Mt for an average strip ratio of 0.82 which includes overburden.

Table 15.11: Tacora Mine Mineral Reserve Estimate (Effective Date of January 1st, 2021)

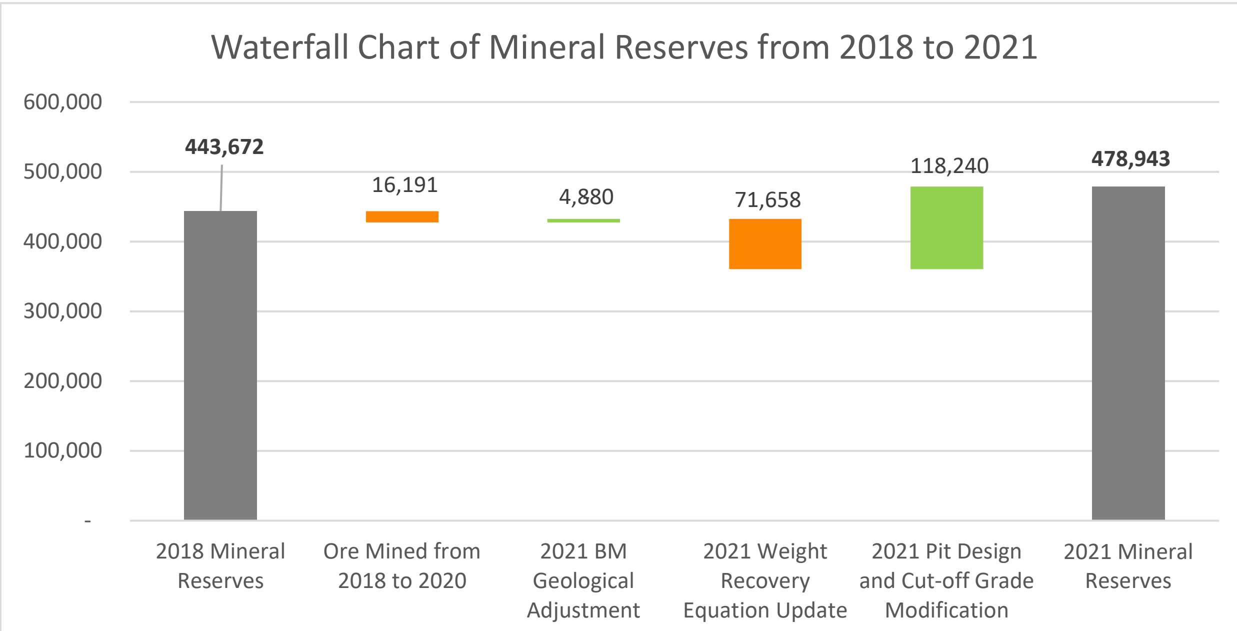
Mineral Reserves by Category		Proven	Probable	Stockpile	Proven & Probable
Crude Ore Tonnage	k dmt	341,439	136,508	997	478,943
Crude Iron Grade	% Fe	34.85	34.97	38.41	34.89
Crude Manganese Grade	% Mn	2.72	2.35	5.31	2.62
Concentrate Tonnage	k dmt	113,577	45,478	369	159,425
Concentrate Iron Grade	% Fe	65.60	65.60	65.30	65.60
Concentrate Manganese Grade	% Mn	1.63	1.53	5.92	1.61
Concentrate Silica Grade	% SiO_2	3.06	3.22	3.00	3.11
Total Weight Recovery	%	33.26	33.32	37.02	33.29
Total Fe Recovery	%	62.62	62.49	62.94	62.59

Notes on Mineral Reserves:

1. *The Mineral Reserves were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards for Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council May 10th, 2014.*
2. *Mineral Reserves based on December 2020 depletion surface merged with an updated Lidar dated September 2017.*
3. *Mineral Reserves are estimated at a minimum 20% Lab weight recovery for all sub-units except sub-unit 52 which is 30%. In addition, sub-unit 34 must have a ratio of weight recovery to iron of at least 1.*
4. *Mineral Reserves are estimated using a long-term iron price reference price (Platt's 62%) of USD 70/dmt and an exchange rate of 1.25 CAD/USD. An Fe concentrate price adjustment of USD 12/dmt was added as an iron grade premium net of a USD 5/dmt marketing charge.*
5. *Bulk density of ore is variable but averages 3.20 t/m^3.*
6. *The average strip ratio is 0.82:1.*
7. *The Mineral Reserve includes a 5.2% mining dilution and a 97% ore recovery.*
8. *The number of metric tonnes was rounded to the nearest thousand. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101.*

The 2021 mineral reserves have increased by 35.3 Mt since 2018. Figure 15.13 presents what has changed since the 2018 Reserves. First, 16.2 Mt of ore have been mined at the Scully mine. Then, for this exercise, the block model has been updated to detail the rock and overburden contact adding 4.9 Mt of ore. The weight recovery equations have been modified based on the data compiled during the last two years of operation. This modification has reduced the weight recovery of ore blocks and converted the marginal ore block to waste block reducing the reserves by 71.7 Mt. Finally, the minimum Lab Weight Recovery was reduced from 27% to 20% which added 118.2 Mt of ore by eliminating the historical high-grading.

Figure 15.13: Waterfall Chart of Mineral Reserves from 2018 to 2021



The 2021 Mineral Reserves are also broken down by pit section in Table 15.12 with the corresponding ore and waste tonnages.

Table 15.12: 2021 Mineral Reserves and Waste Tonnages by Pit Sector

Pit Sector	Ore (kt)	Crude Ore Grades			Waste (kt)	S.R. (W:O)	Total (kt)
		% Fe	% Mn	% SAT			
Boot Pit	47,420	35.67	2.65	1.58	44,792	0.94	92,212
Center Pit West	162,023	34.85	3.65	3.95	90,099	0.56	252,122
Center Pit South	160,919	34.48	2.04	9.26	148,046	0.92	308,965
Center Pit East	63,983	35.12	2.02	2.41	54,665	0.85	118,649
East Pit	43,600	35.25	1.68	6.56	53,259	1.22	96,859
Stockpile	997	38.41	5.31	3.81			997
Total	**478,943**	**34.89**	**2.61**	**5.53**	**390,861**	**0.82**	**869,804**

16 MINING METHODS

16.1 Introduction

The Scully Mine was previously owned by Cliffs until it was closed in February 2014. Since then, the dewatering pumps have been removed and the open pits have filled with water from surface and groundwater inflows. The remaining Mineral Reserves are mined from pushbacks around the existing open pits and from the deepening of certain pit areas. Operations are currently focused on mining pushbacks on upper levels which will provide time for dewatering operations to lower water levels in due course.

Tacora Resources Inc. ("Tacora") has been in production at the site since June of 2019. This Report updates and continues the previous life-of-mine ("LOM") plan and schedule based on updated parameters and constraints which better represent the mining costs, processing capacity and scheduling.

The operation consists of a conventional surface mining method using an owner mining approach with electric and diesel hydraulic shovels, mine trucks and electric drills. Tacora had purchased the loading and drilling fleet in anticipation of the mine restart.

The mine study consists in resizing the open pit based on parameters outlined in this section and producing a LOM plan which targets a production rate of about 6 Mtpy of iron ore concentrate with a crude ore feed to the concentrator averaging 18 Mtpy or about 3 Mtpy per grinding line.

16.2 Mine Designs

16.2.1 Open Pit Phases

The open pit area measures approximately 5.4 km in the east-west direction and 2.1 km in the north-south direction.

Mining of the Tacora open pit is planned with three distinct pits: Boot Pit, Center Pit and East Pit. The Center Pit is further divided into three stand-alone sub-pits / sectors: Center Pit West, Center Pit South, and Center Pit East. The tonnages associated with each mining sector are summarized in Table 16.1 and designs are presented in Figure 16.1. All Pits contain varying degrees of historical mining. Currently, mining is taking place in the Boot Pit and Center Pit West.

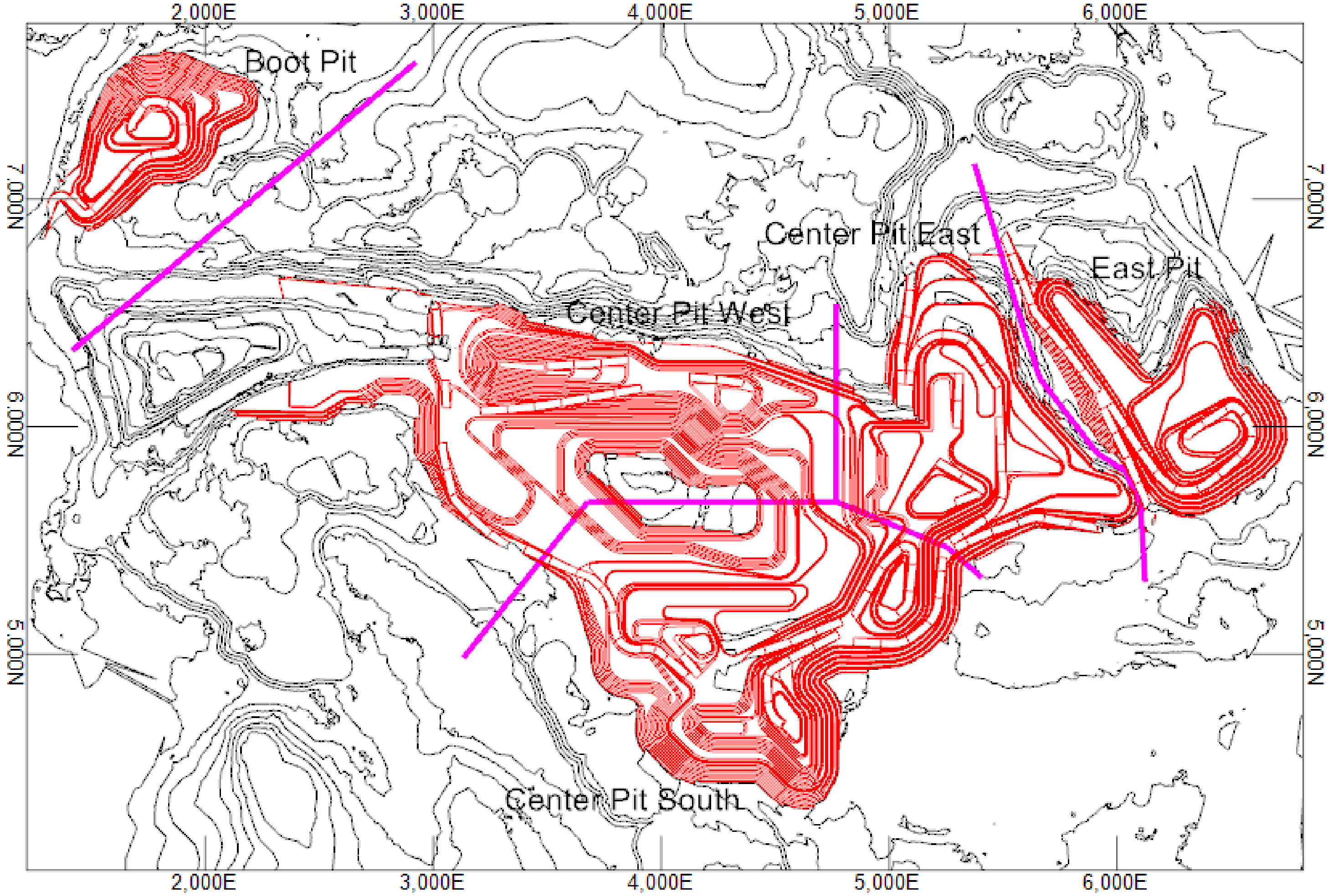
The final design pit contains 477.9 Mt of ore at an average grade of 34.88% Fe, 2.61% Mn and 5.54% SAT. This mineral reserve is sufficient for a 27-year mine life with possibilities for expansion at higher iron ore prices and with the conversion of inferred resources to measured and indicated resources. A total of 868.8 Mt of ore and waste is to be mined for an overall strip ratio of 0.82:1.

Table 16.1: Pit Design Summary by Sector

Pit Sector	Ore (kt)	Crude Ore Grades			Waste (kt)	S.R. (W:O)	Total (kt)
		% Fe	% Mn	% SAT			
Boot Pit	47,420	35.67	2.65	1.58	44,792	0.94	92,212
Center Pit West	162,023	34.85	3.65	3.95	90,099	0.56	252,122
Center Pit South	160,919	34.48	2.04	9.26	148,046	0.92	308,965
Center Pit East	63,983	35.12	2.02	2.41	54,665	0.85	118,649
East Pit	43,600	35.25	1.68	6.56	53,259	1.22	96,859
Total	**477,946**	**34.88**	**2.61**	**5.54**	**390,861**	**0.82**	**868,807**

Note: Ore tonnage and grades inclusive of dilution

Figure 16.1: Final Pit Design



Mining Services

Tacora Mine Final Pit Design

Author: Peter Harquail

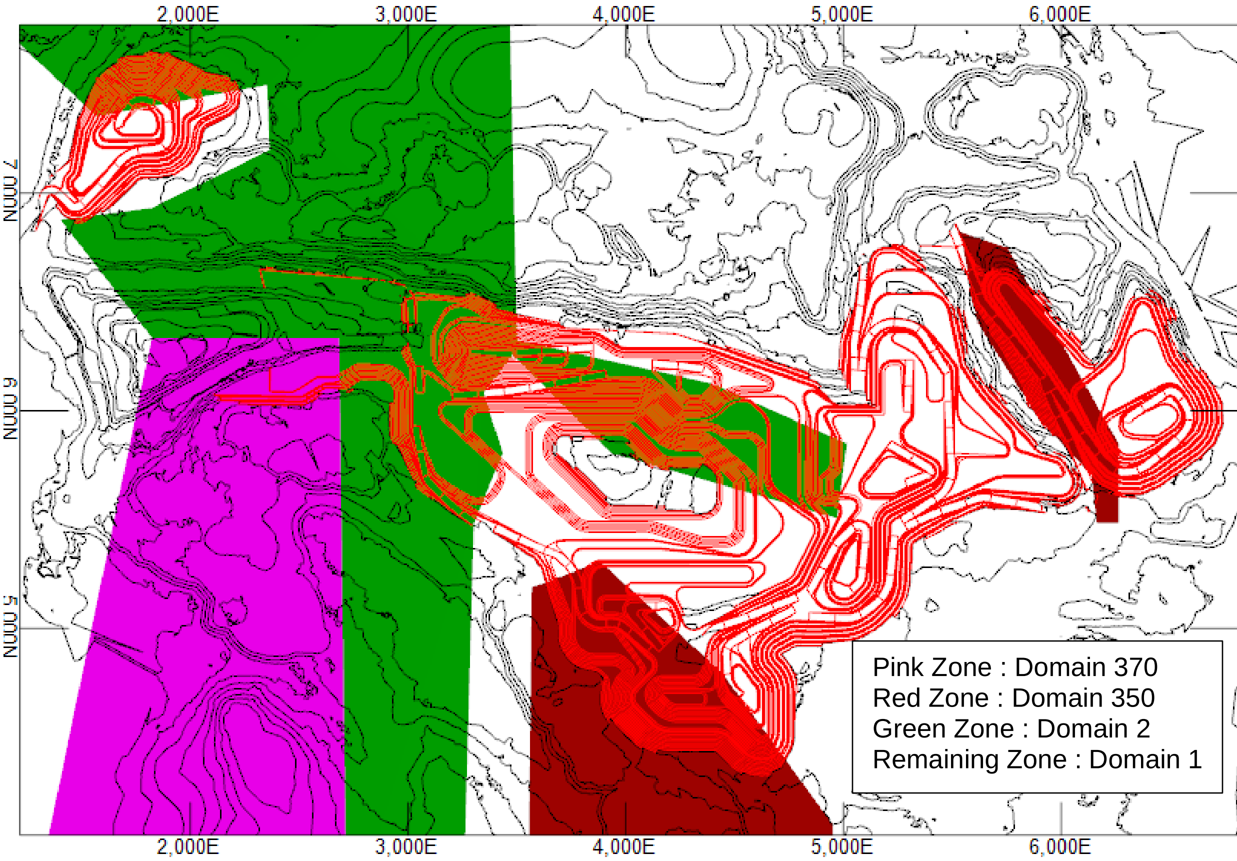
Date: April, 2021

The pit design slope profiles adhere to recommendations generated by Golder Associates according to four main profiles as summarized in Table 16.2. The batter angles vary from 50 to 65 degrees based on a final 12 to 24 m bench height. The DOM2 domain contains single 12 m benches while all other zones allow for 24 m double benches.

Table 16.2: Pit Phase Design Criteria

Design Sector	DOM1	DOM2	DOM350	DOM370	Overburden Slope
Number of Benches (n)	5	10	5	5	1
Vertical Bench Height (m)	24.0	12.0	24.0	24.0	12.2
Bench Face Angle (deg.)	65	50	50	70	26.6
Avg. Catch Berm Width (m)	12.00	9.10	10.00	10.00	12.2
Horz. Slope Distance (m)	116.0	191.7	150.7	93.7	36.5
Vert. Slope Distance (m)	120.0	120.0	120.0	120.0	12.2
Slope (H:V)	0.97	1.60	1.26	0.78	3.00
IRA in Degrees (crest-to-crest)	**46.0**	**32.0**	**38.5**	**52.0**	**18.4**
Ramp Width (m)	36.6	36.6	36.6	36.6	36.6
Nb. Ramp Segments in Slope	0	0	0	0	0
Geotech. Catch Berm Width (m)	0.0	0.0	0.0	0.0	0.0
Nb of Geotechnical Catch	0	0	0	0	0
Horizontal Slope Distance (m)	116.0	191.7	150.7	93.7	36.5
Vertical Slope Distance (m)	120.0	120.0	120.0	120.0	12.2
Slope (H:V)	0.97	1.60	1.26	0.78	3.00
OSA in Degrees (crest-to-crest)	**46.0**	**32.0**	**38.5**	**52.0**	**18.4**

Figure 16.2: Geotechnical Bench Slope Design Recommendations



16.2.2 <u>Overburden and Waste Rock Storage</u>

A total of 390.9 Mt of waste material will be mined throughout the LOM. Waste rock storage is planned in depleted pits and in conventional waste storage area (dumps) outside of pits. Dumping has already taken place in some of the surface dumps. The historical amount of material already dumped in each waste location is unknown and not included in this report. In the early years of the project, waste material will be stored to the south of the pits on waste dumps South and South West and inside the West Extension in-pit dump. Later, as the Project expands to the East, the North Dump will become active. Once the East Pit West is depleted, waste material will be stored in this pit allowing for lower haulage costs and a reduced environmental impact. The North West dump is used for waste rock coming from Center Pit West and Boot Pit in the later years of the Project.

The waste dumps located outside of the pits are mostly designed above historic waste storage locations. The new waste dumps are built in 12 m stacks with 14 m catch benches allowing for rehabilitation when filled to maximum capacity. A 40 m distance was kept from the base of the new waste dumps to the limit of the historic ones upon which they are built. This distance was kept to allow for drainage infrastructures around the new waste dumps.

Waste dump design and location are presented in Figure 16.3. Waste dump metrics are presented in Table 16.3 and Table 16.4.

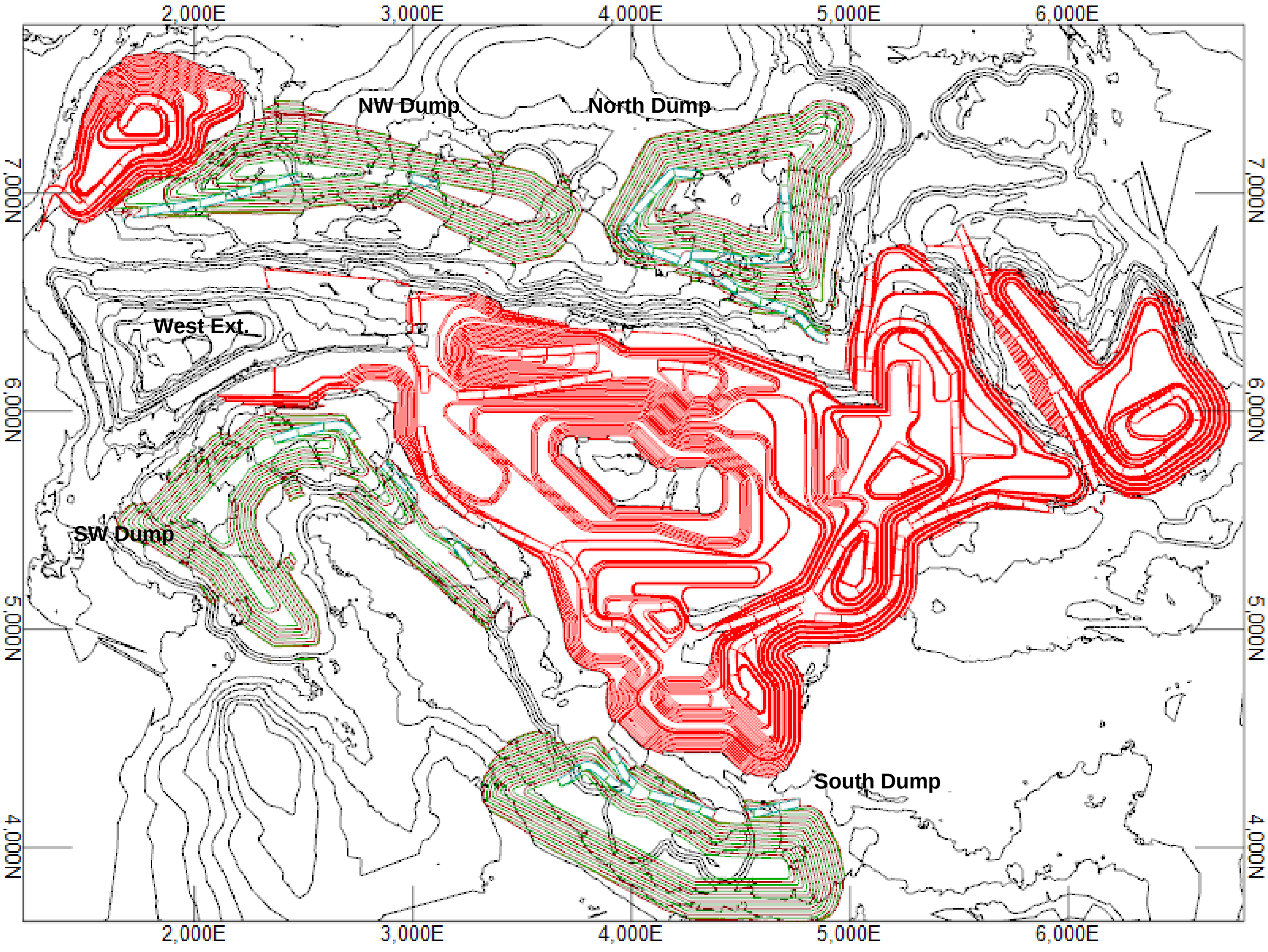
Table 16.3: Waste Storage Capacities

Waste Dump	Capacity (Mt)	Capacity (M m^3)	Surface Area (ha)
South West Dump	82.1	38.1	770.0
South Dump	43.4	20.1	834.6
North Dump	76.6	35.5	760.0
North West Dump	51.5	23.9	703.9
West Extension In-Pit Dump	72.6	33.7	0
Boot Pit In-Pit Dump	92.2	42.8	0
Total	**396.7**	**184.2**	**4,197.2**

Table 16.4: Waste Pile Design Criteria

Waste Dump	Avg. Catch Bench Width (m)	Pile Face Angle (deg)	Overall Slope angle (H:V)	Approximate Height (m)
South West Dump	14	35	2.5:1	60
South Dump	14	35	2.5:1	60
North Dump	14	35	2.5:1	60
North West Dump	14	35	2.5:1	72
West Extension In-Pit Dump	14	35	2.5:1	0
Boot Pit In-Pit Dump	14	35	2.5:1	0

Figure 16.3: Waste Storage Dumps



Mining Services	Waste Storage Dumps	Author: G Mining Date: November 2017

Overburden was not included in this study as most of the overburden present on site was removed during previous operations. It will be important to closely monitor the presence of overburden. If found, it will have to be stored in a specific location to be re-used when rehabilitation of the site is performed.

16.2.3 Ore Stockpiles

Ore stockpiles are located to the East adjacent to the crusher location. The total stockpile capacity is 11.0 Mt, which is greater than the stockpile requirements of 9.5 Mt.

The ore stockpile design criteria are presented in Table 16.5. The area used for stockpiling has extra capacity to allow building multiple stockpiles with different ore characteristics to simplify rehandling and blending activities.

Table 16.5: Stockpile Design Criteria

Ore Stockpile Characteristics	Catch Bench Width (m)	Pile Face Angle (deg)	Overall Slope Angle (H:V)	Approximate Height (m)	Max Capacity (Mt)
Stockpile at Max. Capacity	14	35	2.5:1	30	11.0

16.2.4 Mine Haul Roads

Most of the haul roads already exist. However, adjustments and additional roads are required to access the new waste storage locations and to accommodate the new pit exit locations. Mine haul road profiles are presented in Table 16.6.

Haul Road 1 is the longest road to build. It will link all the southern infrastructures (dumps and pits) to the crusher. An offset of 60 m was kept between the pit rim and railways where applicable. This offset allows for the construction of the 41 m wide haul road and a service road. The cost for building the road was calculated based on a cut & fill analysis performed in AutoCAD.

Haul Roads 2 and 4 require work on historic waste dumps. If built in the warmer months of the year, they will require much less work as waste material can more easily be pushed.

Haul Road 3 is simply a by-pass from the north of the west extension to the south. It will allow for a much shorter haul path to the crusher for material coming from West extension North and Boot.

Table 16.6: Haul Road Segments

Haul Road Segment Number	Length (m)
Mine Road 1	4,120
Mine Road 1.1	1,225
Mine Road 2	727
Mine Road 2.1	689
Mine Road 3	473
Mine Road 4	419

16.3 Production Schedule

16.3.1 Production Schedule Optimization

The LOM schedule was optimized using Minemax Scheduler software to maximize the project value according to various constraints and to achieve various blending constraints. The optimal schedule guided the detailed schedule presented hereafter. The optimization was based on the pit sectors established for the pit and related mineral reserves. Deswik.Blend software was used in conjunction of MineMax to create the detailed mine and blend schedule.

16.3.2 Mine Production Schedule

The mine production schedule is completed on a quarterly basis during the first year (2021) and on an annual basis thereafter. There is a ramp up period over the Year 2021, after which it is expected that the mill will proceed at the design capacity of 6 Mt of concentrate per year. The mine ramp-up period lasts a total of 12 months which is planned to upgrade and commission the processing facilities.

The objective of the LOM plan is to maximize the discounted operating cash flow of the Project subject to various constraints:

- Supply 18 Mtpy of crude ore to the plant to produce 6 Mtpy of iron ore concentrate.
- Supply ore to the plant subject to the contaminant level constraints.
- Limit the mining rate to approximately 36 Mtpy.
- Limit the vertical drop down rate to 5 benches per phase per year.

The peak mining rate is 36 Mt mined and 39.3 Mt moved in Year 2035. The average mining rate is 33.0 Mtpy during operations.

Stockpiling reaches 9.3 Mt in Year 2029. This ore stockpiling level is required to achieve the mill feed constraints and requires certain ore types to be stockpiled and others reclaimed. It also creates a stockpile to draw from for a few months after mining has ended. Over the LOM, a total of 18.8 Mt is rehandled from the stockpile. The annual mine production and stockpile inventory are presented in Figure 16.4 and Figure 16.5.

Figure 16.4: Mine Production

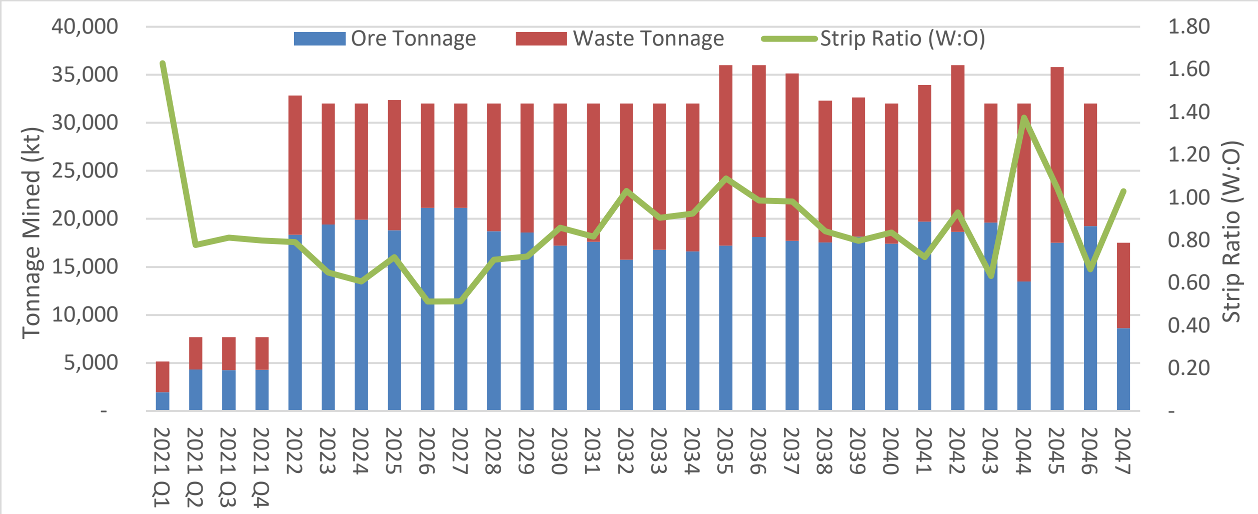


Figure 16.5: Stockpile Inventory

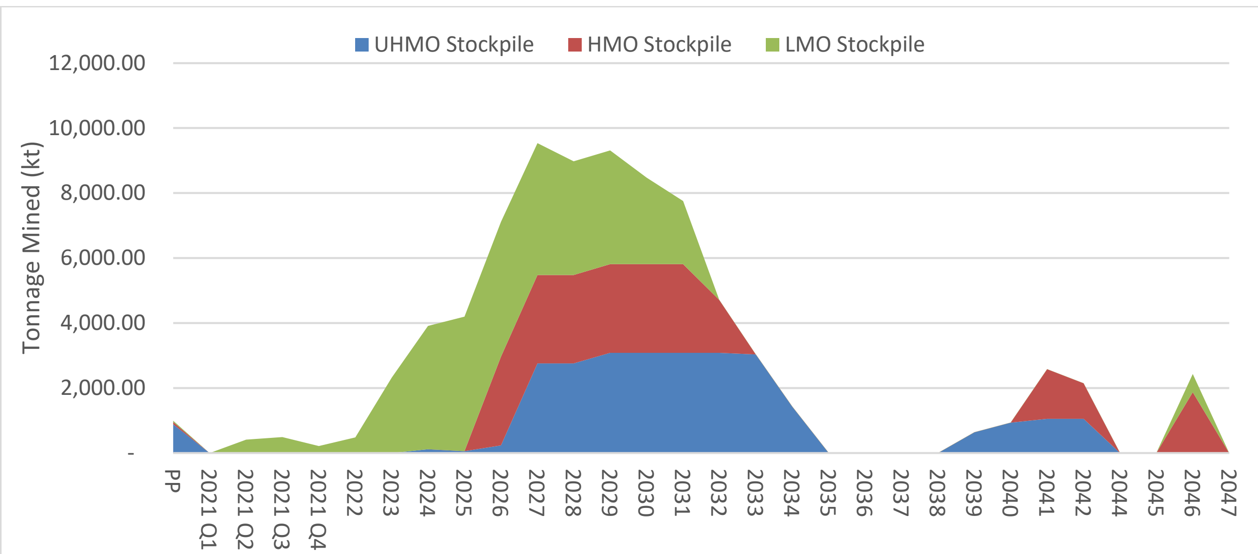


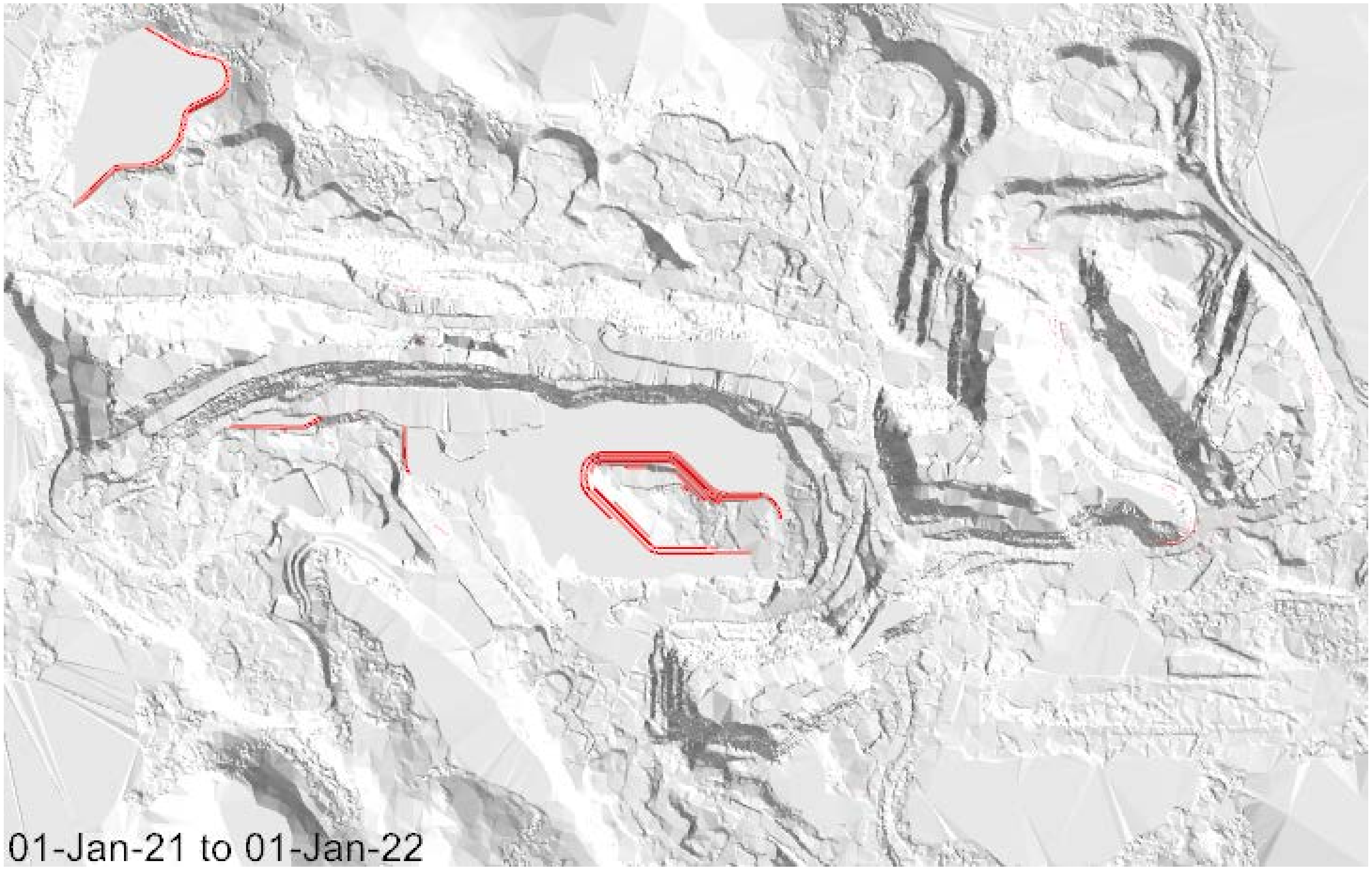
Table 16.7: Mine Production Schedule Detail by Period

Period	Total Mined (kt)	Ore Mined (kt)	Waste Mined (kt)	Strip Ratio (W:O)	SP Rehandle (kt)	Total Moved (kt)
2021 Q1	5,148	1,958	3,190	1.63	997	6,145
2021 Q2	7,693	4,327	3,366	0.78	197	7,891
2021 Q3	7,693	4,247	3,447	0.81	304	7,998
2021 Q4	7,693	4,279	3,415	0.80	267	7,961
2022	32,849	18,345	14,503	0.79	-	32,849
2023	32,000	19,411	12,589	0.65	358	32,358
2024	32,000	19,917	12,083	0.61	-	32,000
2025	32,381	18,821	13,561	0.72	904	33,286
2026	32,000	21,151	10,849	0.51	-	32,000
2027	32,000	21,142	10,858	0.51	93	32,093
2028	32,000	18,732	13,268	0.71	561	32,561
2029	32,000	18,569	13,431	0.72	-	32,000
2030	32,000	17,211	14,789	0.86	835	32,835
2031	32,000	17,613	14,387	0.82	721	32,721
2032	32,000	15,758	16,242	1.03	3,045	35,045
2033	32,000	16,794	15,206	0.91	1,681	33,681
2034	32,000	16,631	15,369	0.92	1,601	33,601
2035	36,000	17,226	18,774	1.09	1,431	37,431
2036	36,000	18,121	17,879	0.99	-	36,000
2037	35,129	17,733	17,396	0.98	-	35,129
2038	32,308	17,539	14,769	0.84	-	32,308
2039	32,631	18,152	14,479	0.80	-	32,631
2040	32,000	17,423	14,577	0.84	-	32,000
2041	33,944	19,720	14,224	0.72	-	33,944
2042	36,000	18,639	17,361	0.93	441	36,441
2043	32,000	19,605	12,395	0.63	-	32,000
2044	32,000	13,476	18,524	1.37	2,965	34,965
2045	35,801	17,528	18,273	1.04	-	35,801
2046	32,015	19,248	12,767	0.66	-	32,015
2047	17,520	8,632	8,888	1.03	2,438	19,958
Total	**868,807**	**477,946**	**390,861**	**0.82**	**18,841**	**887,648**

Table 16.8: Stockpile Inventory Schedule

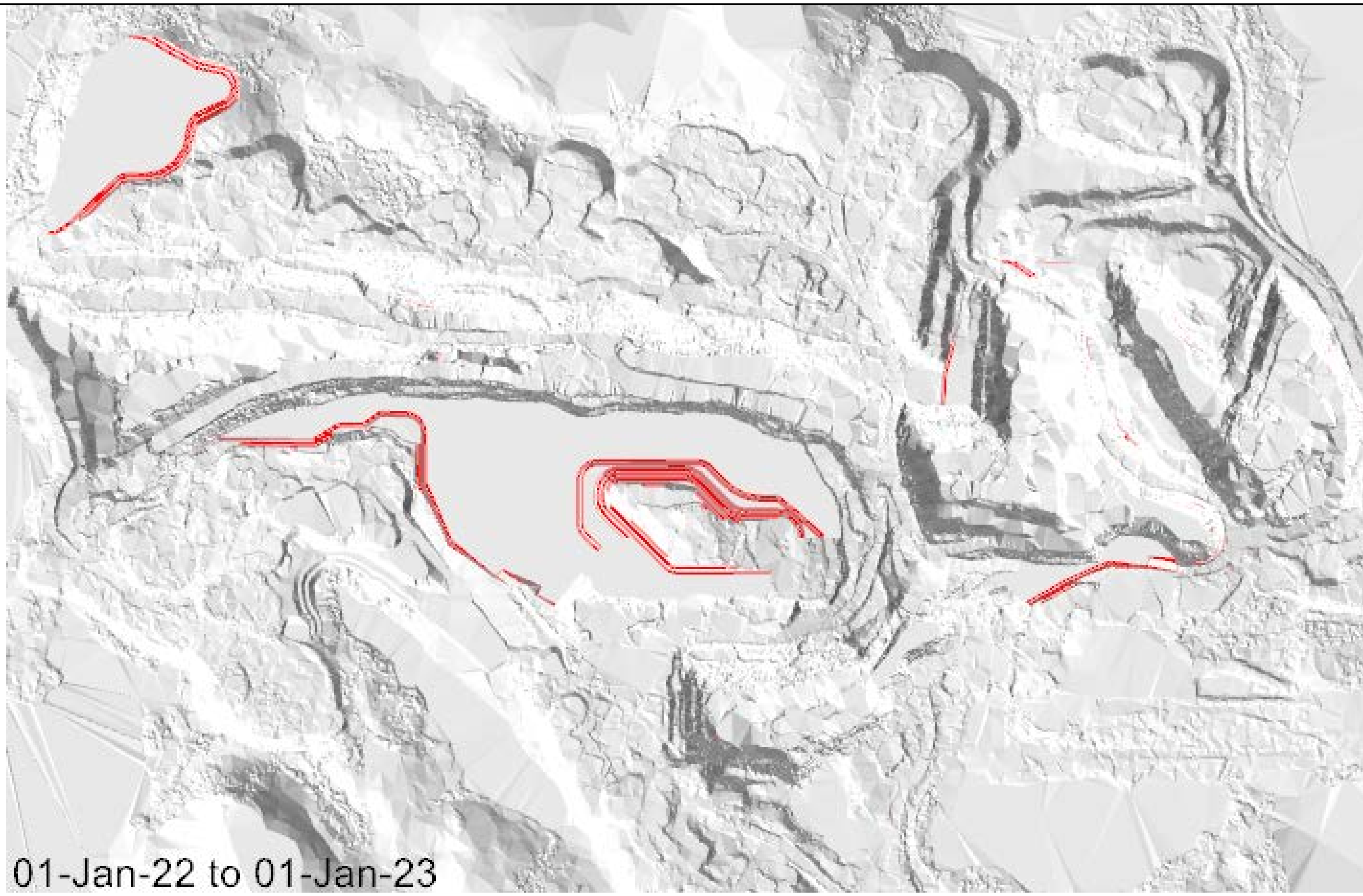
Period	LMO (kt)	HMO (kt)	UHMO (kt)	Total (kt)
2021 Q1				
2021 Q2	414			414
2021 Q3	487			487
2021 Q4	220			220
2022	482			482
2023	2,321			2,321
2024	3,789		121	3,910
2025	4,147		53	4,200
2026	4,147	2,729	235	7,112
2027	4,054	2,729	2,756	9,539
2028	3,493	2,729	2,756	8,978
2029	3,497	2,729	3,085	9,311
2030	2,664	2,729	3,085	8,479
2031	1,943	2,729	3,085	7,757
2032		1,628	3,085	4,713
2033			3,032	3,032
2034			1,431	1,431
2035				
2036				
2037				
2038				
2039			641	641
2040			931	931
2041		1,536	1,051	2,586
2042		1,094	1,051	2,145
2043	297	1,499	1,169	2,965
2044				
2045				
2046	566	1,872		2,438
2047				

Figure 16.6: Production Schedule – Year 2021



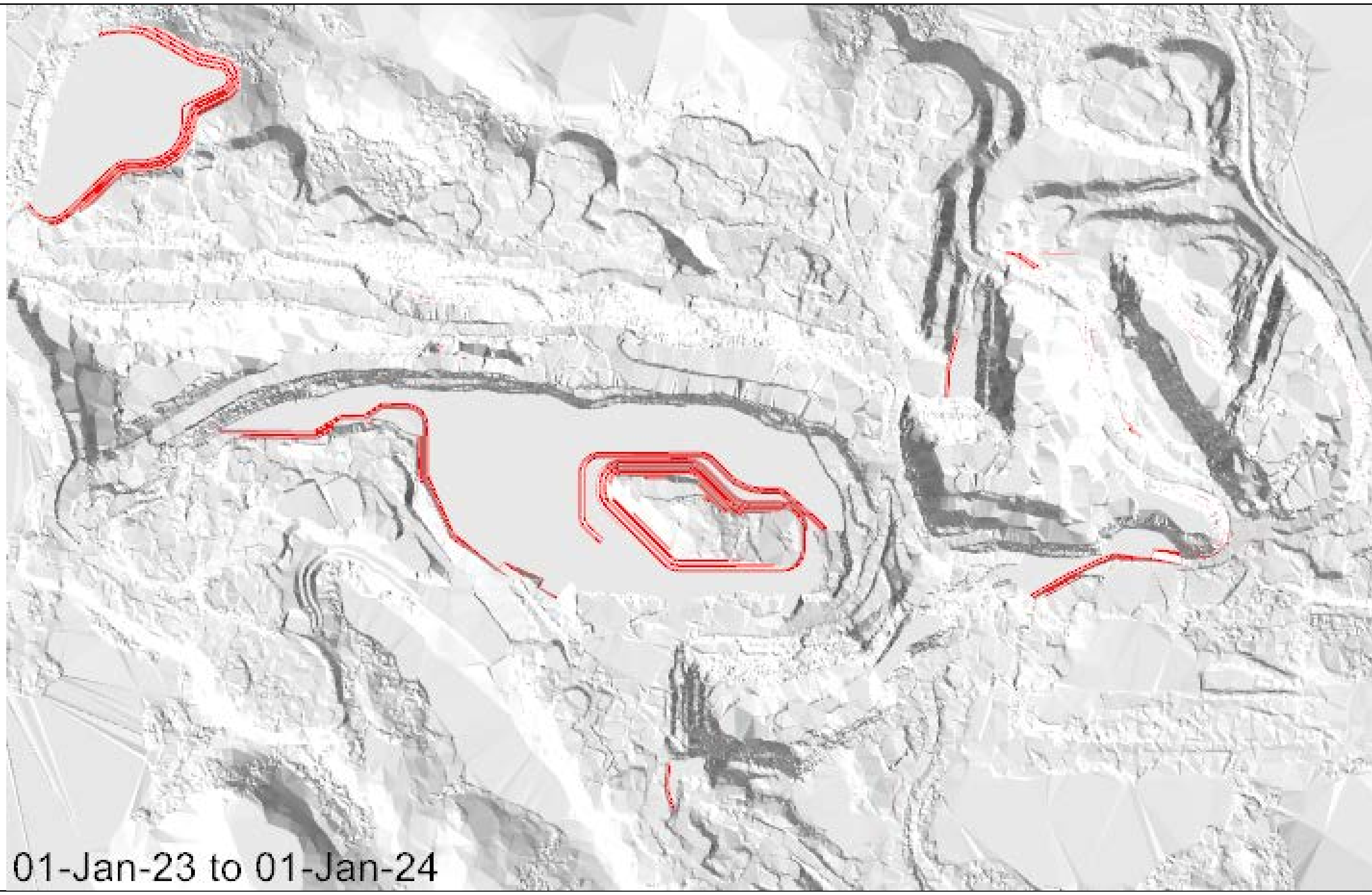
Source: G Mining
Date: April 2021

Figure 16.7: Production Schedule – Year 2022



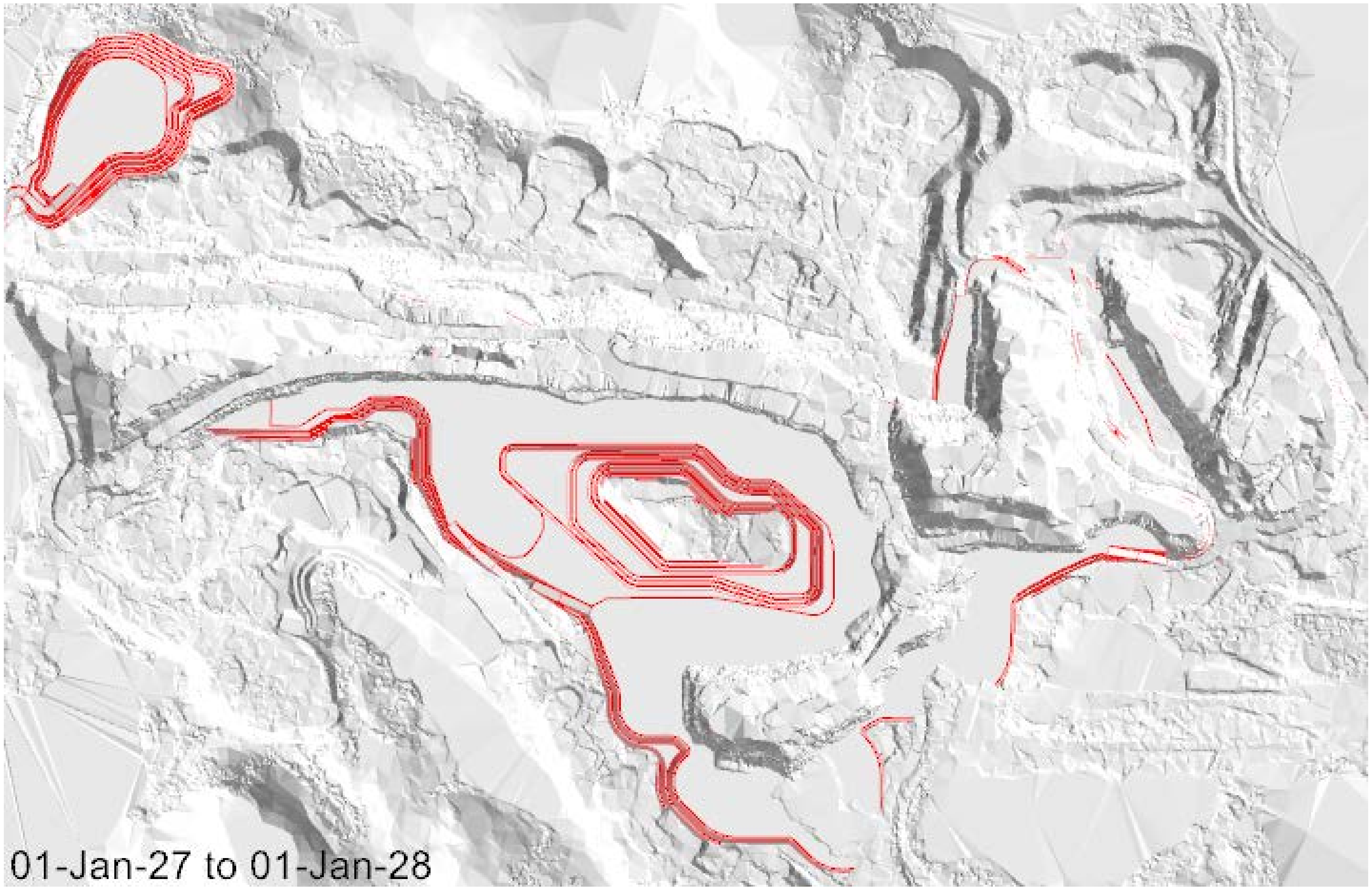
Source: G Mining
Date: April 2021

Figure 16.8: Production Schedule – Year 2023



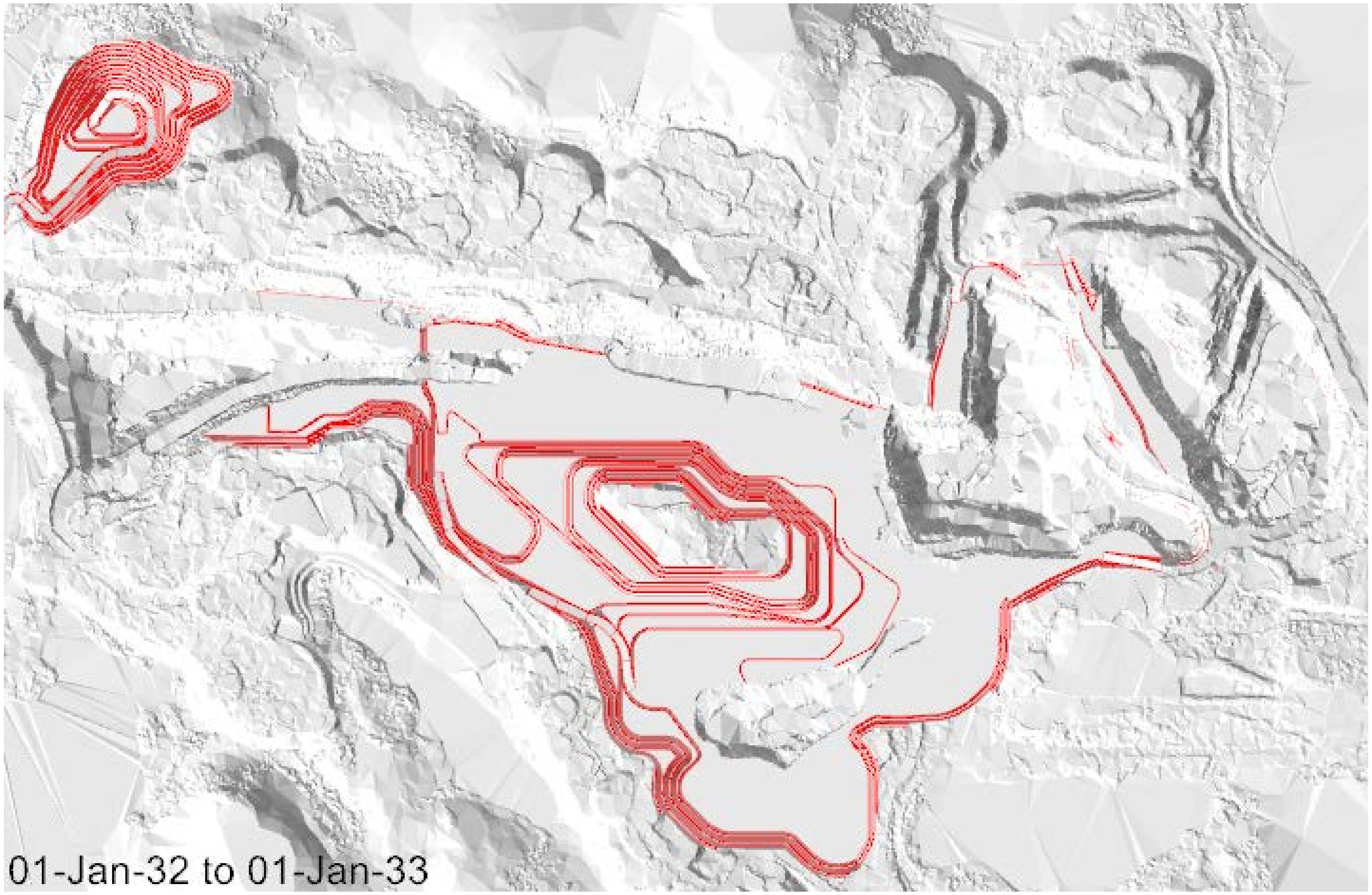
Source: G Mining
Date: April 2021

Figure 16.9: Production Schedule – Year 2027



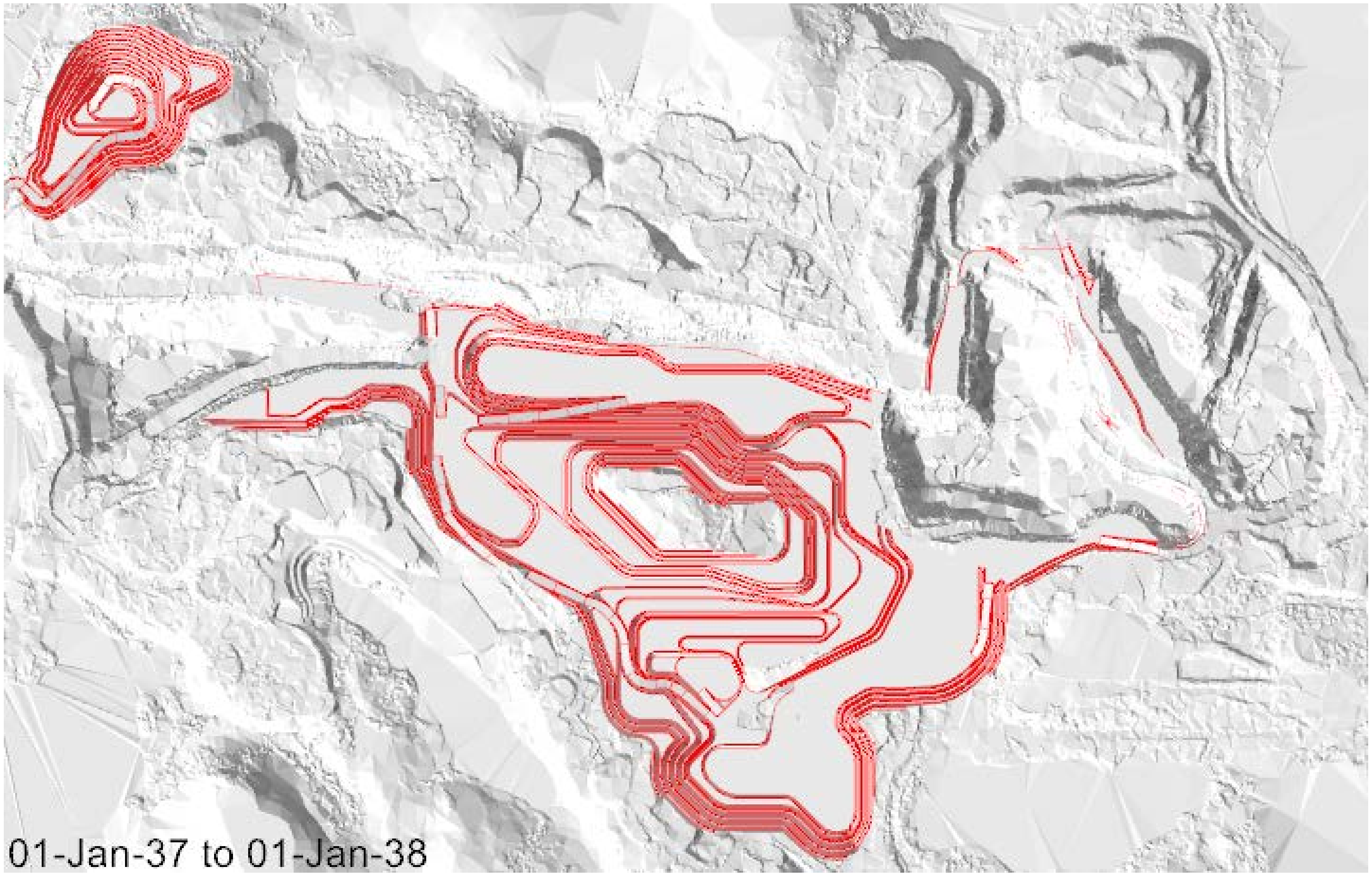
Source: G Mining
Date: April 2021

Figure 16.10: Production Schedule – Year 2032



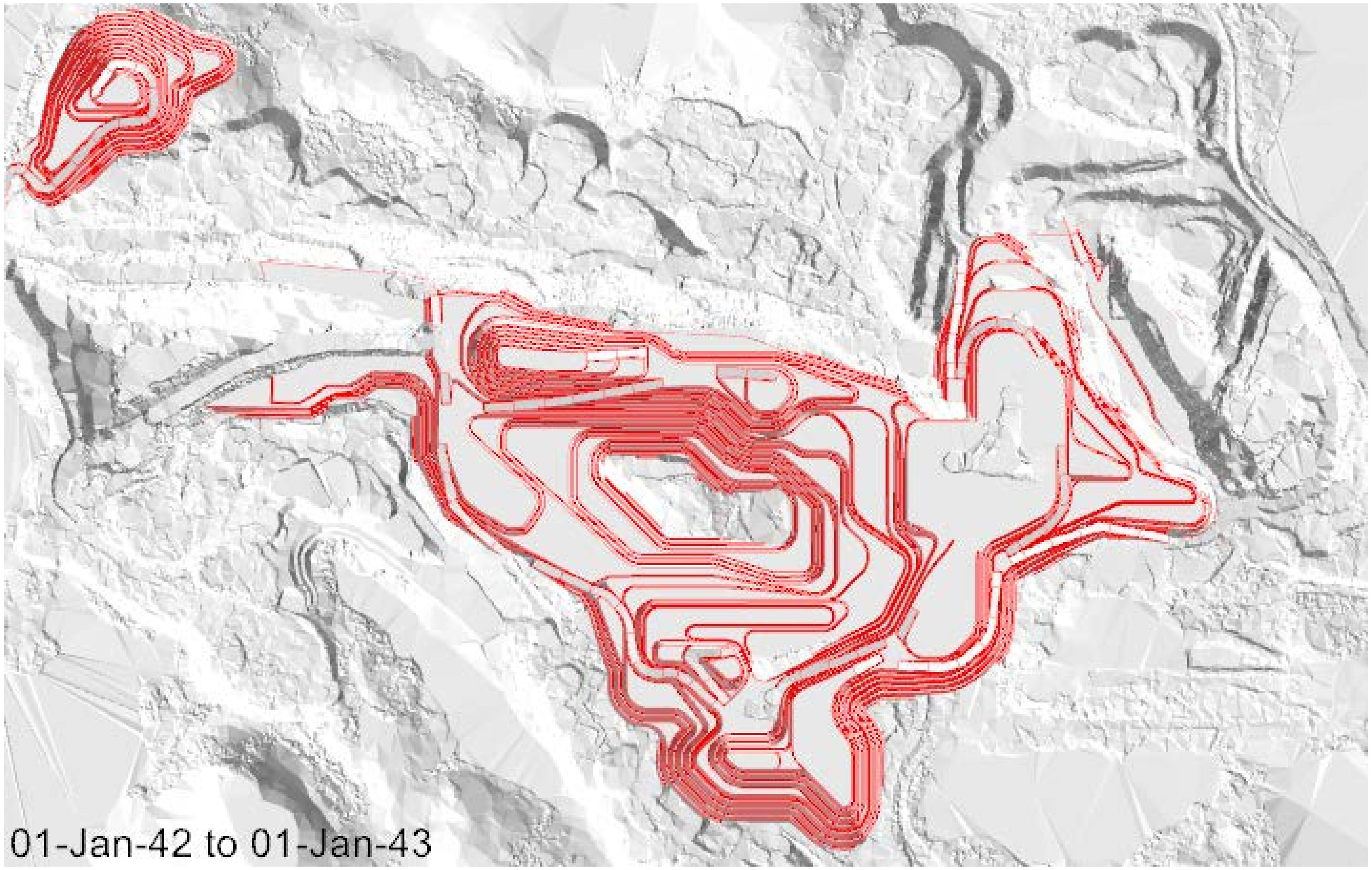
Source: G Mining
Date: April 2021

Figure 16.11: Production Schedule – Year 2037



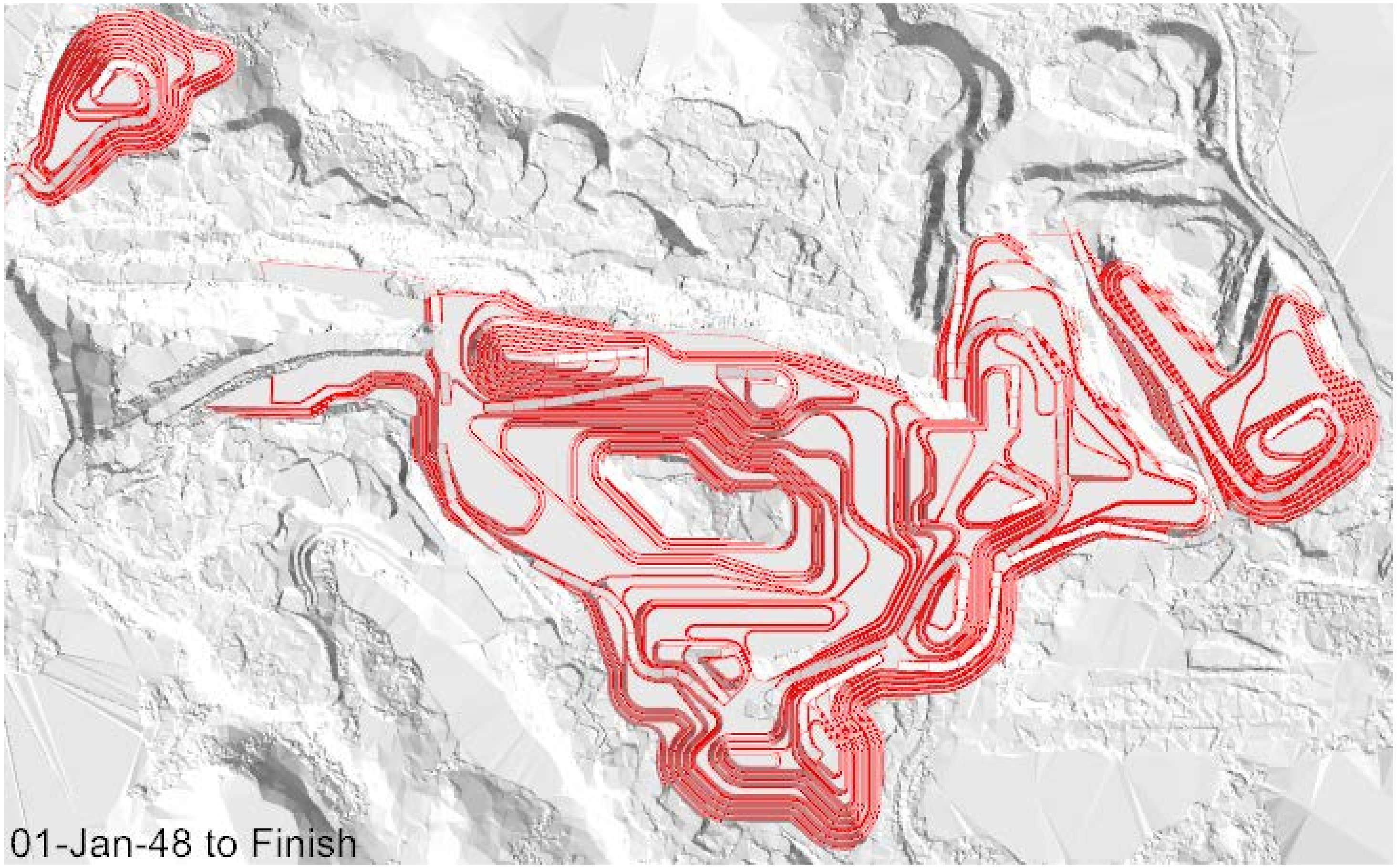
Source: G Mining
Date: April 2021

Figure 16.12: Production Schedule – Year 2042



Source: G Mining
Date: April 2021

Figure 16.13: Production Schedule – Year 2047 (End of Life of Mine)



Mining Services

Source: G Mining
Date: April 2021

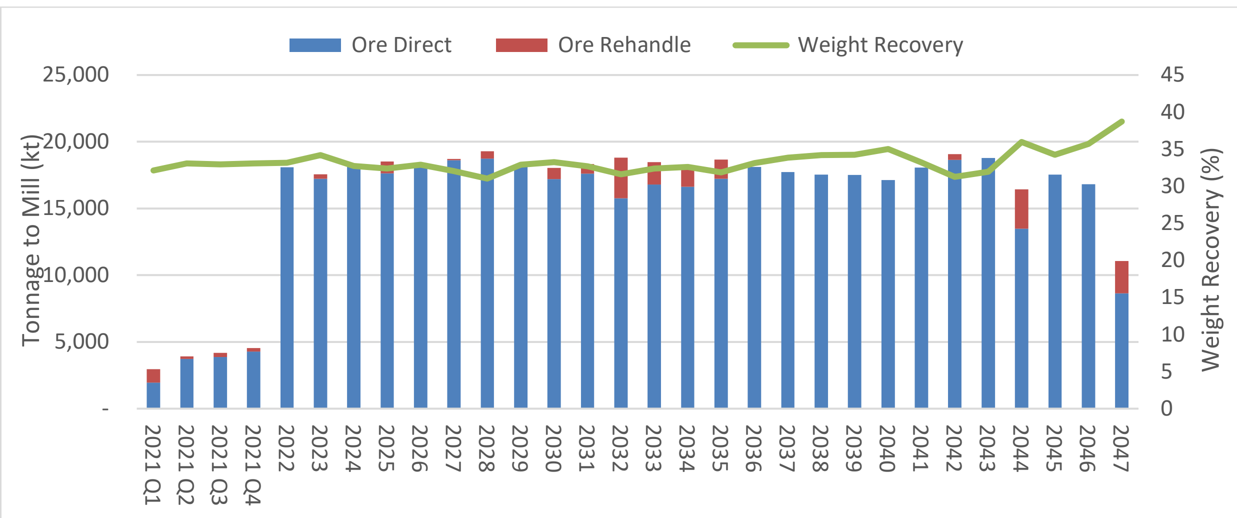
16.3.3 Mill Production Schedule

The mill production schedule is presented in Table 16.10. The ramp-up period during the four quarters of 2021 is detailed in Table 16.9. A total concentrate tonnage of 5.12 Mt is planned for 2021. The mill throughput achieves the design throughput targeted at 6.0 Mt concentrate from Year 2022 onwards.

Table 16.9: Mill Ramp-Up

Mill Ramp-Up Schedule	Period Treatment	Concentrate Tonnage (kt)
2021 Q1	Operations	946
2021 Q2	Operations	1,294
2021 Q3	Operations	1,376
2021 Q4	Operations	1,505
2022 +	Operations	6,000

Figure 16.14: Mill Feed from Source



The mill feed tonnages and corresponding grades are detailed in Table 16.10.

Table 16.10: Mill Production Schedule Detail by Period

Period	Mill Feed (kt)	Conc. Prod. (kt)	Mill Feed Grades			
			% Fe	% Mn	% SAT	Weight Rec.
2021 Q1	2,955	949	34.57	2.32	5.46	32.12
2021 Q2	3,913	1,294	34.96	2.71	5.14	33.08
2021 Q3	4,173	1,376	34.92	2.96	4.35	32.97
2021 Q4	4,546	1,505	34.84	3.07	3.91	33.10
2022	18,084	6,000	34.58	3.45	3.31	33.18
2023	17,573	6,009	34.99	3.73	3.93	34.19
2024	18,328	6,000	34.84	2.96	8.00	32.74
2025	18,530	6,002	34.86	2.87	8.10	32.39
2026	18,239	6,000	34.95	3.25	5.27	32.90
2027	18,715	5,999	34.90	2.62	8.01	32.05
2028	19,292	5,987	35.21	2.49	8.08	31.03
2029	18,236	6,000	34.66	3.29	8.00	32.90
2030	18,043	5,999	34.38	3.23	8.14	33.25
2031	18,334	5,995	34.30	3.00	7.04	32.70
2032	18,802	5,946	34.15	2.32	7.78	31.62
2033	18,475	5,980	34.27	2.68	6.03	32.37
2034	18,232	5,948	34.28	2.52	5.83	32.62
2035	18,657	5,942	34.44	2.32	6.26	31.85
2036	18,121	6,000	34.94	2.86	4.88	33.11
2037	17,733	6,000	35.20	2.79	4.23	33.83
2038	17,539	6,000	35.18	2.70	4.48	34.21
2039	17,511	6,000	35.02	2.33	4.47	34.26
2040	17,133	6,000	35.11	2.05	3.63	35.02
2041	18,064	6,000	35.37	2.06	3.05	33.21
2042	19,080	5,966	35.43	2.18	2.49	31.27
2043	18,785	6,000	34.80	2.11	1.33	31.94
2044	16,441	5,915	35.60	2.03	3.07	35.98
2045	17,528	6,000	34.86	2.00	6.25	34.23
2046	16,810	6,000	35.11	1.59	6.28	35.69
2047	11,070	4,287	36.65	1.28	5.18	38.72
Total	**478,943**	**159,100**	**34.89**	**2.60**	**5.50**	**33.22**

Five constraints are specified in the Deswik.Scheduler to control the blend to the process plant. These constraints are created by the physical properties of the ore through the mill and the limitations of certain impurities in the concentrate.

- Rock Type 52 Blend:

  Due to the grindability of this rock type, there is a limit of 15% of the crusher throughput to ensure stability in the crushing and grinding circuits.

- South Pit Blend:

  Historically there were perceived problems with the grindability of material from the South Pit. There was a previous constraint limiting the South Pit mill feed to a maximum of 30% of the plant throughput. It is suspected that this constraint was related to the high magnetite content of the South Pit ore. The South Pit area has changed significantly, almost tripling in size compared to the previous designs. It is recommended to study the root cause of this historical constraint so that it can be better understood and its impacts for non-compliance.

- Magnetite in ore feed:

  Based on the design capacity of the dry sector of the process, an 8% limitation was imposed on the magnetite context of the crude ore feed.

- Mn Concentrate Limit:

  Magnase levels must be below 2.0% in concentrate to meet concentrate specifications to customers.

- Silica Concentrate Limit:

  Silica levels must be below 3% in concentrate to meet concentrate specifications.

Figure 16.15 summarizes the planned crude ore feed parameters with their respective limitations by year. In early years the Rock Type 52 blend is the limiting factor followed by magnetite near the middle of mine life. Silica is never problematic in the final concentrates, as shown in Figure 16.16.

Figure 16.15: Mill Feed Constraints

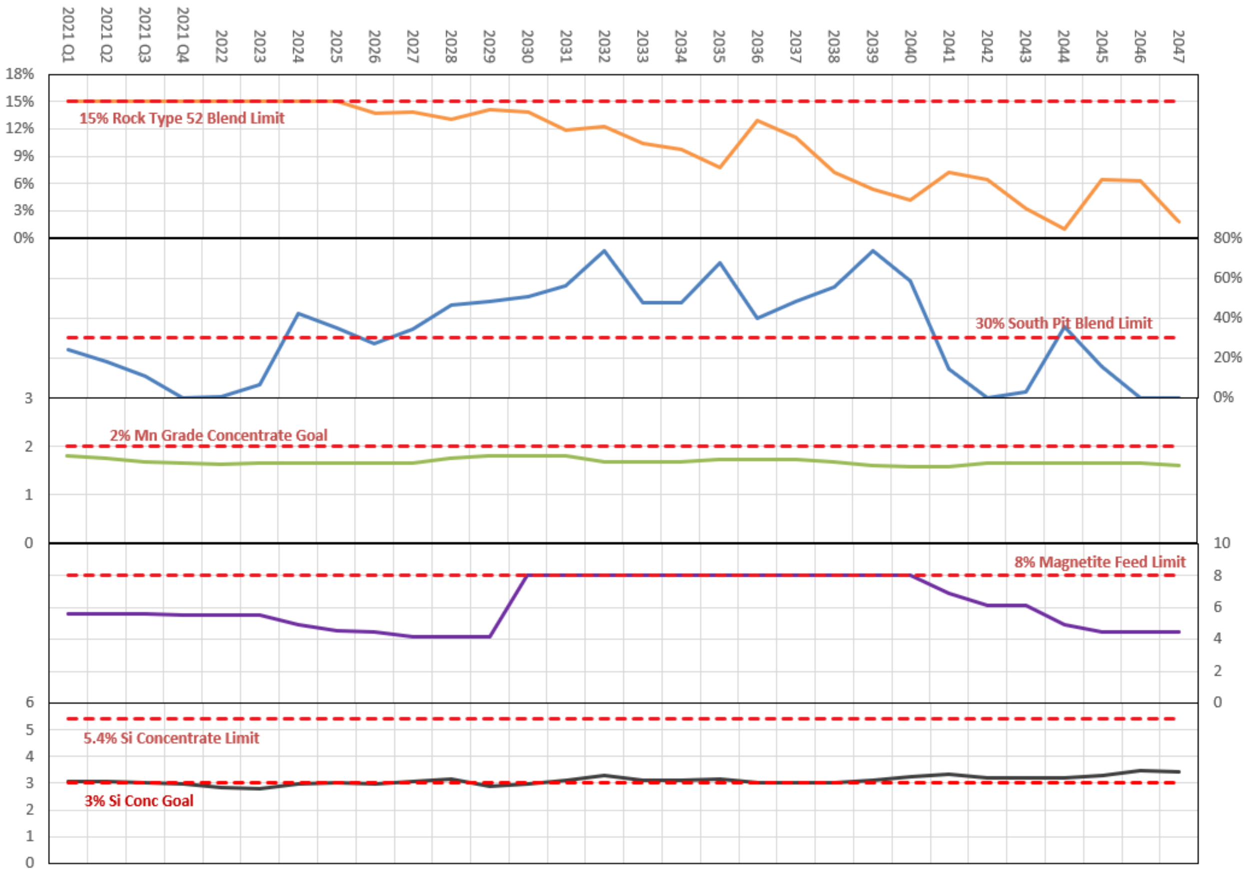
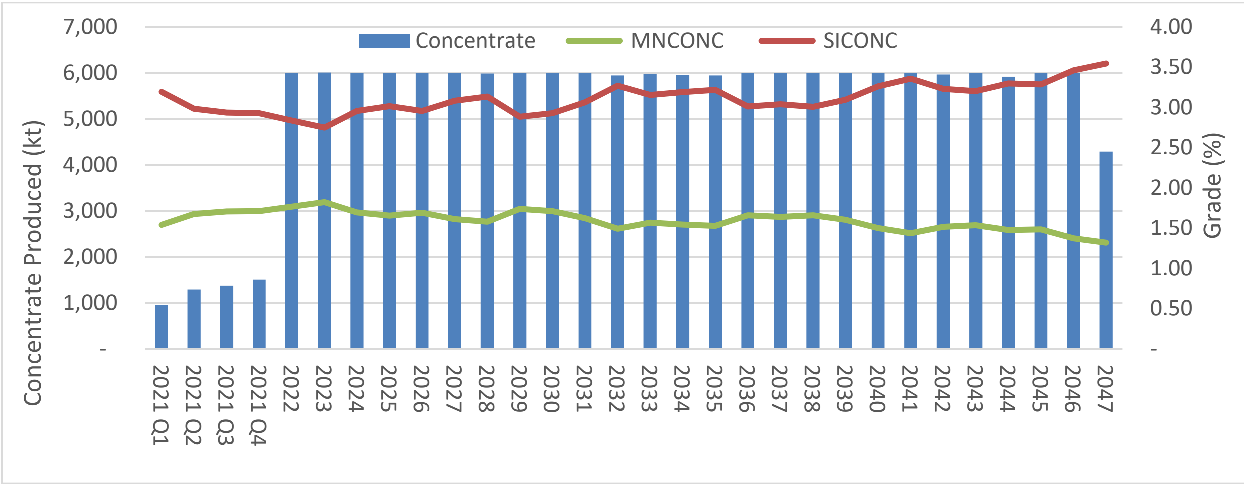


Figure 16.16: Concentrate Produced Grades



16.4 Mine Operations and Equipment Selection

16.4.1 Mine Operations Approach

Mining is being carried out using conventional open pit techniques with electric and diesel powered hydraulic shovels and mining trucks in a bulk mining approach. An owner-mining open pit operation is currently executed, with the outsourcing of certain support activities such as explosives manufacturing and blasthole loading.

16.4.2 Drilling and Blasting

Drill and blast specifications are established to effectively single pass drill and blast a 12 m bench. For this bench height a 349 mm (13¾ in.) blast holes size is used on a 9.25 m burden by 9.25 m spacing with 1.8 m of sub-drill. This blast hole size has been used in the past and production blasting was the same in all areas of the mine with a typical blast involving about 65 holes and about 220 kt per blast. The current blasts typically exceed 1.0 Mt.

The rotary drill rigs currently in use by the project (1x 49 HR electric and 1x CAT MD6640 electric) have a hole size range of 251 mm to 406 mm.

Blasting is accomplished with a high energy bulk emulsion with a density of 1.2 kg/m^3 to generate a powder factor of 0.30 kg/t for ore and 0.27 kg/t for waste. Blast holes are initiated with electronic detonators and primed with 450 g boosters. The bulk emulsion product is a gas sensitized pumped emulsion blend specifically designed for use in wet blasting applications.

Field estimates of intact rock strength range from approximately R3 to R4 (medium strong to strong) for the lower member ores while unit 61 (basal silicates) ranges from R0 to R2 (extremely weak to weak) according to the Brown classification outlined in Table 16.11.

Table 16.11: Classification of Rock Strengths (Brown, 1980)

Grade	Description	Field Identification	Approximate Range of UCS (MPa)
R6	Extremely Strong Rock	Specimen can only be chipped with geological hammer	>250
R5	Very Strong Rock	Specimen requires many blows of geological hammer to fracture it	100 – 250
R4	Strong Rock	Specimen requires more than one blow of geological hammer to fracture it	50 – 100
R3	Medium Strong Rock	Cannot be scraped or peeled with a pocket knife, specimen can be fractured with single firm blow of geological hammer	25 – 50
R2	Weak Rock	Can be peeled by a pocket knife with difficult, shallow indentations made by firm blow with point of geological hammer	5 – 25
R1	Very Weak Rock	Crumbles under firm blow with point of geological hammer and can be peeled by a pocket knife	1 – 5
R0	Extremely Weak Rock	Indented by thumbnail	0.25 – 1

The average drill productivity for the production rigs is estimated at 19.4 m/h with an instantaneous penetration rate of 43 m/h. The overall drilling productivity includes time lost in the cycle when the rig is not drilling such as move time between holes, moves between patterns, drill bit changes, etc. The average drilling productivity inclusive of a 5% redrill factor is estimated at 4,794 t/h in ore and 4,046 t/h in waste due to the different material densities. Drill & blast parameters are detailed in Table 16.12.

Blasting activities are outsourced to an explosives provider who is responsible for supplying and delivering bulk explosives in the hole. The mine engineering department is responsible for designing blast patterns and relaying hole information to the drills via the wireless network.

Blasthole loading and firing activities are performed during day shifts only. When in full production, the blasting team will consist of eight individuals as follows:

- 1x blaster and 1x blaster helper per crew (2 crews on rotation by explosives provider).
- 1x bulk truck operator per crew (2 crews on rotation by explosives provider).
- 1x stemming loader operator (2 crews by Owner).

All accessories and blasting consumables are purchased through the bulk explosive supplier. Budgeting of explosives products and services is based on agreements with the current supplier. The cost for blast movement monitoring has been included in the blasting cost.

In this Feasibility Study, a mobile 18 t emulsion repump truck is planned which will be loaded from two emulsion storage tanks leased from the explosives provider. The explosives magazines present on site are adequate and will continue to be used.

Table 16.12: Drill & Blast Parameters

Drill and Blast Parameters		Ore	Waste
Drill Pattern			
Explosive Density	g/cm³	1.2	1.2
Hole Diameter	mm	349	349
Burden (B)	m	9.25	9.25
Spacing (S)	m	9.25	9.25
Subdrill (J)	m	1.80	1.8
Stemming (T)	m	5.00	7.00
Bench Height (H)	m	12.0	12.0
Blasthole Length (L)	m	13.8	13.8
Pattern Yield			
Rock Density	t/bcm	3.33	2.81
BCM/hole	bcm/hole	1,027	1,027
Yield per Hole	t/hole	3,419	2,885
Yield per Meter Drilled	t/m drilled	248	209
Powder Factor	kg/t	0.30	0.21
Weight of Expl. per Hole	kg/hole	1,012	782
Drill Productivity			
Re-drills	%	5%	5%
Pure Penetration Rate	m/h	43.0	43.0
Overall Drilling Factor (%)	%	45%	45%
Drilling Productivity	m/h	19.4	19.4
Drilling Productivity	t/h	4,794	4,046
Drilling Efficiency	holes/h	1.40	1.40

Final wall blasting practices will be very similar to production blasting and will not involve pre-split holes. Pre-split blasting practices are not recommended due to the incompetent nature of the rock mass. The spacing on the back holes will be reduced and all back row holes will be drilled without subgrade with a lighter charge and no stemming. Along the wall, there will typically be only one hole fired per delay. The final wall blast will be carried out as trim blasts with no more than four rows deep and blasting to a free face.

16.4.3 Ore Control

The ore control program will involve establishing dig limits for ore and waste in the field to guide loading unit operators. A high precision system combined with an arm geometry system will allow shovels to target small dig blocks and perform selective mining. The system will give operators real-time view of dig blocks, ore boundaries, and other positioning information. Physical flagging in the field will also be implemented.

The ore control boundaries will be established by the Technical Services department based on grade control information obtained through blast hole sampling with post-blast boundaries adjusted for blast movement measurements made using a BMM® system. A blast movement monitoring system has been included in the blasting cost for 20% of the blasts.

Table tests are conducted to establish a weight recovery for blast hole samples. With the exception of push-backs where waste lithology is encountered, the goal is to test every sample; blasthole samples (excepting obvious waste samples) are assayed individually to estimate crude iron, manganese and magnetite.

16.4.4 Loading

The majority of the loading in the pit will be done by one electric drive and two diesel drive hydraulic front shovels equipped with a 24 m^3 or 17 m^3 bucket. The shovels are matched with a fleet of mine trucks with a 221 t payload capacity. The project already owns two 24 m^3 hydraulic front shovels (one electric and one diesel) and one 17 m^3 diesel front shovel. The hydraulic shovels are complemented by one production front-end wheel loader (“FEL”) with a 21 m^3 bucket.

Although interchangeable, the hydraulic shovels will primarily be operating in ore and waste while the wheel loader will primarily be used for re-handling operations. The loading units have been optimized to benefit from the lower operating cost of the electric front shovel, while reducing as much as possible the movement of loading units in each loading areas. The loading productivity assumptions for all types of loading tools in ore, waste and overburden are presented in Table 16.13.

In ore, the 24 m^3 shovels are expected to achieve a productivity of 3,557 t/h based on a 4-pass match with the mine trucks and an average load time of 2.6 minutes. The productivity in waste will decrease to 3,027 t/h due to the lower density of the material handled.

The 17 m^3 shovel is expected to achieve a productivity of 2,573 t/h based on a 5.5-pass match with the mine trucks and an average load time of 3.50 minutes in ore. The productivity in waste will decrease at 2,199 t/h due to the lower density of the material handled.

The wheel loader is expected to achieve a productivity of 2,950 t/h based on a 5-pass match and an average load time of 3.30 minutes. In waste, the loader productivity decreases to 2,639 t/h due to the lower density of the material handled.

A 13.5 m^3 wheel loader is planned, being matched with two 100 t class trucks in order to cover the rehandling and complement the main fleet as required.

Table 16.13: Main Fleet Loading Specifications

Loading Unit		Shovel (24 m³)	Shovel (24 mᶜ)	Shovel (17 m³)	Shovel (17 m³)	FEL (21 m³)	FEL (21 m³)
Haulage Unit		**Truck (221 t)**	**Truck (221 t)**	**Truck (221 t)**	**Truck (221 t)**	**Truck (221 t)**	**Truck (221 t)**
Material		**Ore**	**Waste**	**Ore**	**Waste**	**Ore**	**Waste**
Rated Payload	t	221	221	221	221	221	221
Heaped Volume	m^3	147	147	147	147	147	147
Bucket Capacity	m^3	24	24	17	17	21	21
Bucket Fill Factor	%	90%	90%	90%	92%	85%	90%
In-Situ Dry Density	t/bcm	3.33	2.81	3.33	2.81	3.33	2.81
Moisture	%	3%	3%	3%	3%	3%	3%
Swell	%	40%	40%	40%	40%	40%	40%
Wet Loose Density	t/m^3	2.45	2.07	2.45	1.69	2.45	2.07
Actual Load Per Bucket	t	52.92	44.65	37.48	25.86	44.56	39.82
Passes (Decimal)	#	4.18	4.95	5.90	8.55	4.96	5.55
Passes (Whole)	#	4.00	4.00	5.5	7.00	5.00	5.50
Actual Truck Wet Payload	t	212	201	206	221	223	219
Actual Truck Dry Payload	t	206	195	200	215	216	213
Actual Heaped Volume	m^3	86	97	84	107	91	106
Payload Capacity		96%	91%	93%	100%	101%	99%
Heaped Capacity		59%	66%	57%	73%	62%	72%
Cycle Time							
Truck Exchange	min	0.70	0.70	0.70	0.70	0.60	0.60
First Bucket Dump	min	0.10	0.10	0.10	0.10	0.10	0.10
Average Cycle Time	min	0.60	0.60	0.60	0.60	0.65	0.65
Load Time	min	2.60	2.60	3.50	4.40	3.30	3.63
Cycle Efficiency	%	75%	75%	75%	75%	75%	75%
Number of Trucks Loaded/h	#	17.31	17.31	12.86	10.23	13.64	12.41
Production / Productivity							
Avg. Prod. dry tonnes/h	t/h	3,557	3.375	2,573	2,199	2,950	2,639

16.4.5 Hauling

Haulage will be performed with 221 t class mine trucks. The truck fleet productivity was estimated using Talpac3D software. Cycle times have been digitized and estimated for each period and for each dumping location depending on the type of material. The assumptions and input factors for the Talpac simulations are presented in Table 16.14, Table 16.15 and Table 16.16. A speed limit of 50 km/h was applied except for the travel down ramps where the speed was limited to 30 km/h.

Table 16.14: Speed Limits

Site Location	Speed Limit (km/h)
Pit on Working Bench and near Dump Face	50
Downhill Ramp < -5%	30
Mine Road and Ramps	50

Table 16.15: Rolling Resistance

Road Type	Rolling Resistance (%)
Main Road	2.50
Ramp	3.00
Pit Floor and near Dump Face	3.50

Table 16.16: Cycle Time Components

Cycle Time Component	Ore Cycle (min)	Waste Cycle (min)
Truck Average Load Queue Time [1]	1.76	2.03
Truck Average spot Time at Loader	0.85	0.85
Truck Average Loading Time	2.60	3.20
Truck Average Spot Time at Dump	0.85	0.85
Truck Average Dumping Time	0.74	0.74

Note 1: Average Load Queue Time corresponding to ½ spot and loading time

Fuel consumptions were also estimated using Talpac which generates a specific engine load factor depending on the proportion of travel on ramp grades and on flatter gradients. Generally, the fuel burn rate increases with depth as a longer period of time is spent on grade.

Figure 16.17 presents the results from the haulage study. When cycle times decrease going forward in the schedule, it is due to the mining of benches higher in the pit when new mining phases are initiated.

The total haul hours required by period determines the number of trucks required throughout the LOM as shown in Figure 16.18. The truck fleet reaches a maximum of 15 units in 2022 and remains at this level until 2032, then drops to 14 trucks up until 2038 when it starts to decrease as a result of a drop in mining rate.

Figure 16.17: Average Cycle Times by Pit

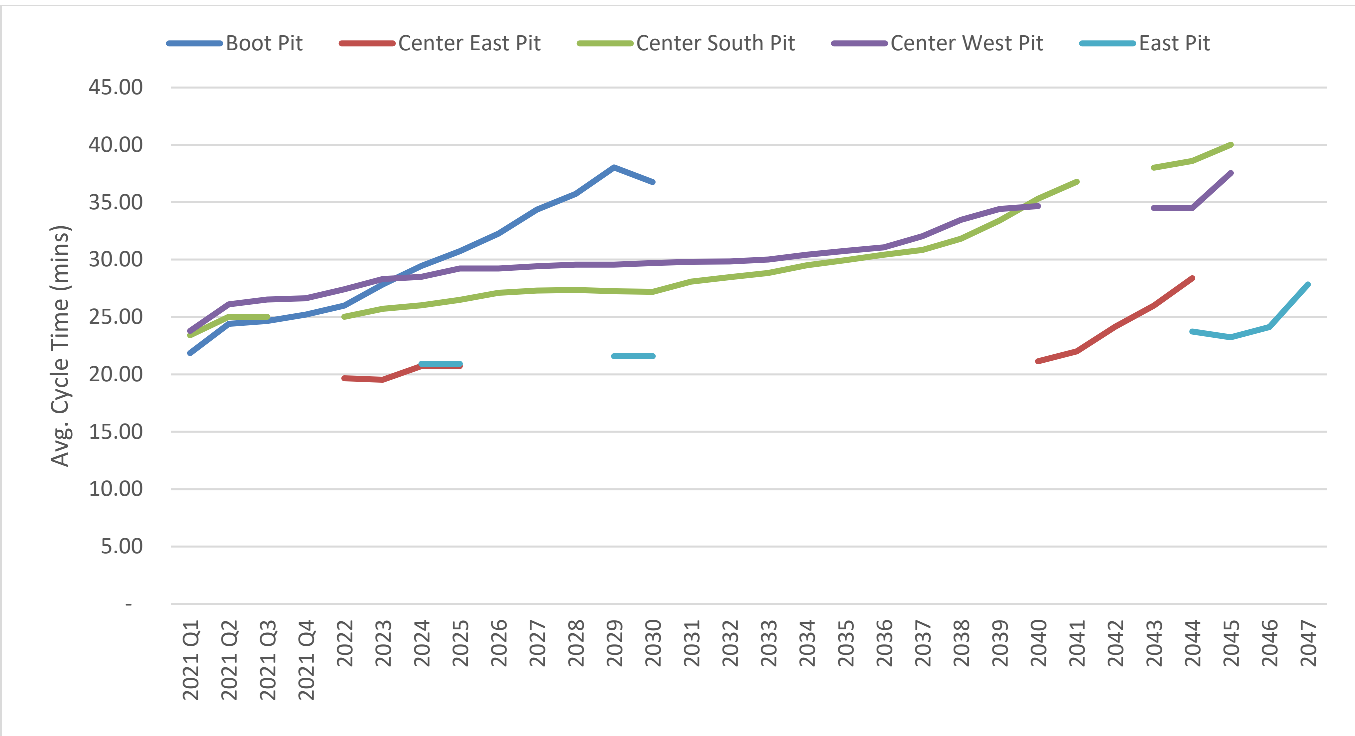
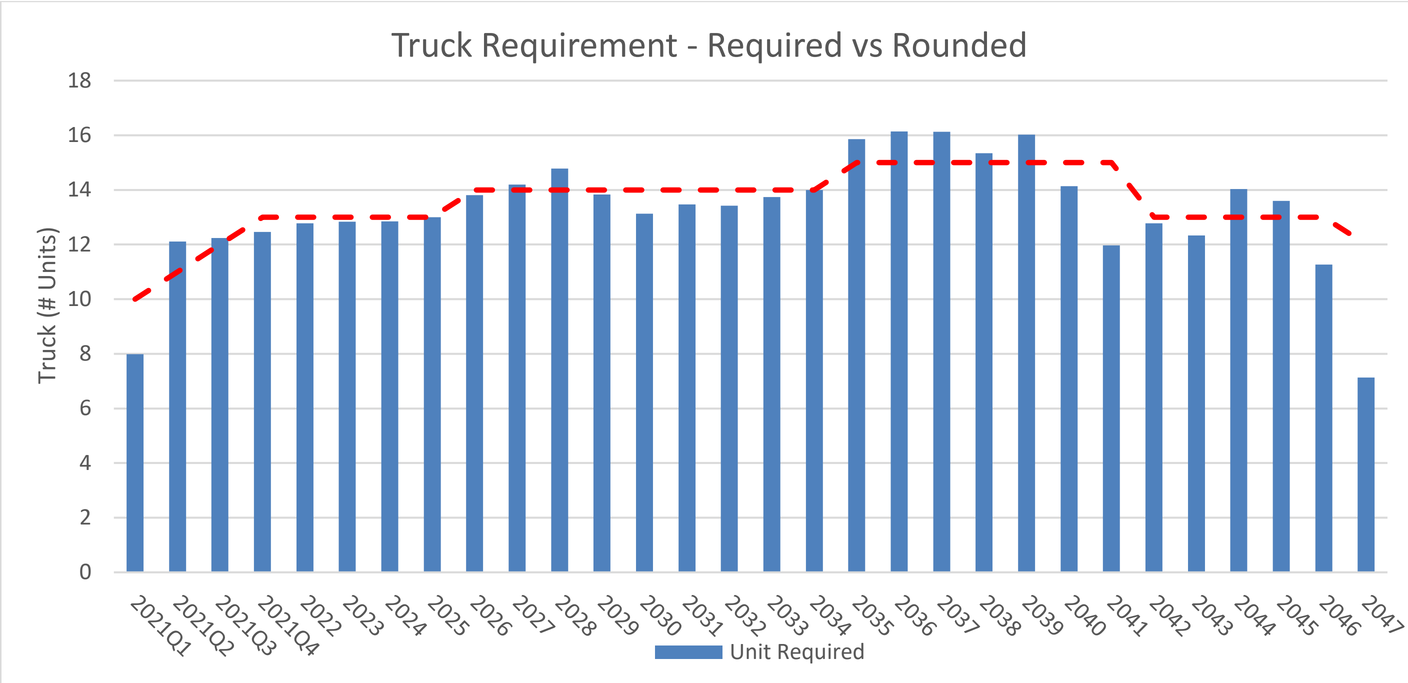


Figure 16.18: Truck Requirement - Required vs. Budgeted



16.4.6 Dewatering

16.4.6.1 Previous Dewatering Experience at Wabush

Major water inflows were observed in the Scully pits. At the mine closure in 2014, twenty dewatering wells were active between the West Pit Extension and Duley Lake. An additional 10 inactive wells are also present between the West Pit Extension and Duley Lake. Pumping rates before the closure of the mine in 2014 were of the order of 26,500 GPM in the West Pit alone (Amec, 2014). At the same time, dewatering rates in the East-Pit-East and East-Pit-West were of 4,000 and 4,500 USGPM respectively (Amec, 2014). Dewatering in the East-Pit-East and East-Pit-West was accomplished using a series of pit sumps (Golder, 2006a).

16.4.6.2 Hydraulic Connection with Surrounding Lakes

Golder (2006b) carried out a geophysical survey to assess the lake bottom conditions at Duley Lake adjacent to the West Pit Extension and at the Little Wabush Lake adjacent to the East-Pit-East.

The survey identified anomalies in the geophysical data, which were identified as potential sinkholes that may connect Duley Lake with the underlying permeable iron ore bearing units.

The geophysical survey did not identify any anomalies indicative of sinkholes in the Little Wabush Lake.

Knoll and Vern Lakes are assumed to be disconnected or weakly connected to the mine's hydrogeological system. This is suggested by the reported negligible final dewatering rate of the South Pit (0 USGPM, Amec, 2014), despite the large elevation difference between the lakes (Vern Lake: 557 masl, Knoll Lake: 540 masl) and the bottom of the existing South Pit (469 masl). In addition, Vern Lake is underlain by the Denault and Wishart formations, which will not be intersected by the projected pit, and Knoll Lake is underlain by the Wishart and Attikamagen Formations, which will not be intercepted by the projected pit. Moreover, Knoll Lake rests to the east of the north-south fault that separates the West Pit from the East-Pit-West. This fault has been interpreted as a groundwater barrier by Piteau (2001).

Similarly, Quartzite Lake, located north of the East-Pit-East and East-Pit-West, at an elevation of 538 masl, is underlain by the Wishart Formation and hence is considered as hydraulically disconnected from the geological units intercepted by the proposed pit expansion.

Figure 16.19: Flooded Pits in 2018

16.4.6.3 Groundwater Model

A 3D numerical groundwater model was created using FEFLOW, a finite element groundwater software that solves the groundwater flow equation. The model was divided into six units, corresponding to the geological units used on site:

- Upper Member
- Lower Member
- Middle Member
- Basal Silicate

- Middle Quartzite
- Wishart Quartzite

The lakebed sediments under Duley Lake were analyzed using a permeameter, which yielded an average hydraulic conductivity of 1.6×10^{-7} m/s (Piteau, 2001). The lakebed sediments are underlain by a till with an unknown hydraulic conductivity, which was assumed to be greater than the lakebed sediments. The model lumped the lakebed sediments and the underlying till into a single layer located below Duley Lake and Little Wabush Lake. This layer was assigned a hydraulic conductivity of 1×10^{-6} m/s and a thickness of 3 m. At Duley Lake, the sinkholes observed during the GPR survey were introduced as elements with the same hydraulic conductivity as the underlying rock formations. Vern and Knoll Lakes were not included in the model since they are assumed to be perched and therefore not connected to geological units present at the mine.

The model was calibrated to match the dewatering rates reported by Amec (2014) prior to mine closure. The model was then used to estimate the steady-state groundwater inflow into the mine once the final target depth is reached at all pits.

The hydraulic properties of the rock at the South Pit, Boot Pit and West Pit are assumed to correspond to those estimated by the calibration of the model at the West Pit Extension.

Two cases were evaluated using the groundwater model: a base case with the calibrated hydraulic conductivities and an upper range case, which used the 80^{th} percentile of the available hydraulic conductivities for units in the East-Pit-East and East-Pit-West (Table 16.17).

The mining sequence was not considered in these estimates and inflow may increase for a given area depending on the mining sequence. A preliminary modelling exercise that considered deepening of the East-Pit-West before deepening the East-Pit-East suggests that if this were to occur, infiltration into the East-Pit-West would be approximately 6,600 USGPM.

Groundwater modelling simulations have shown that the operation of the dewatering system at West Pit Extension would control most of the water inflow coming from Duley Lake. Given that the maximum proposed depth of the deepened West Pit will be similar to that of the existing West Pit Extension (West Pit is planned to be 12 m deeper), large ground water inflow is not expected at West Pit.

Table 16.17: Groundwater Model Results

Pit	Base Case (USGPM)	Upper Range (USGPM)
East-Pit-East	6,600	9,700
East-Pit-West	3,500	7,800
South Pit	3,600	3,600
West Pit Extension	26,500	26,500
West Pit	400	400
Boot Pit	6,200	6,200
Total	**46,800**	**54,200**

16.4.6.4 Dewatering

In the 2018 Study, three (3) distinct dewatering activities were planned. The first activity consisted in the dewatering of the flooded pits to restart mining and allow for in-pit dumping. The first pits to be dewatered were the West and South Pits. Water was pumped using barges and submersible electrical pumps. The East Pit was dewatered using the same methodology. The quantities to subsequently dewater are presented in the Table 16.18.

Table 16.18: Water Quantity in Flooded Pits

Pit Location	000 m^3
West & South Pit	25,000
East Pit	31,150

The second dewatering activity to take place was the long hole dewatering. All the infrastructure is present on the western end of the pit along the Long Lake frontier. It consists of a series of long holes pumping water seepage from the Long Lake into the West Pit. The water pumped is sent back into the Long Lake through a series of pipes. The Eastern infrastructures are yet to be built but an extensive study was done by Piteau Associates Engineering in 2005. The study determined the number of holes to be drilled and the cost for the drilling of the holes plus all required infrastructures. The long hole dewatering will be performed with Gould Pumps; numerous pumps are found on site.

Finally, sump pumps will be installed at pit bottoms or on working benches when water is present. Numerous electrical submersible pumps are still present on site. Water pumped in the pit, will first be sent to a sedimentation pond before being released in the surrounding lakes.

16.4.7 Road and Dump Maintenance

Waste and ore storage areas are maintained by a fleet of two 630 HP track-type dozers and two 350 HP dozers. Also, a 500 HP wheel dozer is available on site, mainly dedicated to the maintenance of the mine roads and the loading areas.

Mine roads are maintained by one 18 ft blade grader and one 16 ft blade motor grader to be replaced by a second 18 ft blade unit. A water/sand truck will be used in summer time to suppress dust by spreading water. During the winter months, the unit will be converted into a sand truck to spread road aggregate for added traction on snow/ice.

For snow removal activities, a 350 HP loader will be used either with a snow blade or with a snow blower for cleaning around buildings and in critical areas. If needed, an IT loader from the maintenance department can give additional support.

16.4.8 Support Equipment

All miscellaneous construction related work, such as berm construction and water ditch cleaning, can be done by one available 49 t excavator. This unit is equipped with a hydraulic hammer for boulders clearing after blasts, if required.

For open pit service, one mechanic service truck, one fuel truck, and one lowboy are available . Four light towers are required to illuminate critical workplaces such as at the loading face, stockpile area, and waste dump points. In other places, the electrical network of the pit is used for lighting.

Several other pieces of equipment are planned to support the mining activities: one boom trucks (28 t crane), one 230 HP IT loader, one small 5 t forklift for the maintenance facility. A MT4400 tractor (200 t class haul truck) with a low-boy trailer is also on site and available for moving tracked equipment around the property.

16.4.9 Mine Maintenance

Tacora has not entered a maintenance and repair contract (“MARC”) for its full mobile equipment fleet. Consequently, the maintenance department has been structured to support this function, performing maintenance planning and training of employees. For activities such as major components repairs, the project will rely on the local dealer and manufacturer support. Thus, a provision for an external contract was budgeted as a yearly contractor fee.

All required tire services for the mining equipment (such as tire pressure and wear monitoring, tire rotation, tire replacement and repairs) is outsourced and the contractor fees have been budgeted as part of the maintenance budget.

Since specialized tooling is required to perform the regular maintenance (lift stands, kidney looping units, machines diagnostic tools, hydraulic torque wrenches and general shop tools such as presses, nitrogen charging kits, air tools), the cost for the replacement and repair of those tools has been included in the operating expenditures throughout the mine life.

A Computerized Maintenance Management System ("CMMS") is necessary to manage maintenance and repair operations. This system tracks all maintenance costs while keeping a service history of each machine.

16.4.10 Mine Management and Technical Services

The mine is headed by a Mine Manager who is responsible for the overall management of the mine. Superintendent positions in operations and maintenance report directly to the mine manager. Technical Services functions including geology and mine planning report to the Manager, Technical Services.

The operations department consists of one foreman and a mine dispatcher per crew (4 in total). To increase operator level performance and organize structured training programs, a mine trainer is planned on day shift for the first three years of the project. Training after the three year period will be outsourced.

The engineering and geology team will provide support to the operations team by providing short-term and long-term planning, grade control, surveying, mining reserves estimation and all other technical functions. Operating costs for this group includes salaries, office supplies, software fees, survey and grade control supplies. The engineering and geology team includes 6 employees.

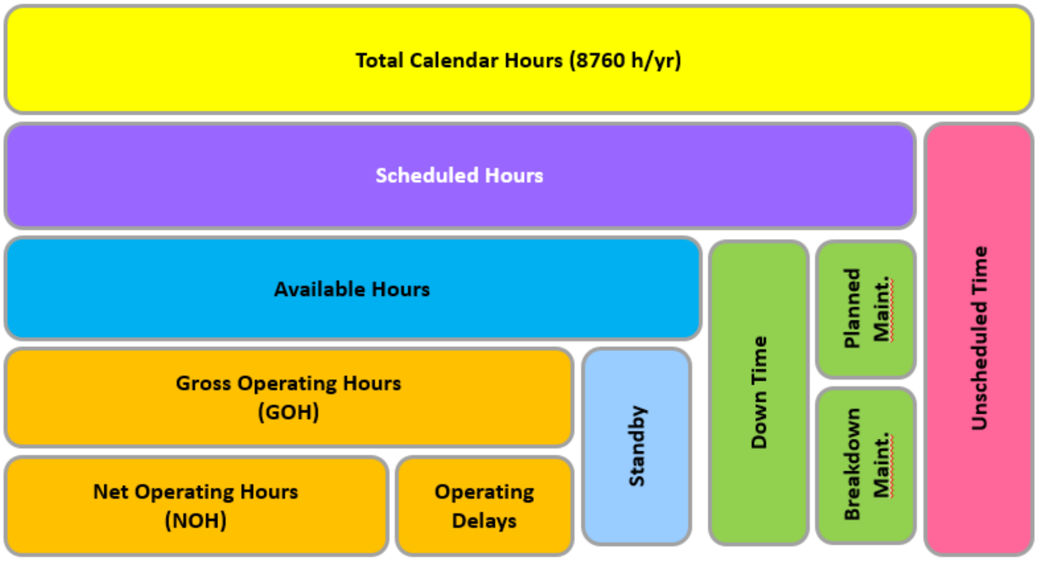
16.4.11 Roster Schedules

The workforce is being sourced locally from neighboring communities such as Fermont (Qc), Labrador City (NL) or Wabush (NL). Except for some administrative positions which are on a standard 8 hour shift 5 days on / 2 days off, the rest of the crews are working 12-hour shifts with a 4 on / 4 off rotating schedule. Four crews are required to operate on a continuous basis 24-hours per day 365 days per year. This schedule for rotational employees results in 2,184 scheduled hours of work per year.

16.4.12 Equipment Usage Model Assumptions

The equipment hour usage model utilized to plan equipment requirements and productivity is illustrated in Figure 16.20. The typical equipment usage model assumptions are established by equipment groupings as presented in Table 16.20. The annual net operating hours ("NOH") varies approximately between 5,000 and 5,400 h/yr.

Figure 16.20: Equipment Hour Usage Model



The definitions are as follows:

- Calendar hours – defined as 8,760 h/y.
- Unscheduled hours – scheduled outages including statutory holidays, planned shutdowns and scheduled down shifts
- Scheduled hours – total calendar hours less unscheduled hours
- Down hours – time unit is mechanically inoperable due to preventive maintenance or unscheduled breakdowns
- Available hours – is scheduled hours less down time
- Standby hours – time the equipment is available but not manned or used
- Gross operating hours (GOH) – available hours less idle or standby
- Operating delay hours - time the unit is available, manned, but not producing or effective
- Net operating hours (NOH) – the difference between GOH and operating delays

KPI Calculations:

- NOH = Net Operating Hours
- GOH = NOH + Delay
- Total Hours = NOH + Delay + Standby + Down
- Availability = (NOH + Delay + Standby) / (NOH + Delay + Standby + Down)
- Use of Availability = (NOH + Delay) / (NOH + Delay + Standby)
- Net Utilization = (NOH + Delay) / (NOH + Delay + Standby + Down)
- Effectiveness = NOH / (NOH + Delay) = NOH / GOH
- Overall Equipment Efficiency (OEE) = NOH / Total Hours

Mining Services	**Equipment Hour Usage Model**	**Source: G Mining Date: Aug. 2016**

Table 16.19: Equipment Usage Model Assumptions

Equipment Usage Assumptions		Shovels	Loaders	Trucks	Drills	Ancillary
Days in period	days	365	365	365	365	365
Weather, Scheduled outages	days	5	5	5	5	
Availability	%	82	80	85	85	85
Use of Availability	%	90	90	80	90	85
Utilization	%	72	72	68	72.3	72.3
Effectiveness	%	87	85	80	85	80
OEE	%	64.2	61.2	54.4	61.2	57.8
Total Hours	h	8,640	8,640	8,640	8,640	8,640
Down Hours	h	1.555	1,728	1,296	1,728	1,314
Delay Hours	h	829	933	175	933	1,266
Standby Hours	h	708	691	1469	691	1,117
Gross Oper. Hours (GOH)	h	6,376	6,221	5,875	6,221	6,329
Net Oper. Hours (NOH)	h	5,547	5,288	4,700	5,288	5,063

16.4.13 Fleet Management

A fleet management system (including hardware and software) manages the operation in real-time, monitor machine health and track key performance indicators ("KPIs"). The system is managed by a dispatcher on each crew who controls the system which sends operators onscreen instructions to work at peak efficiency.

The shovels are fitted with a high precision global positioning system ("GPS"). This system improves the productivity and bench grade control, and can be integrated with the mine planning software.

The production drills are also equipped with a high precision system. This system will help to guide rigs into position and assure holes are drilled to the correct depth and location.

Both capital expenditures ("CAPEX") and operating expenditures ("OPEX") have been budgeted in the Study and are detailed in Section 21.

16.4.14 Pit Slope Monitoring

Slope movement monitoring is planned for the open pit to gather measurements and confirm stability in order to ensure safe working conditions. Initial slope movement monitoring consists in using prisms read by manual or automated surveys with at least two permanent total stations established in climate controlled enclosures to ensure full coverage of the open pit. The initial prism monitoring provides movement response data to verify visual observations and confirm if the slope is performing adequately.

A permanent, fully-automated monitoring system using Lidar or radar could be considered for the long term if warranted based on review of prism monitoring during the initial years.

The slope movement monitoring data is important for the calibration of numerical models required for slope design updates during the mine life. The pit phasing approach will allow for adjustments to the final pit design based on observations and knowledge gained during the interim pit phases.

16.4.15 Electric Cable Handling

An evaluation of capital and operating costs was made for the mining cable management and electrification of the pit since the mine will operate one electric front shovel and two electric drills.

The operation of a 550 HP wheel loader with a removable cable reel attachment has been budgeted to handle the electric cables required for the various production units. The pit circumference is mostly already electrified by aerial lines, but a cost provision has been included to cover for modifications and/or additions to the aerial electric lines. The south aerial lines are first moved to allow for mine infrastructures. A permanent sub-station must also be relocated because of the south-west dump. Once mining starts in East pit east, the power lines to the north have to be relocated along with another permanent substation.

Several portable substations, skid mounted, are present on site.

16.4.16 Aggregate Plant

To support the project and the need for crushed rock of different granulometry, an external contractor was hired. This contractor supplies and operates the aggregate plant and all the associated costs have been calculated as a cost per tonne of crushed rock.

The main usage of crushed rock is for road capping (84% of total). The aggregate will be used to increase traction during winter months, and to maintain and repairs all the site roads during summer time. The other usage of importance is to produce aggregate for stemming all the blast holes (16% of total).

16.5 Mine Equipment Requirements

Most of the mine equipment is already purchased with the exception of the three additional trucks required to increase the hauling fleet to 15 mine trucks. Consequently, the CAPEX provides mostly for the replacement of the existing units and the repair of major components. For the used unit, their initial hour-meter reading was used to evaluate the time of replacement of each unit. The equipment requirements are presented in Table 16.20.

Table 16.20: Equipment Requirements

Equipment Requirement	2021	2022	2023-2032	2033-2038	2039-2041	2042-2044	2045-2046	2047
Major Equipment								
Electric Prod Drill	2	2	2	2	2	2	2	2
Electric Hydraulic Shovel (24 m³)	1	1	1	1	-	-	-	
Diesel Hydraulic Shovel (24 m³)	1	1	1	1	1	1	1	1
Diesel Hydraulic Shovel (22 m³)	1	1	1	1	1	1	1	1
Wheel Loader (20 m³)	1	1	1	1	1	1	1	1
Wheel Loader (13.5 m³)	-	-	1	1	1	1	-	-
Mining Truck (240 t)	13	13	14	15	15	13	13	12
Mining Truck (100 t)	-	-	2	2	2	2	-	-
Track Dozer (600 HP)	2	2	2	2	2	2	2	1
Track Dozer (350 HP)	2	2	2	2	2	2	2	1
Motor Grader 16 ft and 18 ft	2	2	2	2	2	2	2	2
Wheel Dozer 500 HP	1	1	1	1	1	1	1	1
Water/Sand Truck	1	1	1	1	1	1	1	1
Support Equipment								
Excavator (49 t)	1	1	1	1	1	1	1	1
Rock Breaker	1	1	1	1	1	1	1	1
Wheel Loader (4.7 m³)	1	1	1	1	1	1	1	1
Wheel Loader 425 HP	2	2	2	2	2	2	2	2
Wheel Loader 165 HP	1	1	1	1	1	1	1	1
Skid Steer Loader	3	3	3	3	3	3	3	2
Boom Truck 28 t	1	1	1	1	1	1	1	1
Forklift Diesel 4 t	1	1	1	1	1	1	1	1
Manlift 60 ft	1	1	1	1	1	1	1	1
Cable Reel Wheel Loader	1	1	1	1	1	1	1	1
Mechanic Service Truck	1	1	1	1	1	1	1	1
Welding Truck	1	1	1	1	1	1	1	1
Tire Handler Forklift	1	1	1	1	1	1	1	1
Fuel & Lube truck (7.5 kL) ADT	1	1	1	1	1	1	1	1
Tow Haul Truck	1	1	1	1	1	1	1	1

Equipment Requirement	2021	2022	2023-2032	2033-2038	2039-2041	2042-2044	2045-2046	2047
Trailer Lowboy 100 t	1	1	1	1	1	1	1	1
Pick-up Truck	5	5	5	5	5	5	5	3
Diesel Powered Air Heaters	2	2	2	2	2	2	2	2
Dewatering								
Long Hole Dewatering	55	55	55	55	55	55	55	55
In-Pit Dewatering	6	6	6	4	5	5	4	4

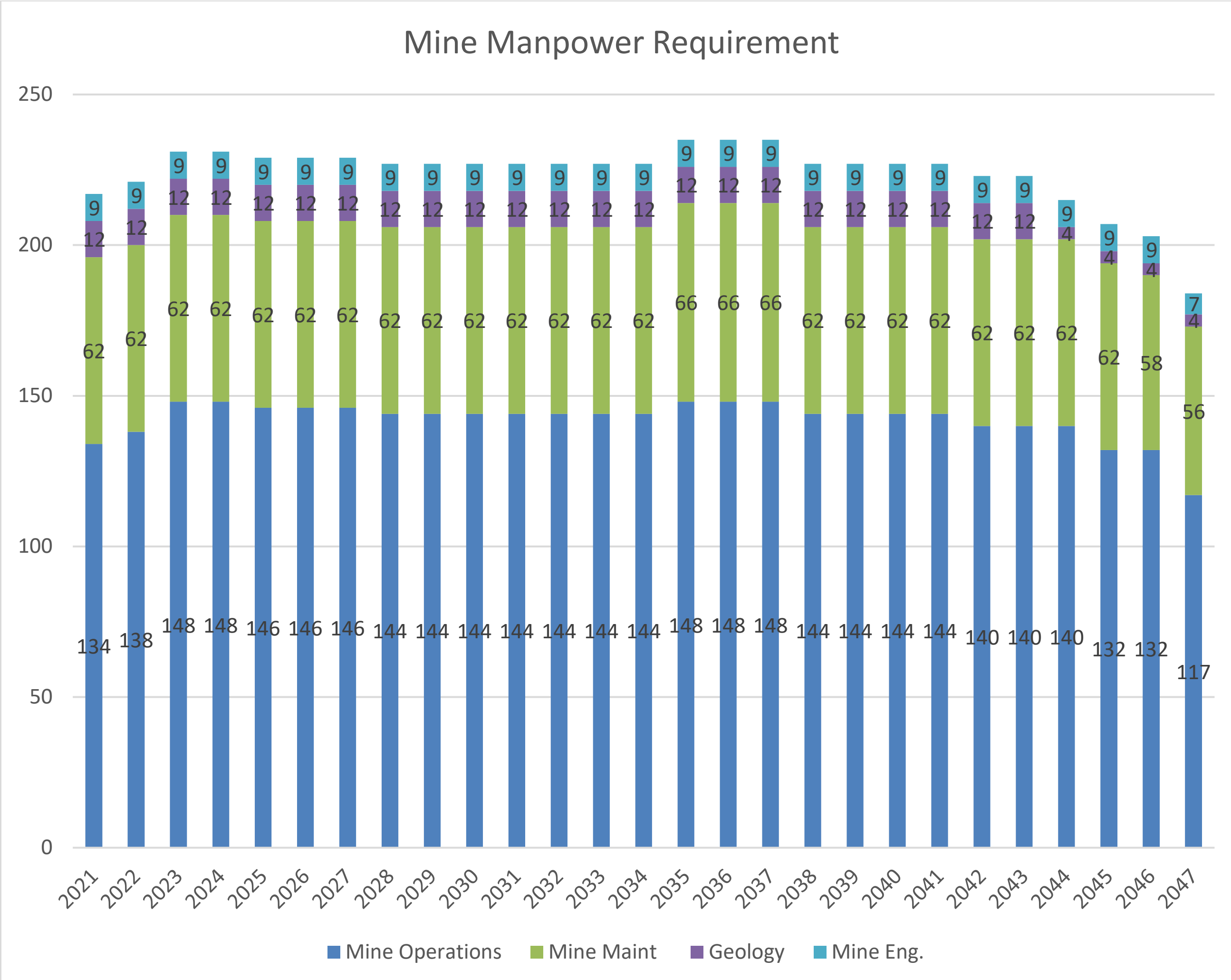
16.6 Mine Manpower Requirements

The mine manpower requirements over the LOM is represented in the Figure 16.21. The total mine operations team will vary over the mine life from 134 to a peak of 148 people. The peak is reached in 2035 then a reduction occurs as the yearly tonnages slowly decreases. In addition, contractors are planned for blasting activities.

The mine maintenance group will get to a maximum of 66 employees of which 11 are staff members, as shown on Figure 16.21.

The mine technical services group will have a total of 7 employees including mine planning, blast design surveying and geology.

Figure 16.21: Mine Manpower Requirements


Mine Manpower Requirement
250
200
150
100
50
0
2021 2022 2023 2024 2025 2026 2027 2028 2029 2030 2031 2032 2033 2034 2035 2036 2037 2038 2039 2040 2041 2042 2043 2044 2045 2046 2047
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Mine Operations
Mine Maint
Geology
Mine Eng.

17 RECOVERY METHODS

17.1 Introduction

The process plant parameters are determined based on the following:

- Current operation data.
- Existing equipment data.
- Studies made by Soutex for the evaluation of the Manganese removal circuit performances.
- Testwork by SGS on the wet circuit scavenger spirals and test work on the high-tension scavenger units realized in 2020 by Tacora Resources Inc ("Tacora").
- Testwork conducted by COREM in 2008 and 2012 (as part of Project T-787, T-919 and T-1298).
- Reports by Cliffs Natural Resources, the previous owner.
- Industry standard practices or relevant literature.

17.2 Design Criteria

For the remainder of 2021, the availability and mill throughput will be gradually increased as modifications of the equipment and process are implemented. The modifications should be completed by the end of 2021 so that in 2022 onward, the concentrate production target could be met. Table 17.1 presents the high-level Scully Mine design criteria once the modifications are completed.

Table 17.1: Scully Mine High-Level Design Criteria as of 2022

Parameter	Unit	Value	Sources
Mill Availability	%	89.5	A
Production Hours	hr/y	7,840	C
Feed Annual Tonnage	t/y	18,110,862	C
Concentrate Annual Tonnage	t/y	6,045,649	C
Fresh Feed Iron Grade	%	35.0	A
Fresh Feed Silica Grade	%	40.6	A
Fresh Feed Manganese Grade	%	2.6	A
Solids Feed Rate - Nominal per Mill	t/hr	385	B
Solids Feed Rate - Nominal Total	t/hr	2,310	C
Wet Circuit Concentrate Iron Grade	%	61.0	A
Wet Circuit Concentrate Silica Grade	%	7.2	A
Wet Circuit Concentrate Manganese Grade	%	2.9	C
Final Tails Tonnage	t/y	11,752,808	C
Final Tails Iron Grade	%	19.4	C
Final Concentrate Iron Grade	%	65.6	C
Final Concentrate Silica Grade (maximum)	%	2.8	A
Final Concentrate Manganese Grade (maximum)	%	2.0	A
Final Concentrate Weight Recovery	%	33.4	A
Final Concentrate Iron Recovery	%	62.6	C
Final Concentrate Tonnage	t/hr	771	C

Sources:

A: Tacora
B: Soutex Recommandations
C: Calculated Data
D: Historical Data

17.3 Process Flowsheet and Mass Balance

The Scully concentrator process flowsheet consists of a crushing and grinding circuit followed by a wet gravity separation circuit and finally, a dry concentration circuit to reduce the manganese and silica concentrations. The process flowsheet is presented in Figure 17.1. The mass balance is presented in Table 17.2 and the water balance is presented in Figure 17.2.

Figure 17.1: Scully Mill Flowsheet

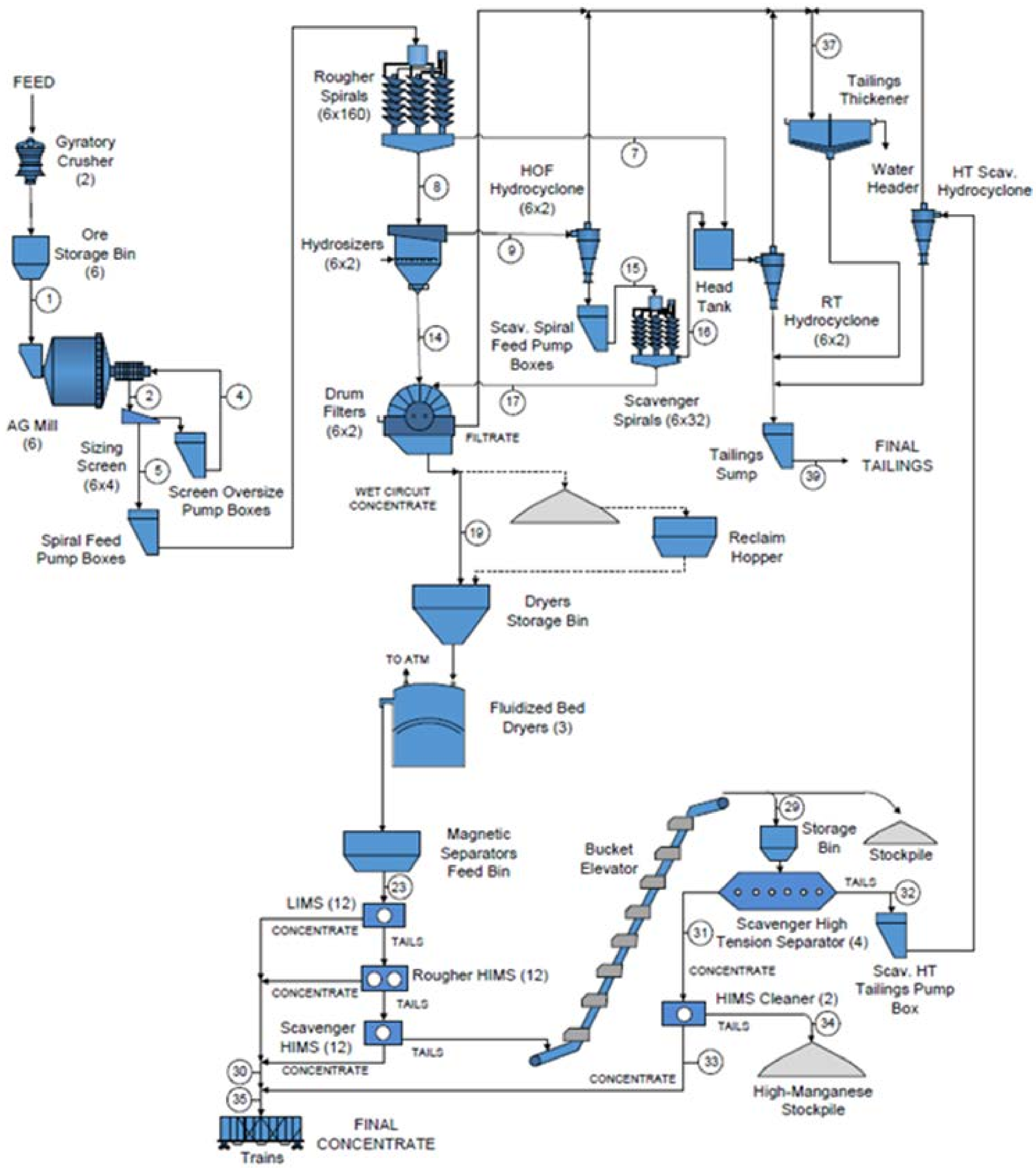
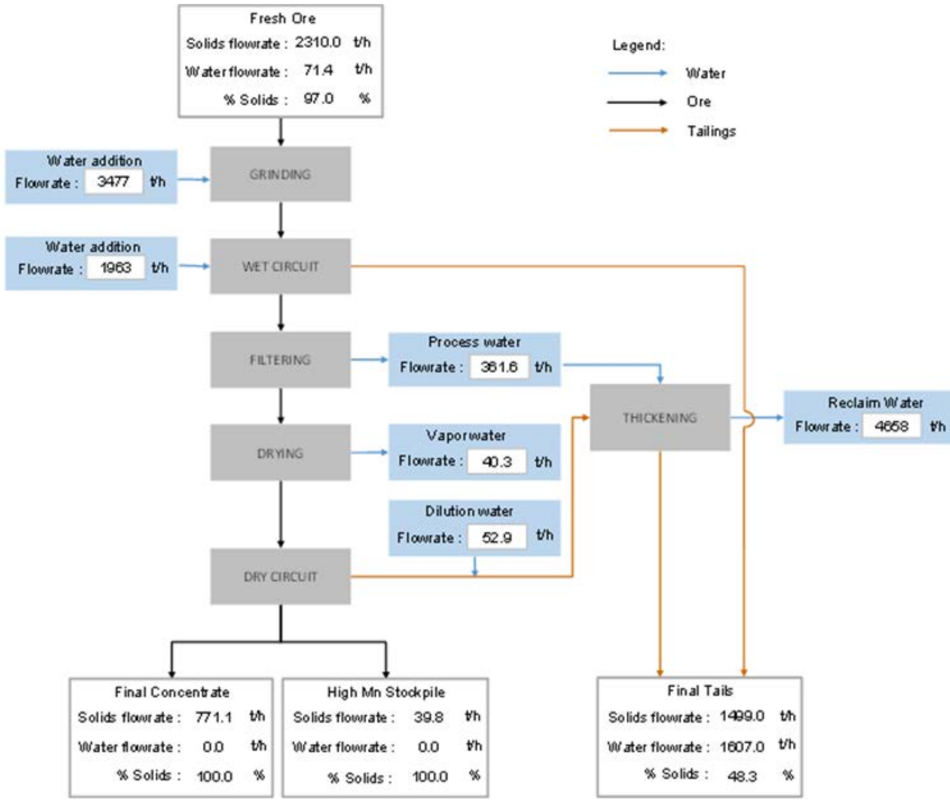


Table 17.2: Scully Mill Mass Balance

#	Description	Solids Flowrate t/h	Water Flowrate t/h	Slurry Flowrate t/h	Solids %	Slurry Volumetric Flowrate m^3/h	Solids sg -	Fe Grade %	SiO_2 Grade %	Mn Grade %
1	AG Mills Fresh Feed	2310	71	2381	97	721	3.6	35.0	40.6	2.6
2	AG Mills Discharge	3003	3670	6673	45	4515	3.6	35.0	40.6	2.6
4	Screen Oversize PB Discharge	693	693	1386	50	888	3.6	35.0	40.6	2.6
5	Screens Undersize	2310	3548	5858	39	4198	3.6	35.0	40.6	2.6
7	Rougher Spirals Tails	1317	4436	5753	23	4857	3.1	19.9	60.0	2.4
8	Rougher Spirals Concentrate	993	1285	2278	44	1514	4.3	55.0	14.8	2.9
9	Hydrosizers Overflow	199	1344	1543	13	1402	3.4	31.0	45.2	2.9
14	Hydrosizers Underflow	795	341	1135	70	511	4.6	61.0	7.2	2.9
15	Scavenger Spirals Feed	159	172	331	48	218	3.4	31.0	45.2	2.9
16	Scavenger Spirals Tails	99	243	342	29	276	3.0	13.0	68.9	2.7
17	Scavenger Spirals Concentrate	60	61	121	49	74	4.6	61.0	5.8	3.2
19	Drum Filters Cake	854	40	894	96	224	4.6	61.0	7.1	2.9
23	Dry Concentrate from Dryers	854	0	854	100	0	4.6	61.0	7.1	2.9
29	Scavenger HIMS Tails	101	0	101	100	31	3.3	26.6	39.5	11.0
30	MRC First Stage Concentrate	753	0	753	100	0	4.9	65.6	2.8	1.8
31	HT Scavenger 1st Stage Concentrate	58	0	58	100	15	3.7	40.5	6.8	17.0
32	HT Scavenger 1st Stage Tails	43	0	43	100	15	2.9	8.0	83.2	2.9
33	Cleaner HIMS Concentrate	18	0	18	100	4	4.8	63.5	2.8	1.8
34	Cleaner HIMS Tails	40	0	40	100	12	3.4	30.2	8.5	23.8
35	Final Concentrate	771	0	771	100	157	4.9	65.6	2.8	1.8
37	Tailings Thickener Feed	138	4914	5052	3	4958	3.2	21.7	58.3	2.6
39	Final Tailings	1499	1607	3106	48	2088	3.1	19.4	60.9	2.5

Figure 17.2: Simplified Mill Water Balance



17.3.1 Concentrate Crushing and Ore Handling

The ore is hauled by 240 short ton trucks. The trucks discharge into two (2) 54″ x 74″ primary gyratory crushers each driven by a 335 kW (450 hp) motor, at a nominal feed rate of 2,150 tph. Each crusher has a capacity of 3,861 tph so only one (1) unit is required to operate at a time. Two (2) hydraulic hammers (one for each crusher) can break down any oversize rocks in the crusher cavity. The ore is crushed to minus 8″ (203 mm) and transferred by two (2) 60″ belt conveyors to a transfer conveyor equipped with a tripper to feed the bins ahead of the grinding circuit. The transfer conveyor belt will be replaced to increase the mill feed availability.

The capacity of the crushing circuit is currently limited to 3,200 tph by the transfer conveyors. The crude ore storage bins have to be kept full at all time. Each transfer conveyor has an excess of 1,000 tph capacity compared to the AG mills combined throughput capacity to allow building the storage in the event of feed interruption.

A belt magnet system is present to remove the tramp metal and avoid stopping the transfer conveyors for metal removal on the belt.

The primary gyratory crusher auxiliaries include a crusher lubrication unit (including oil heaters and oil cooling fans), a spare main shaft and a complete set of relining platforms and liners removal pans. The crusher area is serviced by an overhead crane to facilitate maintenance and operation.

17.3.2 Grinding

The ore is fed by vibratory feeders from the 22,000 Mt capacity crude ore storage bins onto the mill feed conveyors. The conveyors each feed one (1) of the six (6) fully autogenous ("AG") mills. The mills are 24-foot diameter and eight (8) feet long. The gear of two (2) AG mills will be replaced to eliminate severe vibration problems and the mill lubrication systems will be improved on the six (6) AG mills.

The mills discharge through a trommel screen where the screen undersize is fed into four (4) wet vibrating screens with 1 mm width poly slots. Screen undersize is pumped to the spirals while the screen oversize is pumped back into its AG mill through the discharge end. These pumps are problematic, and wear becomes an important issue since the slurry contains mostly coarse particles and very little water. To ensure proper slurry displacement, a significant amount of water is added which reduces the grinding efficiency. The pump wear issue will be addressed in order to reach the targeted throughput.

17.3.3 Wet Concentration and Tails Thickener

The first stage of the wet concentration circuit is made of 160 rougher spirals per line supplied by Mineral Technologies in 1998. The spiral concentrate is processed in hydrosizers where the lighter waste material is removed from the denser iron-ore concentrate through the use of a concurrent upflow of water through a bed of concentrate. A total of twelve (12) 7-foot square Linatex hydrosizers (2 by line) were installed in 1997. The iron ore concentrate recovered at the underflow is sent to the drum filters. The hydrosizer tails are sent to a cyclone and its underflow is sent to new scavenger spirals to recover additional iron and increase the overall plant recovery.

The rougher spiral tails and the scavenger spiral tails are sent to a cyclone, where the overflow is directed to a 280-ft thickener to increase the tailings density to 35% solids. The cyclones underflow and the thickener underflow are sent to the tailings disposal facility.

The wet concentrate recovered from the hydrosizer underflow and the scavenger spirals is deposited on rotating drum filters (two (2) by line plus one (1) spare) where steam and vacuum pressure is used to remove excess water prior to drying the concentrate.

17.3.4 Dryers

The concentrate from the wet circuit is fed to the solid dryers where hot air is used to fluidize a bed of concentrate. The solid is heated indirectly by electrically generated steam and directly by Bunker C fuel burners.

The solid dryers were installed in 1963 and some work was completed before restarting the plant in 2019. The dryers' combustion system was upgraded to improve safety, reliability, efficiency, and ease of operation.

17.3.5 Dry Concentration

The dry concentrate is processed in a manganese removal circuit made of low intensity magnetic separators ("LIMS"), rougher and scavenger high intensity magnetic separators ("HIMS") followed by a high-tension separator and a cleaner HIMS. Each stage is used to separate hematite and magnetite from silica and pyrolusite to produce a lower manganese concentrate.

First, the material is processed in a LIMS where the low-manganese high-magnetite concentrate is obtained while the tails are processed in a rougher HIMS. The rougher tails that contain silica, pyrolusite and some remaining hematite are sent to the third stage of the Mn Removal Circuit, the Scavenger HIMS.

In the Scavenger HIMS, hematite is separated from silica and pyrolusite to produce the Scavenger HIMS Concentrate, while the Scavenger tails containing most of the silica and pyrolusite fall on the tails conveyor #20.

The dry tails from the Mn reduction circuit enter the High Tension ("HT") scavenger separators. The tails are rejected, and the concentrate is directed to Cleaner HIMS to obtain an additional final concentrate. The LIMS and HIMS concentrates are combined to obtain the final low-manganese concentrate. The cleaner HIMS tails are sent to a high-manganese stockpile which could have an economic value. Tacora plans to develop a sales strategy to sell and ship this material in future years.

The biggest concern with the high-manganese stockpile at the Scully Mine is fugitive dust. While the Scully Mine is operating, Tacora will have a program in place to ensure the stockpile is regularly wetted. If the manganese by-product is not sold as ore, the pile will be contoured and seeded at closure to establish a vegetative cover to prevent subsequent erosion and dust dispersion.

17.3.6 Concentrate Silo Load-Out

The concentrate from the dry concentration is transferred by a belt conveyor to the load-out silos over the rail track. The transfer conveyor is equipped with a belt scale to monitor plant production. From the load-out silos, the concentrate is loaded into a freight train for transportation to the port facilities. The load-out silos are equipped with a dust collection system. Out of five (5) 3,000 Mt capacity silos, four (4) were used as live storage to fill freight trains (up to 130 cars) while the fifth silo was used as manual top-off. The operation is based on 168 car freight trains. This represents more storage capacity than required, but it will provide the plant with additional buffer storage.

17.3.7 Tailings Management

The tails from the wet circuit and the dry circuit are processed in cyclones. The cyclones underflow streams are transported to the tailings area while the cyclone overflow streams are fed to a tailings thickener. The tailings thickener feed is dosed with flocculants to avoid fines build-up in the process water distribution system and to increase the sedimentation rate. The thickener overflow is directed to the process water tank.

The tailings thickener underflow, which is around 35% solids, is pumped to the tailings disposal pump box for pumping to the tailings pond. The slurry is pumped through three (3) lines each having two (2) pumps connected in series. In addition, on each line, there is a total of 10 booster pumps located along the pipeline between the plant and the tailings disposal area.

According to their evaluation, Hatch confirmed that in Phase 5 (Year 2024) a pumphouse with 12 pumps, two (2) pumps in series per tailings line will be installed to serve the extension up until 2030. In mid Phase 7 (Year 2030), a second pumphouse with 12 pumps, two (2) pumps in series per tailings line will be installed to serve the extension up until the end of mine life. Further engineering studies are recommended to optimize the timing of pump installations based on the deposition plan and mill production for each operating year. These studies may present cost savings and efficiency improvements over installing more pumps which reduce system reliability.

17.4 Instrumentation and Controls

Most of the existing instruments present before the plant shutdown are still in use at the concentrator. An inspection of the instruments was made prior to restarting the plant and if required, instruments were removed, repaired, and calibrated to achieve operational readiness.

Following the first months of operation since the restart, some instruments were identified as missing to achieve adequate process data monitoring and process control. A few instruments were identified as critical for the process and they will be replaced or added during 2021.

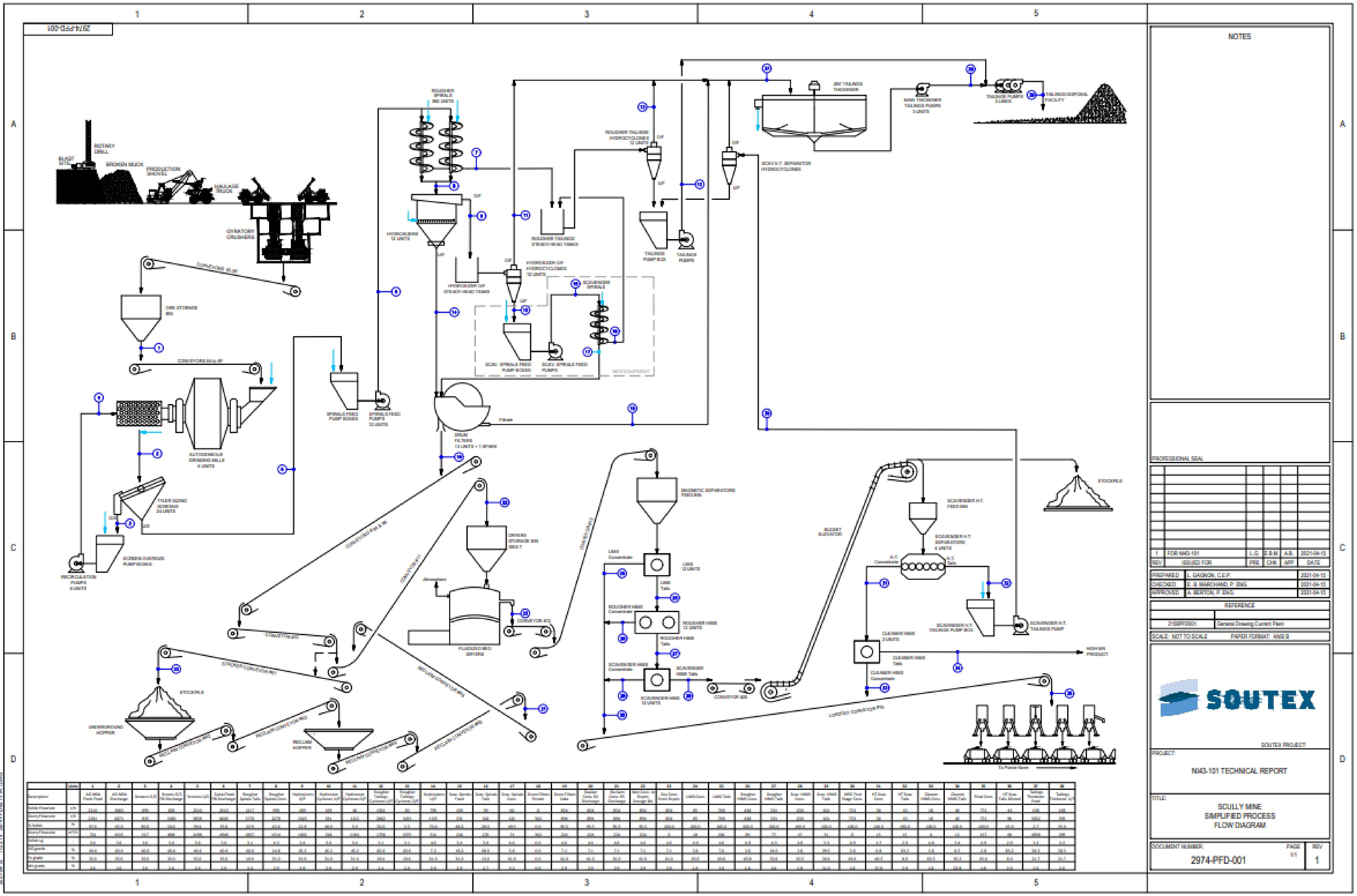
A flowmeter will be added to measure and control the amount of water added with the fresh ore into the autogenous ("AG") mills. Such flowmeter will allow adding the right amount of water with the ore to prevent the feed chute from plugging. Hence, this will avoid adding water in excess which would cause poor grinding and increase the number of fines being sent to the gravity circuit which in turn would decrease recovery.

The spiral feed flowmeters will then be replaced with new ones since the current ultrasonic flowmeters used to measure the spiral incoming flowrate are unreliable and inaccurate. To overcome this issue, the feed to the spirals was being controlled by velocity to the spirals, with more liberty to increase the velocity so there would be less sanding in the pipes. Hence, the amount of water added is not optimized and such a low density in the spirals harms the iron recovery. Also, the instruments present do not allow assessing accurately both flow and density which are required to ensure the good performance in the spirals and provide a steady and controlled flowrate to the spirals.

New scales will be installed on the mill feed conveyors to improve the accuracy of the process controls, reporting and mass balancing across the concentrator. This shall prevent discrepancies between the values obtained on the different conveyors and allow better mass balance reconciliation and process troubleshooting.

In order to achieve the target throughput, a new control system must be programmed to adjust the feed tonnage to the AG mills to optimize the throughput into the mills by aiming for the highest motor power usage. The mill throughput is currently controlled by the operators, who tend to constrain the mill throughput at low values for various reasons without prioritizing the optimization of the power draw. The operation data has shown that the current power utilization is low and there is ample potential for improvement.

Figure 17.3: Scully Mine Simplified Process Flow Diagram



18 PROJECT INFRASTRUCTURE

18.1 General

The vast majority of the existing plant infrastructure is in use. A detailed description is available in Section 5. Most of this infrastructure is more or less in similar condition to when the plant was shut down; nevertheless, some major upgrade work was performed at the concentrator prior to the plant restart.

18.2 Rail & Port Infrastructure

As during previous operations, transportation and handling of the concentrate is performed by rail using rented railcars.

18.2.1 On-Site Track

The track owned by Tacora Resources Inc. ("Tacora") includes all tracks and sidings up to the connection with the former Northern Lands Line which is now the property of Quebec North Shore and Labrador Railway ("QNS&L"). The railway network includes some internal siding tracks such as:

- Tank Farm track
- Loop track
- Car Shop Repair track
- Material Storage track.

18.2.2 Railcars

Four hundred and fifty-one (451) railcars are rented from *Société ferroviaire et portuaire de Pointe-Noire* ("SFPPN") and maintained on a cost-plus basis. Currently, Tacora operates three trains of 124-136 cars but the objective is to operate 2 trains of 168 cars. Both options can fulfill the 6 Mt of concentrates per year requirement.

The car loading and unloading is as per the original operations (top loading / bottom unloading using the existing hatch opening mechanism).

18.2.3 Main Railway

The on-site track connects to the QNS&L track at the Wabush Lake junction, east of the overpass over route H-500. Trains are brought to this junction by G&W and picked up and transported to Sept-Iles Arnaud junction under the responsibility of QNS&L.

18.2.4 Train Unloading and Ship Loading

Trains are picked up by SFPPN at Arnaud junction and moved to the existing unloading facilities at Pointe-Noire where the concentrate is unloaded and stockpiled. The concentrate is reclaimed by the reclaimer and then conveyed to either the existing dock #30 or the new multi-user dock and ship loader (#35). The refurbishment of existing equipment as well as the operations in Pointe-Noire are the responsibility of SFPPN. Tacora will invest USD 15.9 M in 2022 to support the planned improvements.

18.3 Buildings (Process and Non-Process)

18.3.1 Structural Repairs

Several areas of the plant will require a structural steel inspection. In 2012, the consultant BBA performed a detailed structural audit of the plant. The audit identified 480 items to be repaired and each item was assigned a risk rating and a repair timeline. Tacora has reviewed these critical items and advised that most (but not all) of the repairs have been completed. A full inspection of the structural components of the Plant is required to identify the works completed and the remaining items. This inspection could also identify additional items to be considered.

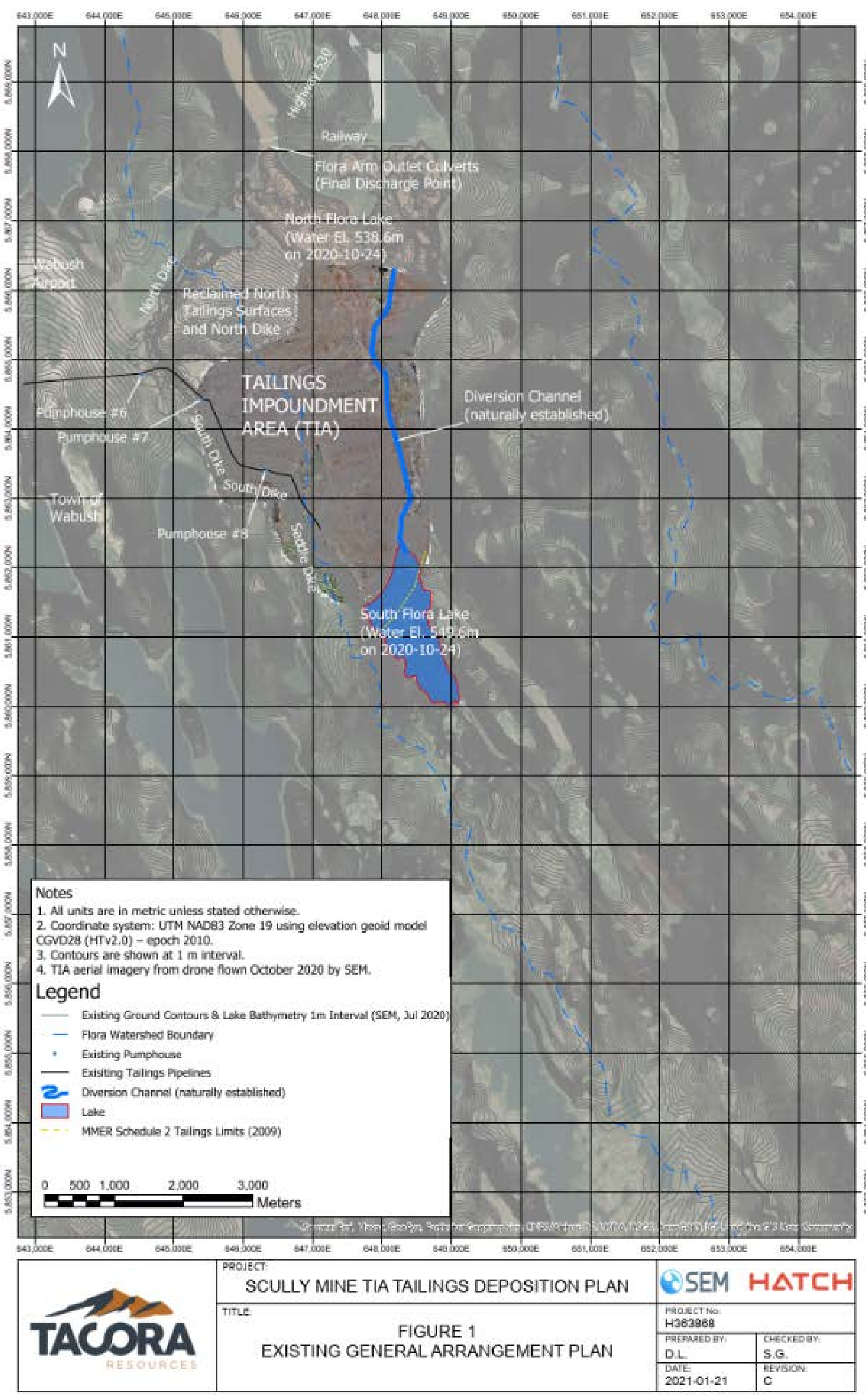
18.4 Tailings Impoundment Area (TIA)

18.4.1 Existing Tailings Facility

The general arrangement plan of the existing Tailings Impoundment Area (“TIA”) is shown in Figure 18.1.

Fine tailings at the Scully Mine are thickened to a slurry form in an 85 m diameter thickener. Coarse tailings are cycloned and delivered directly to the tailings pumpbox. The density of the slurry to be pumped is approximately 35 to 40% solids by weight initially. Three 305 mm (12-inch) diameter rubber-lined pipelines are utilized in the disposal system to transport the tailings from the concentrator to the TIA. The slurry is pumped at approximately 3.7 m/s (12 ft per second) or 15.1 m^3/minute (4,000 gallons per minute) per line. Two of the three lines must be operational for normal processing plant operations. At present, a series of pumping stages located in the concentrator pump bay and eight pump houses on each pipeline maintain the required volume capacity and velocity over the 7 km distance from the plant to the tailings basin. Two of these pump houses are situated on the dike crest. There are 10 pumps available for each of Tailings Lines 2 & 3 and eight pumps for Line 1.

Figure 18.1: Tailings Impoundment Area – Existing Arrangement



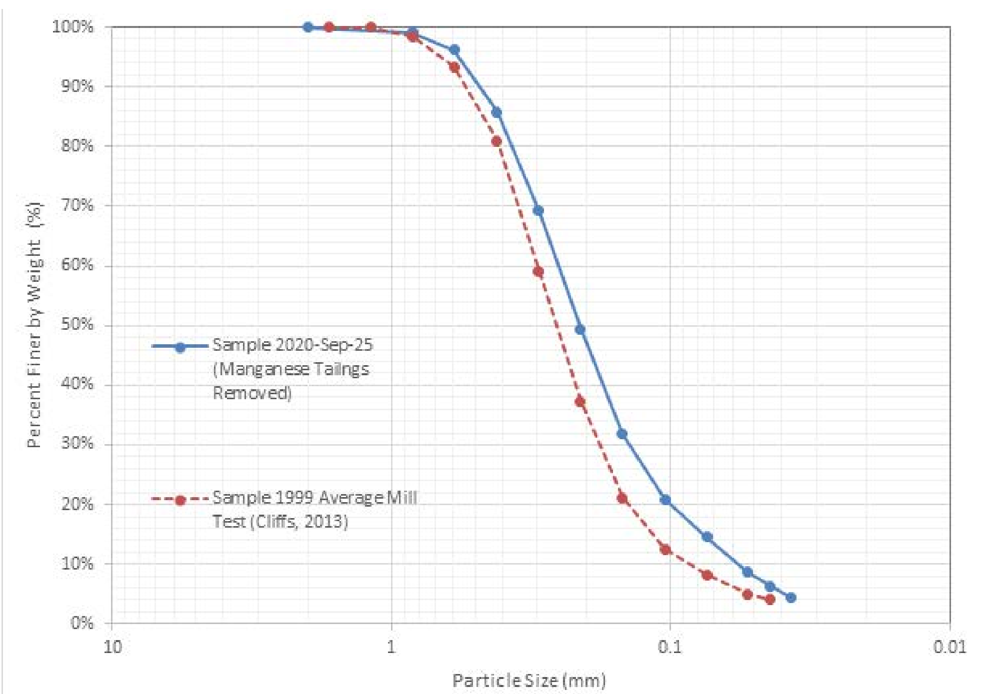
18.4.2 Tailings Characteristics

Tacora iron tailings are mostly sand-sized particles with fines less than 15% by weight passing a 75-micron screen. Figure 18.2 illustrates the particle size distributions of the tailings samples produced during the historical operation (sampled in 1999) and produced during restart operation (sampled in 2020). The following observations were noted:

- Tailings representative sample of the recent restart operation appear to be slightly finer than the historical operation. This could be due to the addition of new manganese removal circuit considering that the grinding equipment and ore minerology have not changed since the historical operation. The variation is not significant, and it is not expected to have a notable change in the characteristics of the tailings.
- The extracted manganese concentrate from the manganese removal circuit is presently stockpiled near the plant area. However, the separated manganese concentrate, if not marketable, is planned to be put back into the main tailings stream and sent to the TIA for disposal.

The specific gravity of the tailings solids could range from 2.6 to 5.0 and is typically 3.15, based on the Cliffs Report (2013).

Figure 18.2: Particle Size Distributions of Scully Mine Tailings



The thickener typically produces pulps of 35 to 40% solids by weight, expected to be increased gradually to the target density of 48% solids by weight. The increased solids content of the slurry will result in changes in the tailings beach slopes from those observed during the past operations. The tailings beach slope should be regularly surveyed, and adjustments made to the deposition planning and future estimated storage capacity.

As a contingency for potential variance in storage, a conservative deposit tailings dry density of 1.6 t/m³ was adopted for the storage estimates. Deep, consolidated tailings are likely to have a higher bulk density, up to 1.9 t/m³, locally, resulting in lower storage volume requirements and costs for dike construction, thus prolonging the serviceable life of the existing and future storage capacity.

In planning, forecasting the size and geometry of the deposited tailings is required. Two important items in forecasting the tailings storage are the particle size distribution of the tailings, slurry density (percent solids in the slurry by weight) and discharge velocity. These parameters contribute to the formation of the tailings deposition slope.

The discharged tailings slurry forms a naturally segregated delta in the basin. The coarse fraction of tailings deposit near the discharge point and finer fractions are hydraulically transported and deposited further out into the basin. The denser and slower the slurry being pumped, the steeper the deposit slope. The coarse materials settle out first to form the steeper slope and the remaining fine fraction forms a flatter slope.

The tailings are considered low risk tailings from a potential for acid generation and metals leaching; therefore, seepage containment and effluent treatment system will not be required for tailings storage. More details are available in Subsection 20.2.7 “Tailings and Waste Rock”. As no flotation stream is involved in the process of the tailings, the material pumped to the TIA is relatively coarse, allowing for good drainage and use as construction material for future dike embankments.

18.4.3 Design Basis and Dam Consequence Classification

The design and operation of the TIA is in accordance with the CDA Dam Safety Guidelines (CDA, 2013) and the associated Application of Dam Safety Guidelines to Mining Dams Technical Bulletin (CDA, 2019).

A range has been assigned to describe both the Population at Risk (“none to temporary”) and Incremental Loss of Life (‘nil to unspecified”) as there are no permanent residential or commercial buildings downstream of the TIA and only possibly mine staff, and the dam (i.e. dike) consequence corresponding classification has been assessed as “Significant”. Classification was assessed as being the same for both flood and fair-weather conditions because there is no pond retained by the dikes, and the deposited tailings are considered to have low liquefaction potential under a design earthquake event.

The Inflow Design Flood ("IDF") for 'Significant' classification would be 1/3 between 1/1,000 and probably maximum flood ("PMF"). Such an event would result in a minor water level rise within the TIA relative to the downstream environment (there is no impounding cross-valley dam structure). Further, the minimum crest elevation (approximately El. 558 m at the North Dike) is approximately 18 m higher than the existing water levels in North and South Flora Lakes. For these reasons, the freeboard and top-of-core requirements are considered moot for the TIA dikes.

The Earthquake Design Ground Motion ("EDGM") should be the 1:2,475-year event. The Peak Ground Acceleration ("PGA") for the 1:2,475-year event is 0.039 g.

18.4.4 Tailings Storage Requirement and Deposition Plan

The Concentrator will produce approximately 320 Mt (200 Mm^3 at assumed dry density of 1.6 t/m^3) of tailings over a period of 27 years based on a production reference point at the beginning of 2021 (Year 2021 to end of Year 2047). The required tailings storage capacity for the life of mine on an annual basis was obtained from G Mining in April 2021. On a yearly average, 18 Mt of ore will be processed to produce 6 Mt of iron concentrate with 12 Mt of tailings left to be disposed in a TIA.

The tailings are transported from the plant to the TIA via three separate steel pipelines (two in operation and one spare at any given time) and intermediate pump booster stations (up to Pumphouse #8). Presently, the tailings are pumped end discharged from the dike crest at the southern extent of the TIA.

The pipeline end discharge arrangement will be modified to allow multiple spigot discharges and spigot discharge locations will be relocated regularly to allow drainage of the coarse material near the discharge point and to avoid thick layers of frozen tailings which may not fully thaw during future construction periods. Spigot discharge methods will consider winter operations to target longer term deposition areas and limit frequent spigot valve operations over the cold weather season.

Figure 18.3, to Figure 18.11 show the proposed conceptual phased progression of the tailings deposition plan for the TIA. A summary of the tailings deposition phases is presented in Table 18.1. The description of the deposition plan phases is provided in the following:

Phases 1 to 3 (Figure 18.3, Figure 18.4 and Figure 18.5) will fill in the available space of the existing tailings area and the existing storage is estimated to be sufficient without impacting North Flora Lake or the

El. 555 m level of South Flora Lake and impedance of the natural drainage path (i.e. Diversion Channel). The existing storage capacity, with the current dikes, is sufficient for at least 3 years of storage. Capital cost in the form of new equipment or dike raises is not required for this period, with the exception of slurry pipeline extension and implementation of a spigot discharge system as described in Subsection 17.3.7 “Tailings Management”. It is assumed that the current pumping capacity is sufficient to discharge at the planned locations up to Phase 3.

Phases 4 to 6 (Figure 18.6, Figure 18.7 and Figure 18.8) will require raising of the existing Saddle Dike. During Phase 5, it is expected that the deposit could impede the Diversion Channel and raise the South Flora Lake level above the El. 555 m limit of Schedule 2. An amendment to the Schedule 2 is underway to permit tailings expansion further south up the Flora valley.

Phase 7 (Figure 18.9) will envisage a significant extension of the tailings deposit south and it will be completed in two subphases with multiple dike raises over a 10-year period. South Flora Lake will be completely filled. The extension south will require development of a greenfield area with construction of the Starter Dikes with associated foundation stripping, underdrainage and tree clearing within the proposed extended basin area.

Phase 8 (Figure 18.10) will continue to raise within the area of the Phase 7 deposit to maximize the vertical storage capacity as well as maintain the natural drainage pathway of the Diversion Channel.

Phase 9 (Figure 18.11) will raise above the Phase 8 deposit area with additional 3 m dike raises as follows:

- Phases 9a to 9c – Dike raised to El. 614 m for deposition over previous Phases 7 and 8 areas.
- Phase 9d – Dike raised to El. 617 m for deposition over previous Phase 9b area. Prior to deposition, a deflector berm (4 m high) will be constructed on tailings beach slope using construction method similar to cut-to-fill of tailings dike raises. The deflector berm will be required to minimize pinching of Diversion Channel by the raised tailings deposit which in turn avoids raising the large existing lake (south of the TIA) above its closure level of 560 m – 562 m.
- Phase 9e – Deposit east side of large knoll to fill in low ponded area. This requires slurry pipeline extension along constructed deflector berm; given lower discharge outlet elevation relative to last pumphouse, it was assumed no additional pumphouse would be required besides additional 1,700 m extension of main delivery pipelines. Furthermore, it was assumed a discharge bench would be constructed on the existing knoll crest at El. 580 m which would enable gravity flow of slurry pipeline discharge from the western dike crest. A formed pond on the east side of the knoll

will be reduced by the deposited tailings to a small narrow geometry to maintain flow path of the Diversion Channel.

Due to the expected increase in slurry thickness compared to previous operations and in order to validate design assumptions of beach slopes and average deposit density, the tailings beach should be surveyed every 6 months to 1 year to update these projections with observed tailings beach angles.

Fugitive dust emissions and control will be managed by an annual hydroseeding campaign over exposed tailings beach surfaces.

Table 18.1: Summary of the Tailings Deposition Phases

Phase	Figure	Description	Incremental Deposit Volume m^3	Cumulative Volume m^3	Average Monthly Production m^3/Month	Duration Months	Start	Finish	Dike Crest Elevation m	Discharge Elevation m	Predicted Lake Level Elevation m	MMER Schedule 2 Limit Exceedance
Phase 1	Figure 18.3	South Flora Lake	2,472,169	2,472,169	258,799	9.6	1-Nov-20	19-Aug-21	585	584.5	550	No; within current limits.
Phase 2	Figure 18.4	South Dike Low Areas	3,917,118	6,389,287	258,799	15.1	1-Nov-20	5-Feb-22	585	584.5	550	
Phase 3	Figure 18.5	Saddle Dike Crest El. 583 (Starter)	2,494,227	8,883,514	414,079	6.0	19-Aug-21	19-Feb-22	583	583	550	
Phase 4	Figure 18.6	Saddle Dike Raise Crest El. 586	11,574,359	20,457,873	587,234	19.7	19-Feb-22	13-Oct-23	586	586	551	Possible; exceedance by tailings beach shoreline limits.
Phase 5	Figure 18.7	Saddle Dike Raise Crest El. 589	6,684,908	27,142,781	570,108	11.7	13-Oct-23	4-Oct-24	589	589	554	Yes; exceedance by South Flora Lake level 555 m and inundation of tributaries.
Phase 6	Figure 18.8	Saddle Dike Raise Crest El. 591	7,220,059	34,362,840	593,154	12.2	4-Oct-24	11-Oct-25	591	591	554	Yes; exceedance by both shoreline limits and lake level El. 555 m. Amendment approvals required to continue operation.
Phase 7	Figure 18.9	South Extension Dike Crest El. 594 (Starter)	67,934,766	102,297,606	606,286	112.1	11-Oct-25	18-Feb-35	594	594	560	
Phase 8	Figure 18.10	South Extension Dike Crest El. 607 (Starter)	60,266,205	162,563,811	557,307	108.1	18-Feb-35	29-Feb-44	607	607	560-565	
Phase 9	Figure 18.11	South Extension Dike Raise Crest El. 614	37,149,861	199,713,672	569,247	41.3	29-Feb-44	11-Aug-47	617	617	560-565	

Figure 18.3: Tailings Impoundment Area – Future Tailings Storage Phase 1 (Years 2020 to 2021)

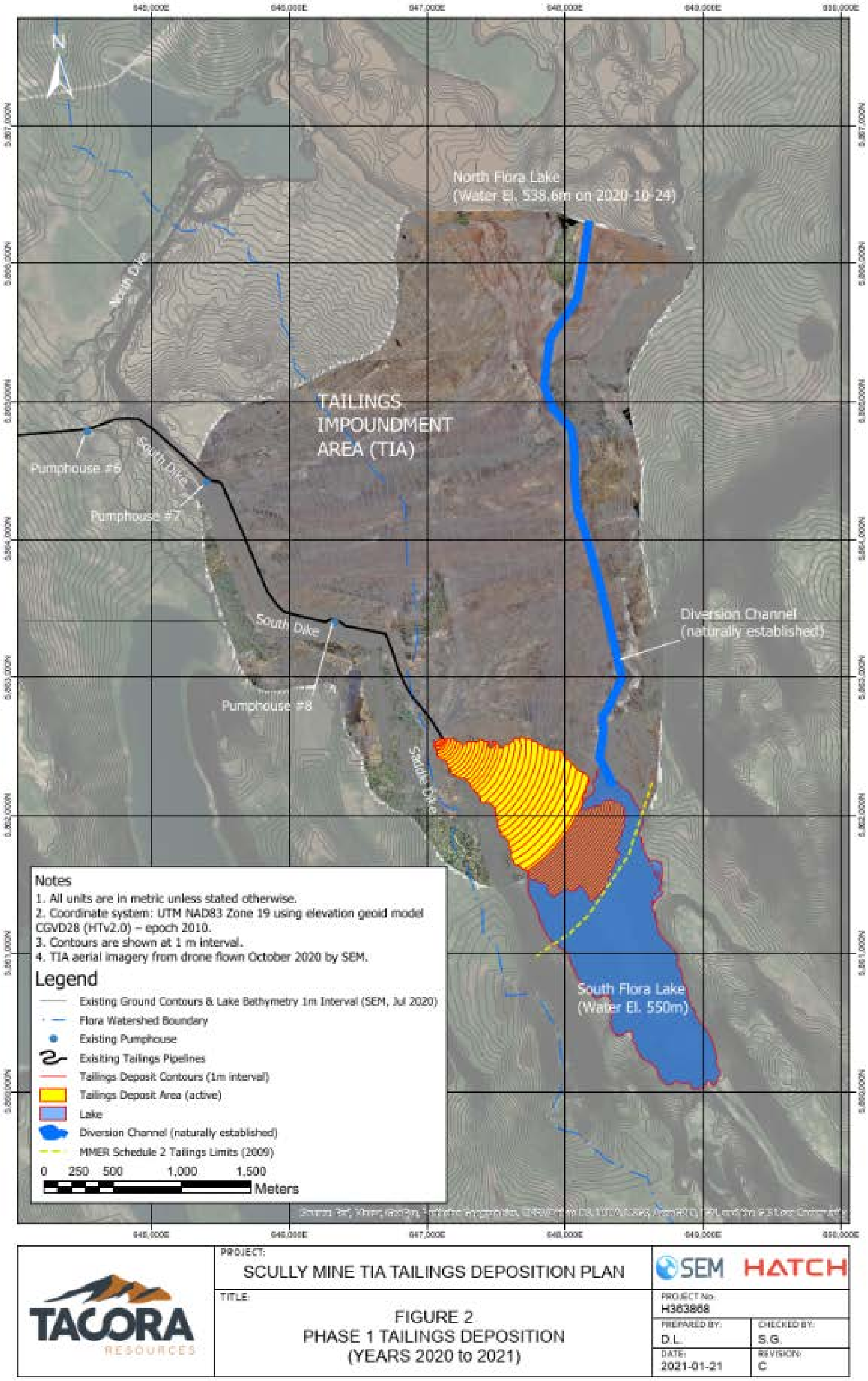


Figure 18.4: Tailings Impoundment Area – Future Tailings Storage Phase 2 (Years 2020 to 2022)

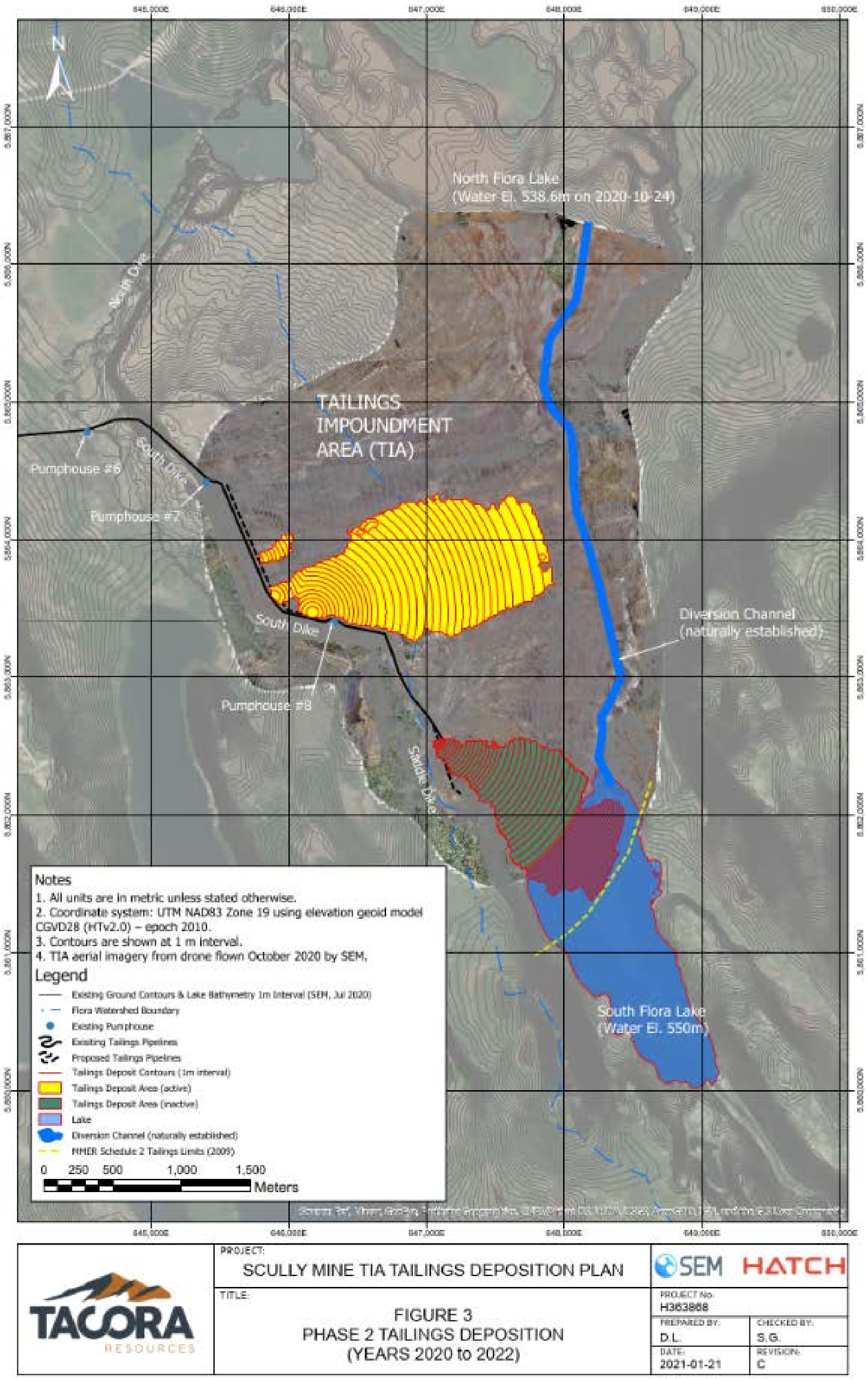


Figure 18.5: Tailings Impoundment Area – Future Tailings Storage Phase 3 (Years 2021 to 2022)

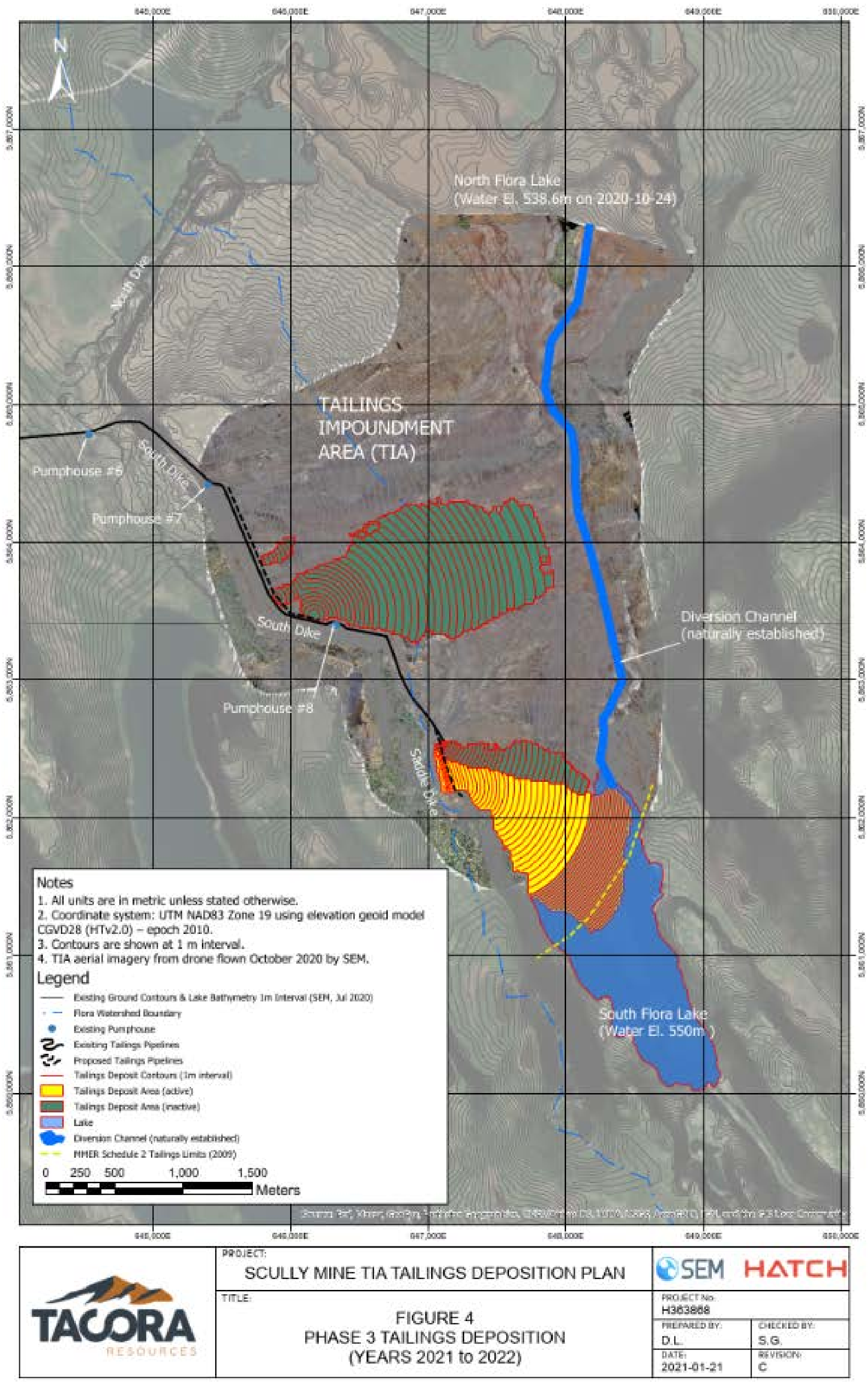


Figure 18.6: Tailings Impoundment Area – Future Tailings Storage Phase 4 (Years 2022 to 2023)

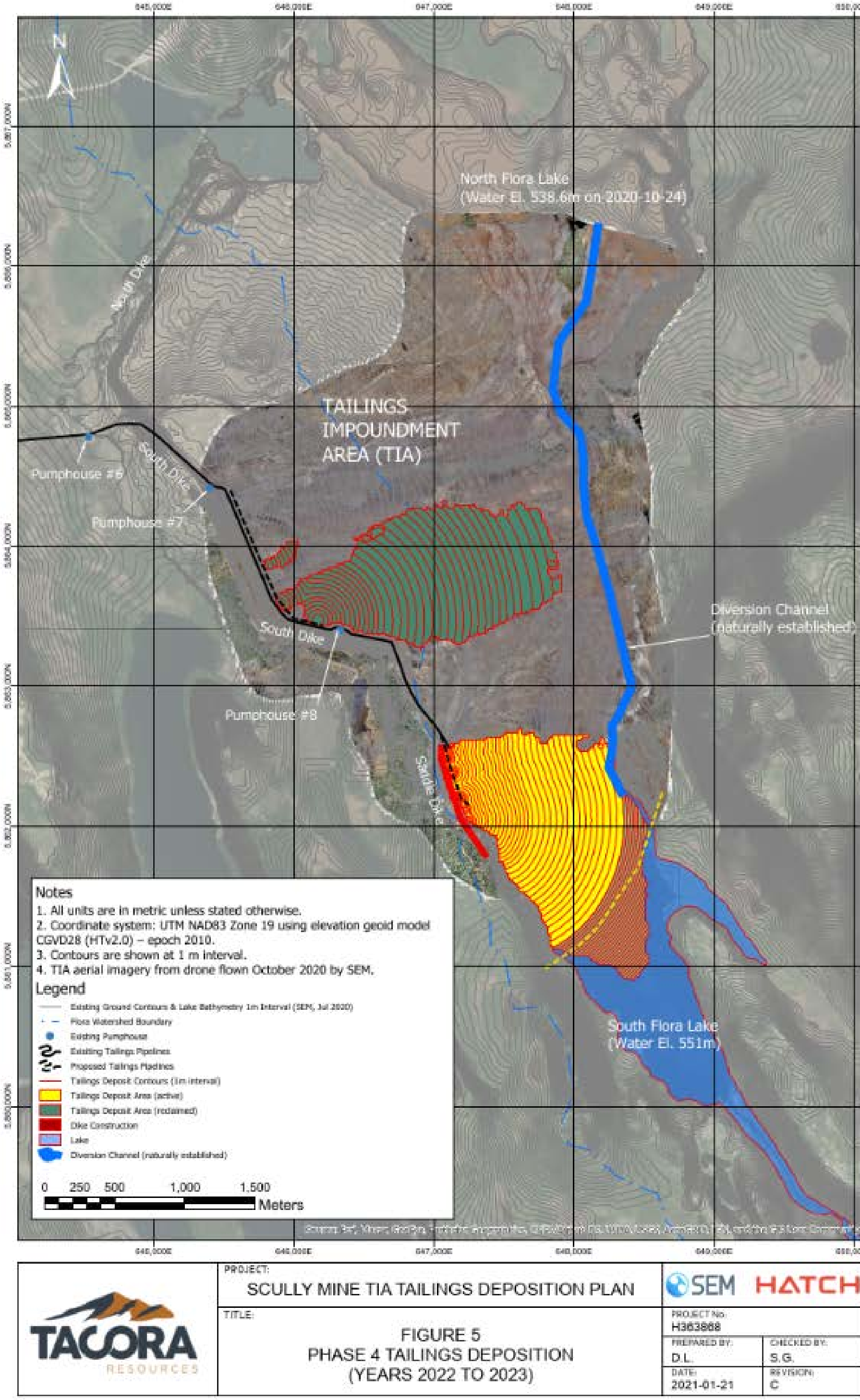


Figure 18.7: Tailings Impoundment Area – Future Storage Phase 5 (Years 2023 to 2024)

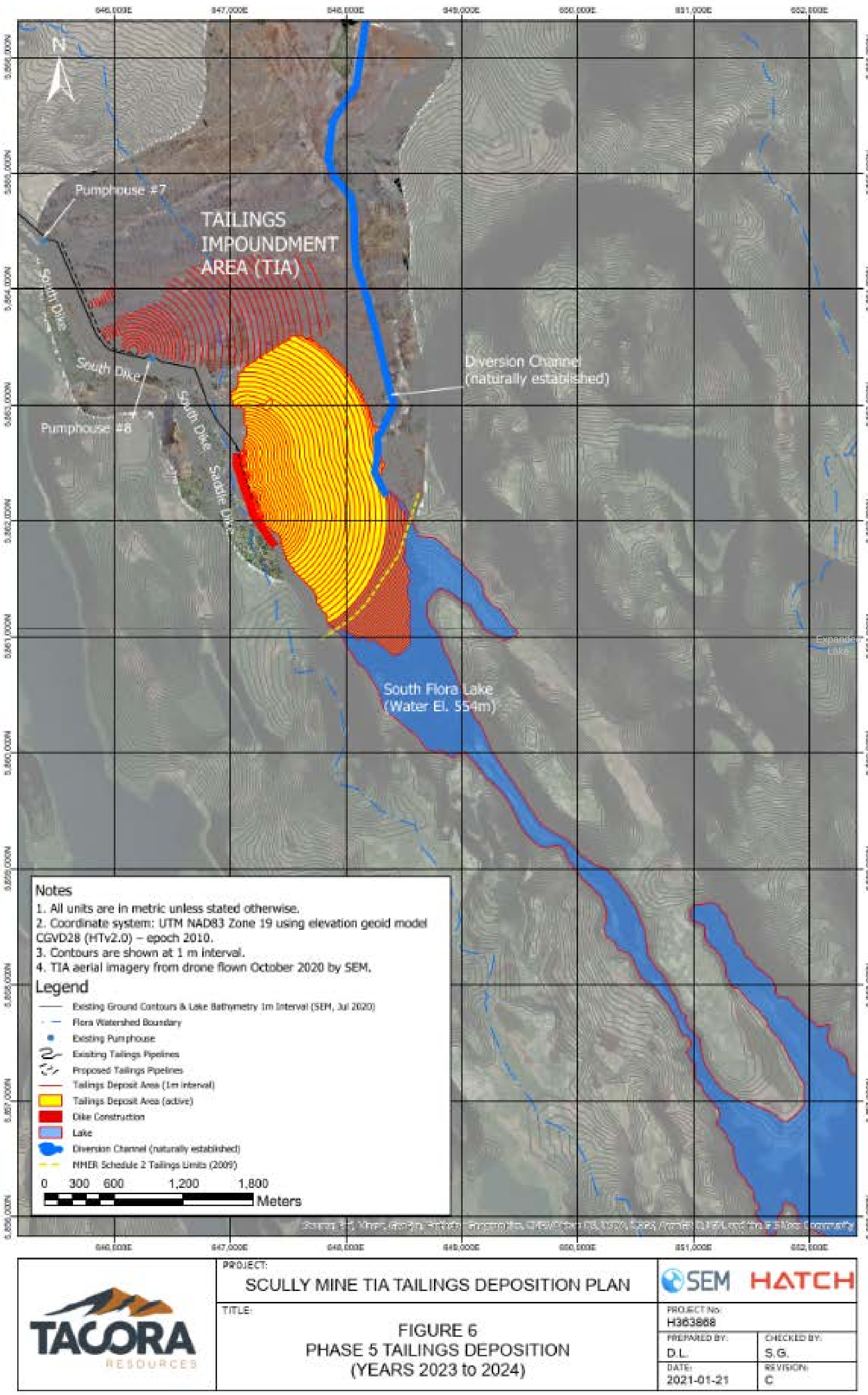


Figure 18.8: Tailings Impoundment Area – Future Tailings Storage Phase 6 (Years 2024 to 2025)

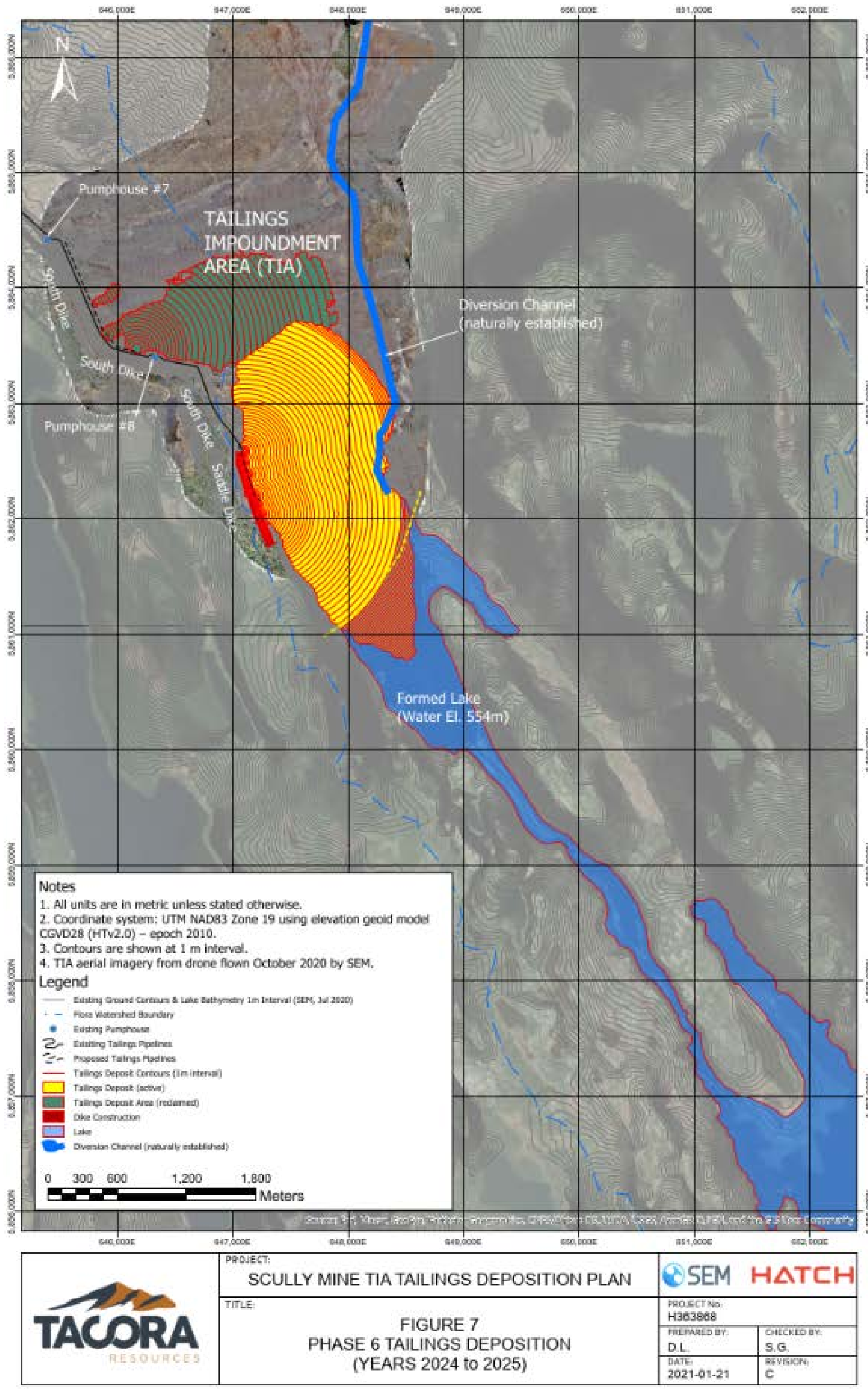


Figure 18.9: Tailings Impoundment Area – Future Tailings Storage Phase 7(Years 2025 to 2035)

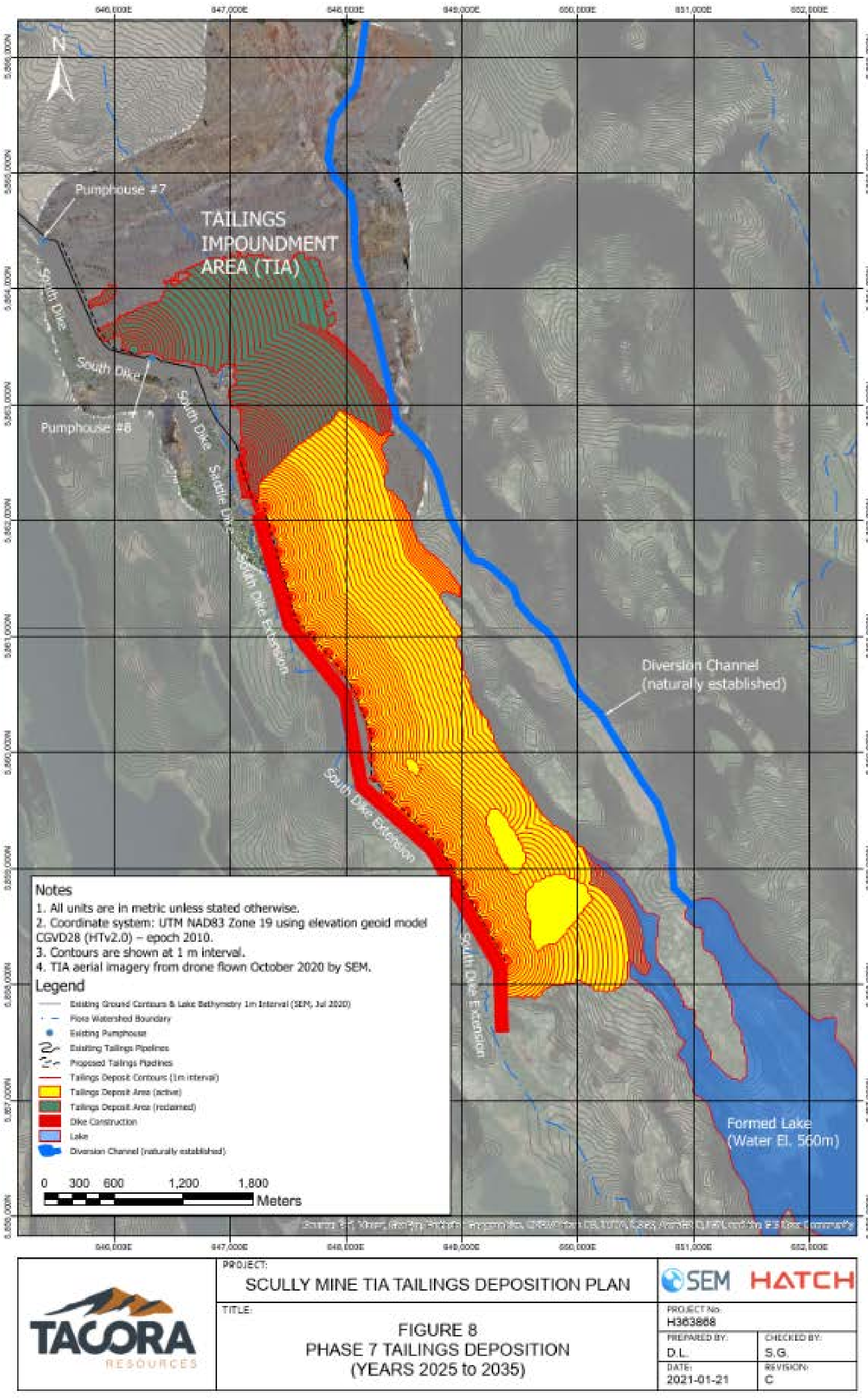


Figure 18.10: Tailings Impoundment Area – Future Tailings Storage Phase 8 (Years 2035 to 2044)

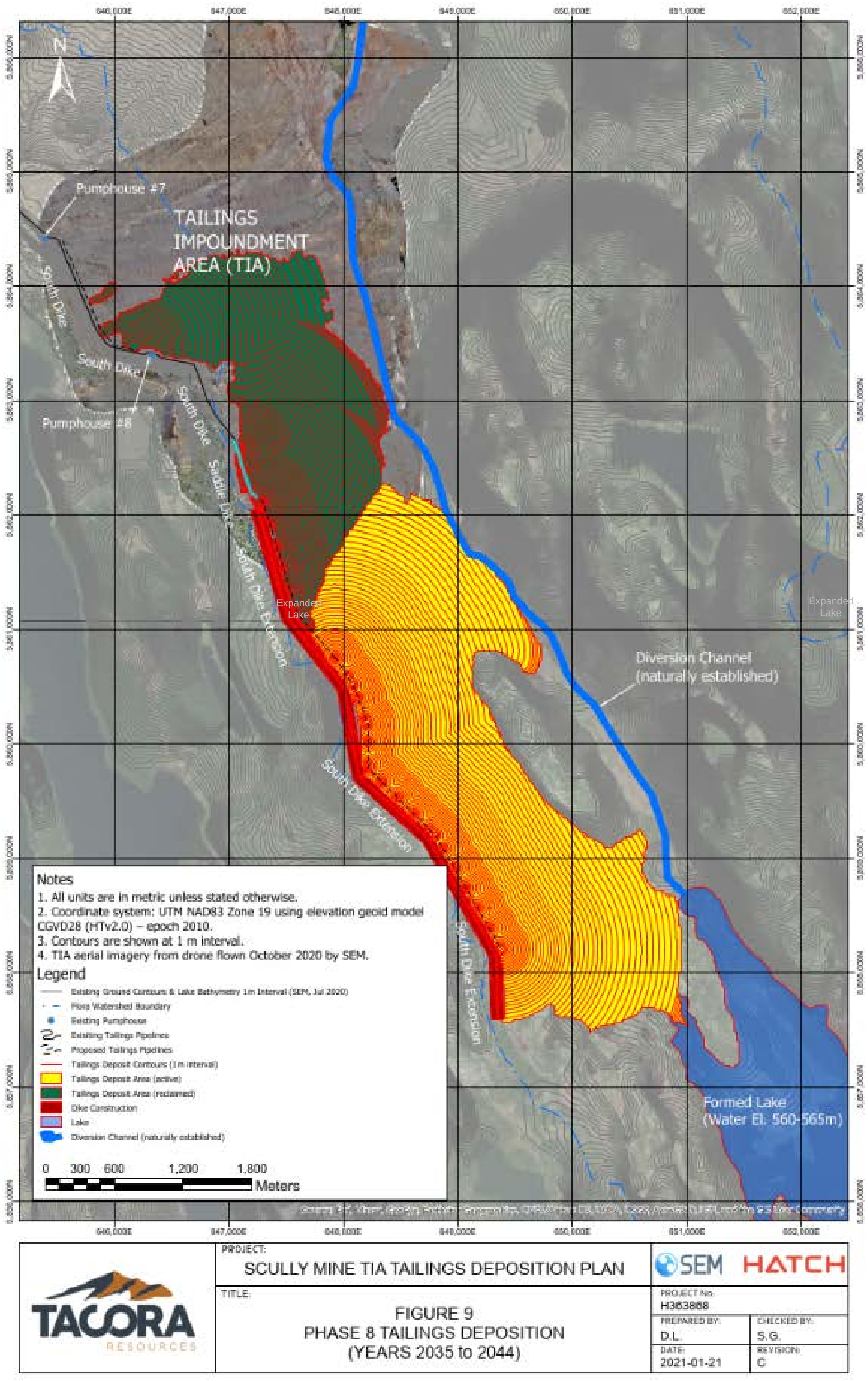
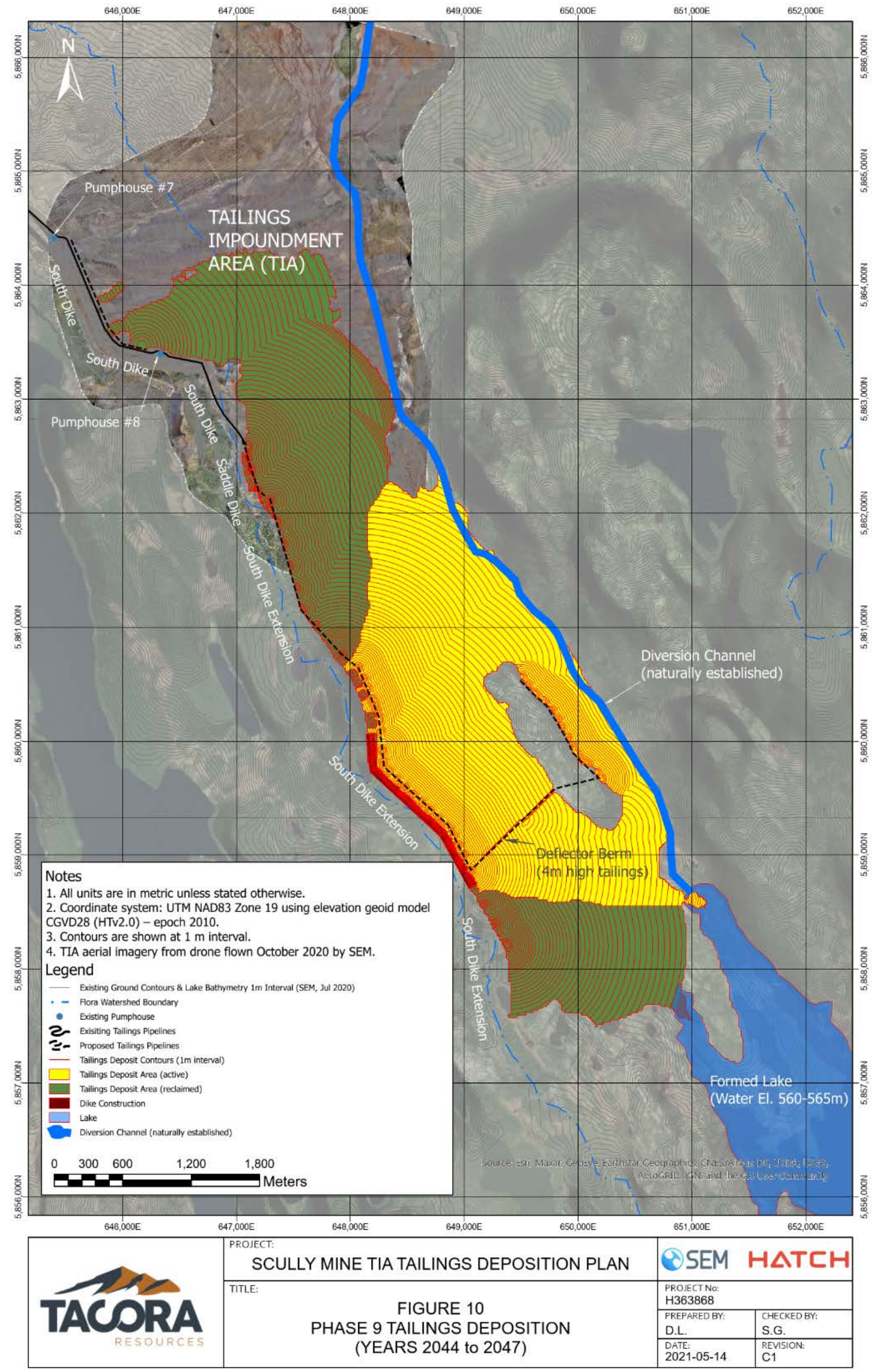


Figure 18.11: Tailings Impoundment Area – Future Tailings Storage Phase 9 (Years 2044 to 2047)



18.4.5 Dike Design and Construction Method

To facilitate the deposition plan and achieve vertical storage, dikes will be constructed along the natural topographic ridge (west side of the Flora watershed divide) and will be raised in the upstream direction over free draining, consolidated, coarse particle tailings deposits. This methodology follows the proven and stable dike structures constructed during the historical operation of the TIA (e.g., the existing North and South Dikes constructed at least 7 years ago). This methodology and conceptual design assumptions are contingent on geotechnical investigations, detailed engineering and permitting.

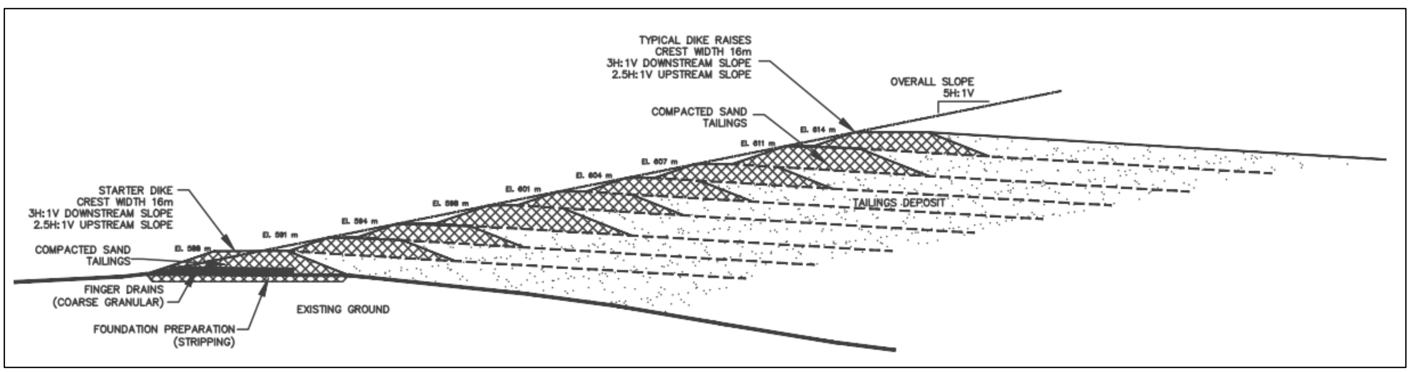
The conceptual dike section is summarized in Table 18.2 and illustrated on Figure 18.12.

Due to the proximity of the Wabush Airport, there is an imposed height restriction zone on the northern portion of TIA with maximum elevation of 590.5 m for constructed structures such as dikes. This zone restriction impacts the majority of the existing TIA (North and South Dikes are in compliance). Future TIA expansion will be outside of this restriction zone.

Table 18.2: Design Summary of Typical Dike Section

Design Parameter	Value	Basis of Assumption
Overall Slope	5H:1V	Historical design for North and South Dike
Bench Slope	3H:1V Downstream	Reduce surficial erosion/gulling and facilitates placement of vegetation for closure reclamation
Bench Height	Maximum 4 m (typically 3 m)	Rate of raise
Bench Width	Minimum 6 m	Satisfies overall slope requirement and sufficient to allow single articulated haul truck
Crest Width	16 m	Allowance for construction vehicle traffic, safety berms and 3 tailings slurry pipelines

Figure 18.12: Typical Dike Design Section for Southern Extension



There are no available geotechnical data and foundation characterization in the new southern extension areas of the TIA beyond the existing Saddle Dike. Future geotechnical investigation programs and detailed engineering studies are planned in advance of construction requirements. To evaluate seepage and stability for a conceptual level of dike design as presented in this present Study, assumptions were adopted from available geotechnical foundation data in the historic North and South Dike areas, as well as limited data on the tailings characteristics.

Seepage analyses were carried out to evaluate the anticipated phreatic level within the dike section. As the South Flora Lake level would be significantly lower than the foundation base of the dike, the source of water within the dike section could only be derived from precipitation infiltration and free water released from the slurry discharge operation. Based on historical and present operations, the phreatic levels within the dike section can be observed from installed piezometer instruments to be elevated but quickly drained once the active slurry discharge has ceased or has moved away sufficiently. The Starter Dike design section will incorporate finger drains at the base to improve drainability and lower the phreatic level, particularly in the toe section, thereby improving the local stability of the dike embankment.

Stability analyses were carried out on the typical design dike section for the critical section with the maximum dike height of the southern extension. Given that there are no geotechnical foundation data for the new dike extension area of the expanded TIA, a conservative foundation model similar to the South Dike was adopted for the analyses. The minimum safety factors were satisfied for all relevant loading conditions (short-term, long-term, pseudo-static and post-seismic) in accordance the guidelines of the Mining Dams Technical Bulletin (CDA, 2019). The design section with an overall slope of 5H:1V follows similar overall slope of the existing North and South Dikes which have remained stable without signs of instability since their construction at least 7 years ago.

Starter Dikes will be constructed for the new southern extensions beyond the existing Saddle Dike beginning in Phase 7. The foundation preparation of the dike including the stripped material will be stockpiled outside of the deposition basin for future reclamation or used for additional organic matter as part of ongoing progressive rehabilitation. Clearing and chipping of trees within the basin area will be required to minimize impedances of the deposition flows and beach formation; however, stripping or grubbing is not anticipated. The slurry deposition will naturally segregate the coarser particles near the spigot discharges from dike crest locations with the finer particles being washed / carried further down the beach. The coarser grained tailings beach deposits in the inactive area offers convenient access for dike construction materials with suitable permeability, strength, and drainage properties; it also provides competent foundation to construct the raises.

Finger drains that are to be constructed at the base of the Starter Dike section, may improve drainage and help reduce potential for seepage exiting the higher elevation on the slope face of the new dikes during active deposition in the vicinity. The finger drains would consist of coarse granular mine-produced material with mostly gravel sizes. It is recommended to review and optimize this drainage design in detailed engineering.

Embankment construction should be carried out in non-freezing months whenever possible to avoid construction over frozen tailings with high ice content. Due to the short construction period, raises should not exceed more than 4 m per year to allow drainage and pore pressure dissipation. Planning of the Starter Dike and Dike Raise constructions should be completed a season prior to deposition requirement and the schedule should account for early new geotechnical investigation campaigns and detailed engineering studies.

18.4.6 Water Management at the Tailings Impoundment Area

According to the Schedule 2 permit, tailings are not permitted to be deposited directly into the North Flora Lake which will serve primarily as a polishing system for the inflows from upstream active tailings deposition and runoff from the upper Flora valley watershed. The Lake will help prevent suspended sediments from being carried downstream towards Wabush Lake.

Considering the side valley discharge approach of the TIA, there are effectively no cross-valley impounding dam structures, no decant pond formed and no spillway structure constructed for the operation of the TIA. Hence, the water management strategy considers the following requirements:

1. Tailings deposition must be controlled to ensure the beach development does not impede the drainage pathway (i.e., Diversion Channel) for the upper Flora Lake watershed. Where topographic high points (knobs) exist, a channel construction through or around the knob will be necessary to maintain conveyance of the valley drainage pathway as the tailings deposit obstructs the original natural valley pathway.
2. Maintain flow capacity of existing Flora Outlet Arm Culverts (Regulatory final discharge point of the TIA) at the Railway and Highway 500 crossings. It is also the outlet of the whole Flora Lake watershed including the TIA operation. The water quality is monitored at this final discharge point.

There is no water reclaim pumping system at the TIA given there is no requirement for process water re-use. The natural runoff of the Flora Lake watershed including the TIA operation are routed via the Diversion Channel (drainage pathway).

18.4.7 Instrumentation and Monitoring

To address any risk related to the stability of the dike embankments, the following practices are established in the Operation, Maintenance and Surveillance ("OMS") Manual:

- Instrumentation installed for monitoring of phreatic levels at the existing dike and future instruments to be installed in critical sections of the new dike extension. Instruments include standpipe piezometers which will also facilitate groundwater sampling for environmental water quality monitoring.
- Instrumentation installed for monitoring the level of South Flora Lake to comply with the existing Schedule 2 permit. This monitoring can be discontinued once Schedule 2 Amendment approval is granted.
- Implement dike safety practices for surveillance and monitoring of the dike in accordance with Canadian Dam Association Mining Dams Technical Bulletin (CDA, 2019) and Mining Association of Canada Operation, Maintenance, and Surveillance Manual (MAC, 2019) guidelines for dam safety and tailings management system. These include routine inspection by trained site staff, frequent dam safety inspections by the Engineer of Record and periodic dam safety review audits by qualified third-part consultant.
- Drone survey of the active beach slopes and the drainage pathway of the Diversion Channel to ensure deposition is in accordance with the deposition plan and ensure no impedance or blockage of the Diversion Channel drainage path.

18.4.8 Closure Considerations

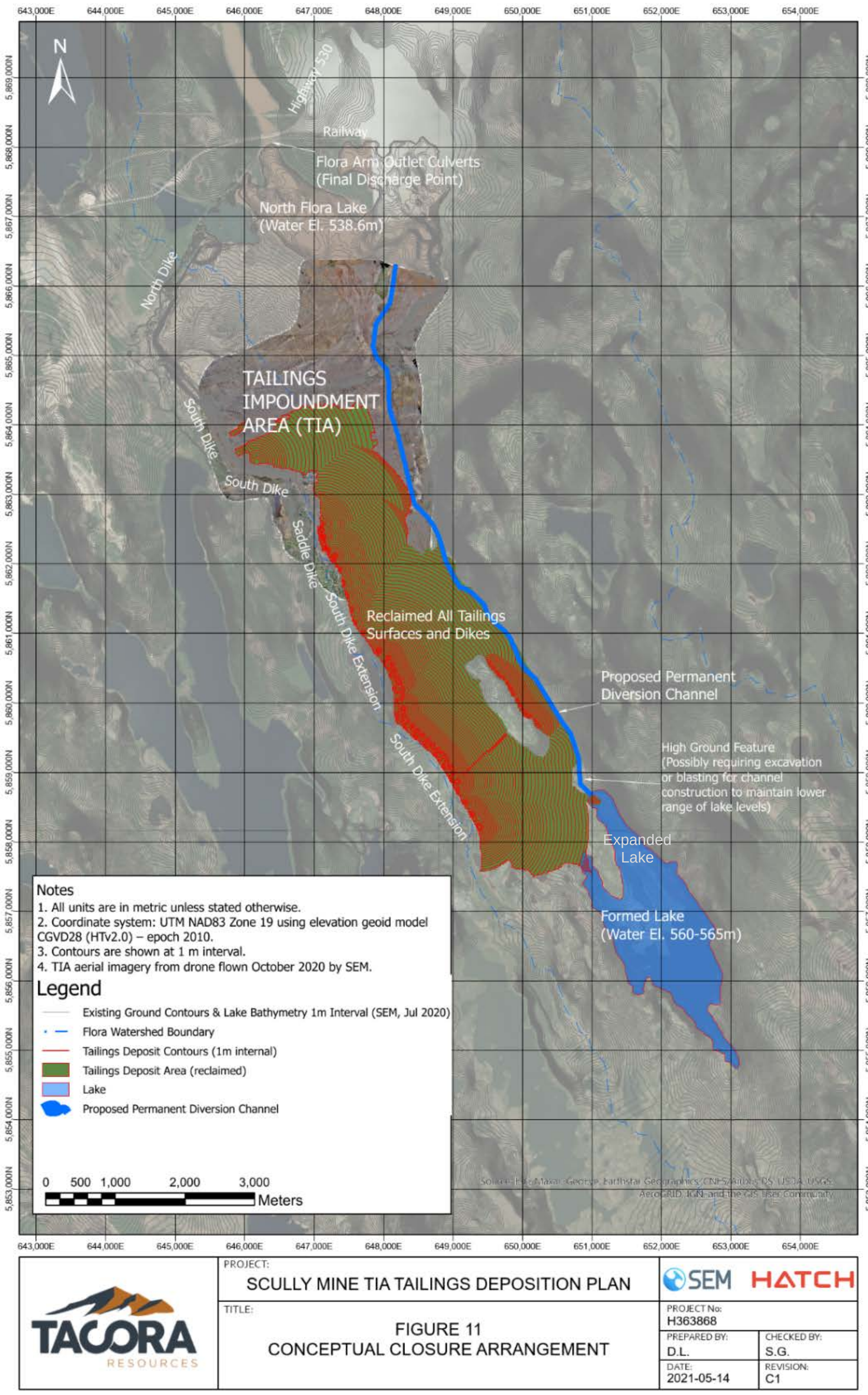
The closure strategy of the TIA incorporates progressive reclamation of inactive areas where tailings deposition has been completed; Figure 18.13 illustrates the final closure configuration of the TIA at end of operation. The closure activities include the following:

- Rehabilitation of all exposed inactive tailings surfaces and disturbed areas with vegetation.
- Regrading of dike bench slopes where required mostly on the over-steepened bench slopes of the existing South and North Dikes.
- Construction of the engineered Diversion Channel with appropriate riprap erosion protection and conveyance capacity.
- Construction of the surficial drainage ditches along the formed lows/creases of the tailings beach surface where concentration of runoff is anticipated.

Progressive reclamation of inactive areas occurs from the northern sector of the tailings beach as the active tailings deposition develops southward and along the Flora Lake valley. This early reclamation will minimize fugitive dust emissions and formation of erosion gullies which could washout tailings solids and potentially carry into North Flora Lake.

Progressive reclamation and revegetation of all exposed tailings beaches and disturbed areas will involve hydroseeding with select native species seed mix, spread of fertilizers, minor regrading of dike embankments and construction of ditches for management of surface drainage. The learnings from the successfully completed closure reclamation plots on the north rehabilitation area of tailings beach will be adopted to implement a similar revegetation program for the future tailings beaches. No allowance has been made to borrow and apply organic topsoil as an element of the vegetation cover.

Figure 18.13: Tailings Impoundment Area – Conceptual Closure Arrangement Phase 10



18.5 Geotechnical

No additional geotechnical study is required except for elements stated in Subsection 18.4.

18.6 Electrical Installations

18.6.1 46 kV Power Distribution

Scully Mine is fed from Wabush Terminal Station by two power lines at 46 kV. Each line can individually provide full load for the mine operation.

The two power lines feed the main outdoor Switchyard that distributes the power at 46 kV to the Scully Mine, Tailings, Pumphouse and the Mill Outdoor Switchyard.

The Mill Transformer Switchyard is equipped with an aerial busbar supported by a metal structure and insulators. There are six main power transformers connected to the busbar via 46 kV breakers. Transformers T1 & T2 are 46/4.16 kV-10/13.3/16.6 MVA and provide power to a double busbar, 3,000 A, 4.16 kV switchgear. Transformer T4 is 46/4.16 kV - 20 MVA and feeds directly Electric Boiler No.1. Transformers T5 & T6 are 46/13.8 kV - 15/20/25 MVA and feed a double busbar, 1,200 A switchgear that provides power to all low voltage loads (600 V and below).

18.6.2 4.16 kV & 600 V Power Distribution

At mine re-start, most of the electrical equipment was reused. All major cables, transformers, MCCs and switchgear had been inspected and tested before re-energization. All electrical rooms were cleaned and painted.

The existing emergency generators were refurbished and are now operational. Old Variable Speed Drives installed in the plant were Rockwell 1336 model which is no longer supported by the manufacturer. Most of these drives were replaced during re-start of the site and the remaining ones are still functioning, but will be replaced when they fail.

The 5 kV MCC of the main switch-room was the cause of multiple downtimes in recent years of operation and needs to be replaced by a new Allen-Bradley Centerline. This upgrade of $1.15 M includes the synchronous motor control, protection, and exciter of the mill motors, and will be done in 2021.

At the steam plant electrical room, MCC and Switchgear will be replaced and automated in 2021 at an evaluated cost of $615,000 including the 70 starters and feeders of the area.

18.7 Instrumentation and Controls

18.7.1 Instrumentation

Most of the existing instruments were reused at start-up. Some cleaning, calibration, and testing was required on some of the instruments.

All the twenty-one densimeter nuclear sources were replaced with new sources of the same type prior to the restart of the process plant.

In this new phase, several instruments will be replaced in 2021 with new ones across the site at a cost of $851,000 to improve the accuracy and functionality of the control system.

18.7.2 Controls

Most of the old PLC system (model Rockwell PLC 5) was replaced with a new ControlLogix system. One area which is still to be migrated to the new PLC platform is the Steam Plant. All field devices and motor control centers will then be automated.

A major upgrade of the control room equipment was performed, including new work-stations, server, historian and firewall prior to the restart of the process plant.

19 MARKET STUDY

AME Consulting Ltd., ("AME") a firm specialized in the study of metals markets, was retained by Tacora Resources Inc. ("Tacora") to provide information on the strongly related steel and iron ore markets. Most of the information in this section was provided by AME, but some complements were also provided by other specialists and analysts.

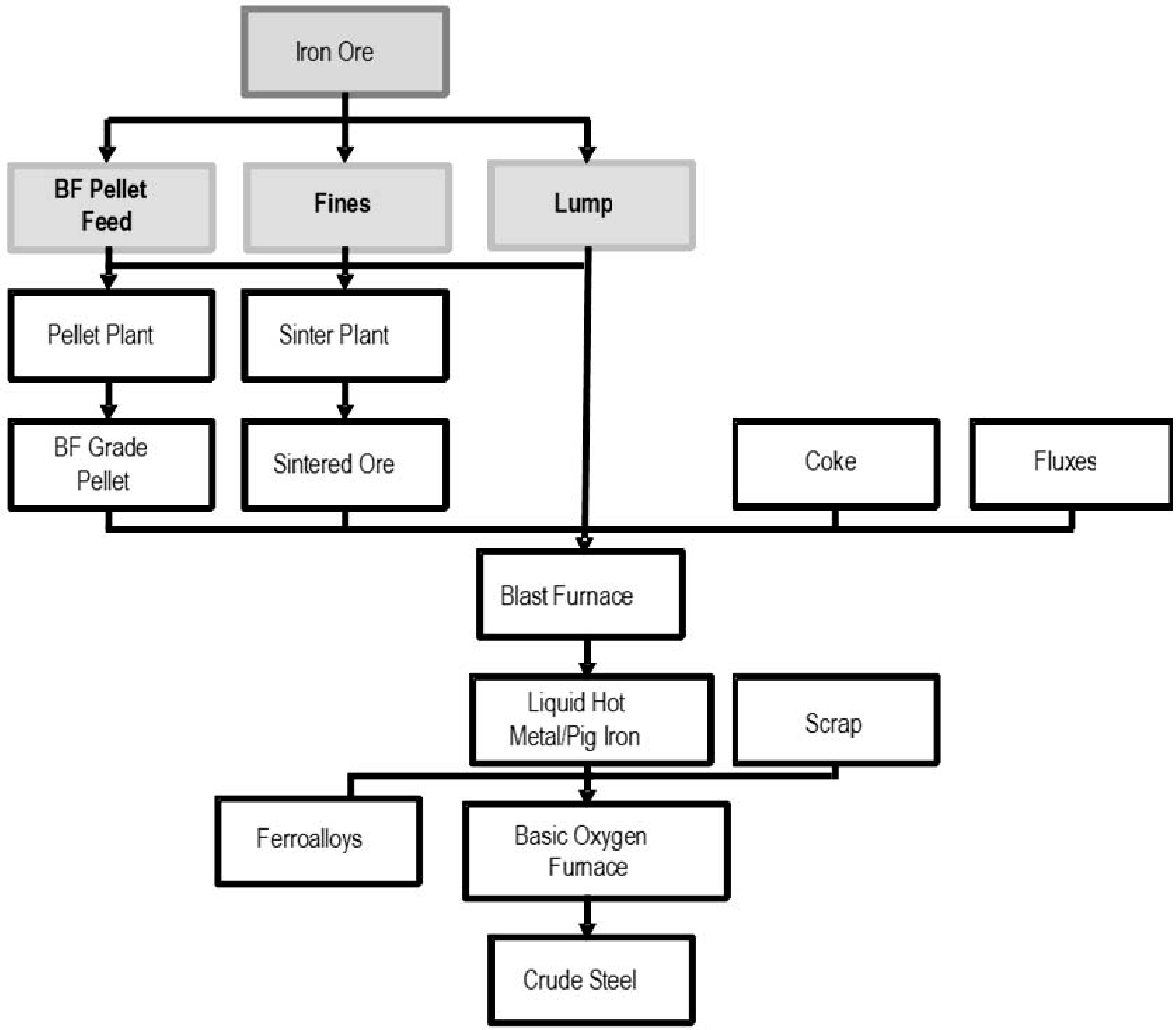
19.1 Product Use

Iron ore is primarily used in the steel industry and is one of the key raw materials in the iron making process along with coke and limestone in a blast furnace ("BF"), and natural gas or coal in a direct reduction furnace.

The blast furnace is the most commonly used iron making process. Blast furnace feedstocks include DSO lump and sinter, as well as blast furnace grade pellets agglomerated from pellet feed. In China, blast furnaces continue to be the dominant steel-producing method. In a blast furnace, iron ore is converted into primary iron (hot metal/pig iron) which is around 94% iron, and subsequently becomes feed material for the Basic Oxygen Furnaces ("BOF") in the production of steel. Pig iron can also be used in Electric Arc Furnaces ("EAF"). The BF / BOF production flow chart is presented in Figure 19.1

Direct reduction is a process used to make solid or molten iron products by using natural gas or coal as a reductant. The most common feed material for the direct reduction process is pellets produced from high grade magnetite concentrate. Direct reduction plants are generally operated in the Middle East as the region has access to inexpensive and abundant supplies of natural gas and energy. Indian coking coal quality is generally poor, but availability of thermal coals means some producers rely more heavily on the direct reduction process. There are typically three products produced in a direct reduction plant: direct-reduced iron ("DRI"), hot briquetted iron ("HBI") and hot DRI. DRI and HBI products, as well as scrap, are generally used as feed material in EAFs to produce molten steel. The DR / EAF production flow chart is presented in Figure 19.2.

Figure 19.1: BOF Steel Production Flow Chart



Source: AME

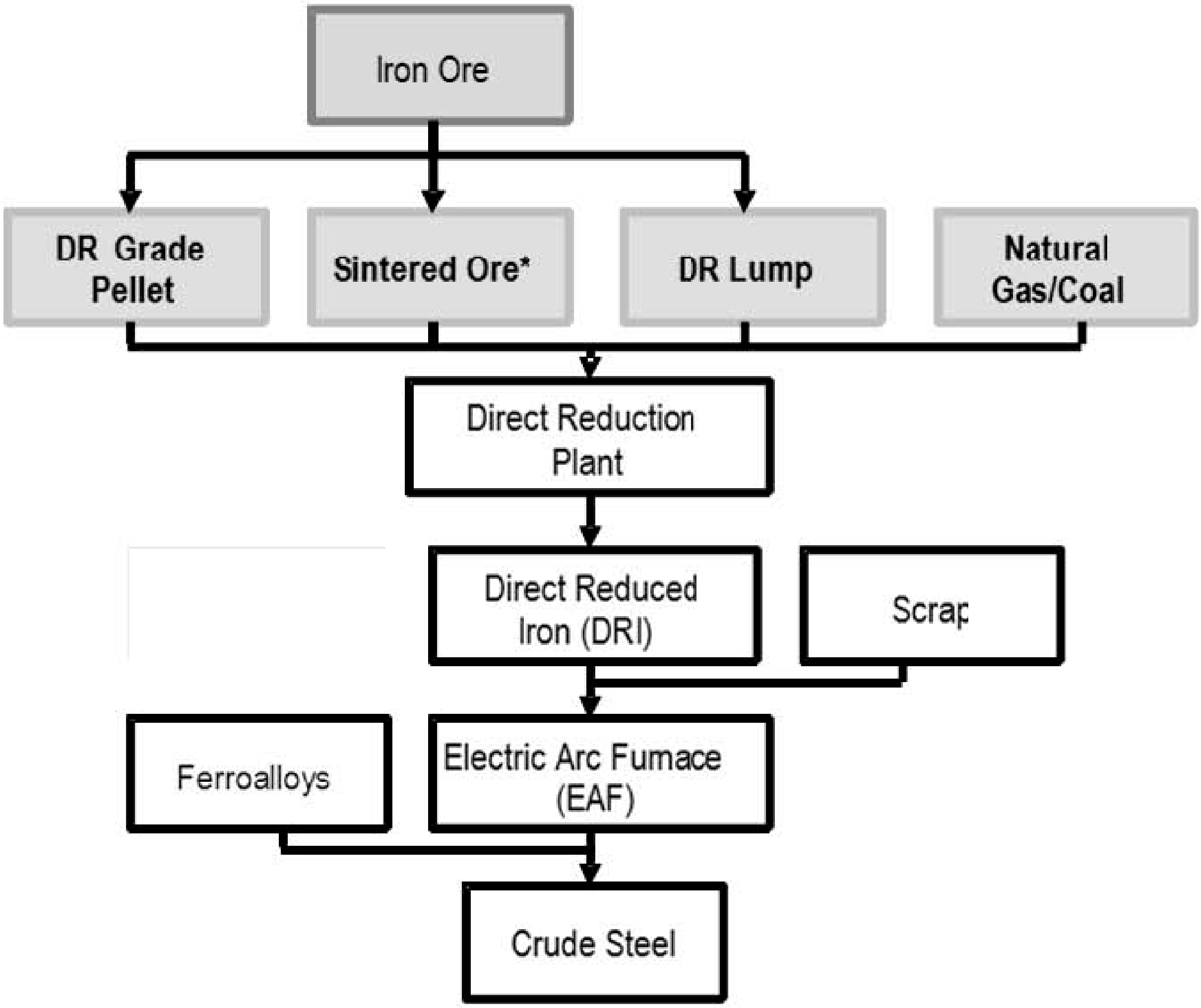
Steelmaking is the second step after iron making and involves the refining of the products from the iron making stage into liquid steel. This process can be accomplished in a BOF or an EAF. The BOF process injects gaseous oxygen into the furnace as the primary agent for auto-thermic generation of heat. This results in the oxidation of dissolved elements like carbon, silicon, manganese, and phosphorus — and to a limited extent, the oxidation of the iron. BOF steelmaking is the most widely used production method, accounting for around 74% of crude steel production in 2017. To produce steel through BOF, the proportion of pig iron as part of the feed is typically over 70%, with scrap accounting for the remainder of the feed.

EAF, on the other hand, is generally included in the basic design of a typical mini-mill plant for melting scrap or for taking sponge iron from a DRI plant. The main advantage of the arc furnace lies in its flexibility in accepting charge materials in any proportion, namely scrap, molten iron, pre-reduced material and pellets.

It is possible to have precise control of the refining reactions because the electric power can be carefully controlled to impart heat to the bath at different desired rates.

The EAF produces molten steel, which is used for high-grade alloy steel-cutting tools, die steels, and stainless steel, where the metal must be refined and melted under rigidly controlled conditions to minimize the introduction of impurities.

Figure 19.2: EAF Steel Production Flow Chart



Source: AME

19.2 Iron Ore Physical Parameters

Iron ore is generally produced from two types of ores: haematite and magnetite. The type of iron ore deposit will often determine the final iron ore product that can be produced. Magnetite ores are mostly lower in iron content than haematite and as such, must be beneficiated to produce finer grained concentrate products.

Haematite

Haematite ore (“Fe_2O_3”) is a high-grade ore mainly found in large deposits of haematite rock formed by the in-situ enrichment of a protore already enriched in iron, most commonly a banded iron formation (“BIF”) which consists of thin layers of iron oxides. Generally, the range of ore head grades, or contained iron content, for haematite deposits is greater than 55% Fe and may reach levels up to 70% Fe. Haematite ore has commonly been found in large-scale deposits in Brazil, Australia, and India. When haematite ore is of sufficient contained iron content, it may be mined and processed using crushing and screening procedures before being exported for use in steel mills as direct ship ore (“DSO”).

Magnetite

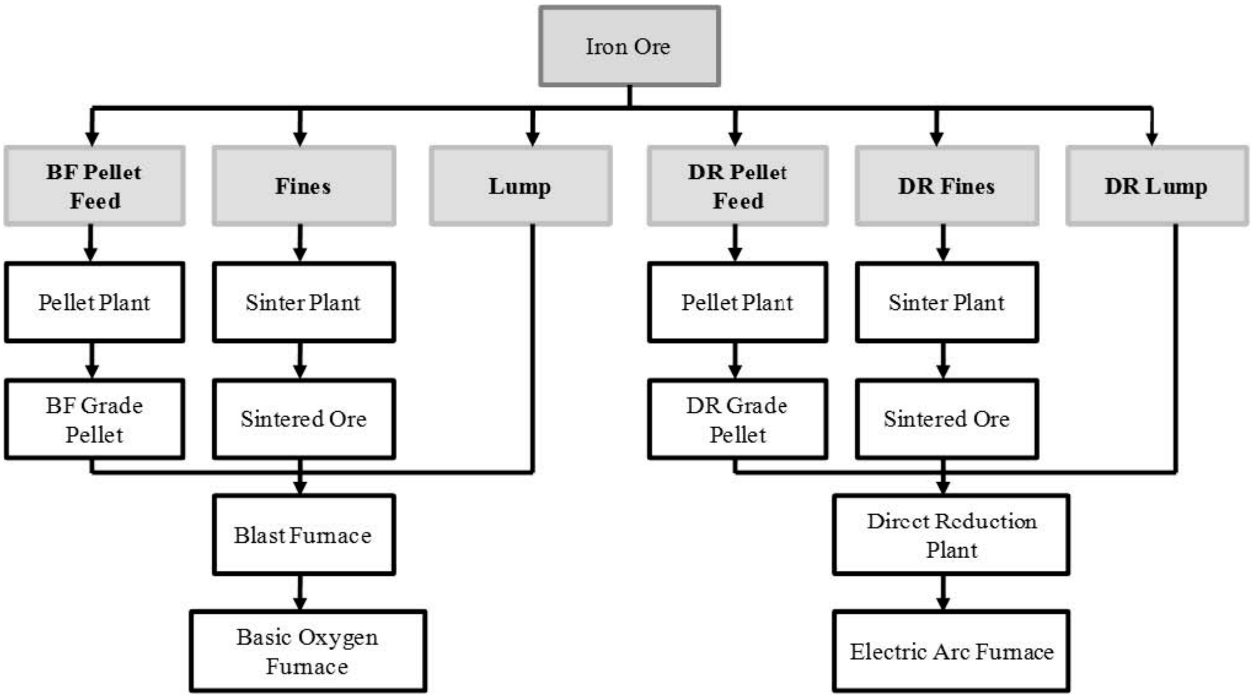
Magnetite (Fe_3O_4) deposits have relatively lower contained iron content than haematite deposits, typically grading between 25-40% Fe. It is mainly found in BIFs located in several countries, including China, Russia, Ukraine, and the Americas.

Due to its lower iron content compared to haematite, magnetite ore requires beneficiation to be converted into a higher grade concentrate product to be viable for commercial use. Magnetite is processed into an iron ore concentrate via the beneficiation process. This concentrate product subsequently becomes feedstock for sinter or pellets, depending on grain size, which are used in the production of steel.

Magnetite is generally magnetic and possesses polarity in its naturally occurring state. The magnetic properties of magnetite permit magnetic separation techniques to produce iron concentrates. Due to its lower iron content compared to haematite ores, magnetite requires an energy intensive grinding process to liberate the iron ore from its associated natural matrix. Different ore bodies require different grind sizes to allow adequate concentration of the iron content. The grinding phase is a costly process in the beneficiation stage which reduces the ore to a fine size and separates the iron from undesirable non-iron impurities such as silica or sand. This is universally achieved through magnetic separation. A variety of magnetic separation techniques can be employed. Each utilises the high density or magnetic properties of the ore to separate the desired iron content from the undesirable non-iron materials.

The most important quality of an iron ore product is the Fe grade, or contained iron content, which is required to be within a particular range for commercial use in different steelmaking processes. While iron ore in-situ grades vary widely within a range of 25% up to around 70%, iron ore products generally contain iron content levels above 50% depending on the steelmaking process. The steel type is largely dictated by the amount of processing each product must undergo before it can be used to make steel, as shown in Figure 19.3.

Figure 19.3: Iron Ore Process Flow



Source: AME

Lump

Lump ores, derived from DSO deposits which are typically haematite and/or goethite rich, are naturally occurring, un-beneficiated coarser lumps of iron ore. Lump products are commonly over 6.3 mm and under 31.5 mm in size. Upon crushing and screening of the ore, lump ores are shipped directly to the steel mills, at which point the product is further screened by size before being fed into the blast furnace. Lump ores that do not break into smaller pieces during transportation and do not quickly decrepitate (break down under thermal load) in the blast furnace are highly valued by steel makers. Lump ores typically command a premium over fines as the product can be directly charged into the blast furnace and relatively fewer deposits worldwide produce lump ore.

Lump ores suitable for direct reduction plants — which require a high percentage of iron concentration, low acid impurity and very low friability feed — are even more limited. Currently, major lump-producing countries include Australia, South Africa, Brazil, and India.

Sinter Fines

Sinter fines make up the bulk of the world's run-of-mine ("ROM") ore supply, estimated to be around 53% for 2017. Due to their small size (below 6 mm), fines are generally considered to be unsuitable for use in a blast furnace without first being processed. Fines are agglomerated into coarser BF / DRI feedstock such as sinter or pellets prior to being used in the iron making process.

Fines products containing iron content closer to the standard 62% Fe benchmark (with relatively limited processing) are commonly derived from haematite DSO fines. These typically range from 55% to 65% Fe grade, and generally consist of particles less than 6 mm in diameter. This product is created through simple processing stages involving screening, crushing, and sometimes wet processing if required. Primarily used as sinter feed, fines are shipped to the customer or a downstream sinter plant for further agglomeration.

Pellet Feed/Concentrates

Pellet feed/concentrates are fines and ultra-fines which have undergone a beneficiation process. Pellet feed/concentrates are generally beneficiated from magnetite ore and exhibits higher iron grades and lower levels of impurities compared to DSO products. While magnetite deposits account for the vast majority of beneficiated ore bodies, there are several notable beneficiated haematite deposits, particularly in North America. These are the Tilden, Carol Lake, Wabush and Mont Wright Deposits. These iron formations consist mainly of haematite, magnetite, and quartz whereby haematite is generally present as specularite.

Concentrates are commonly ground and used as feed to produce pellets used in both the blast furnace and direct reduction processes of iron making. On occasion, concentrate product may be blended with sinter feed in the sintering process.

Relative to DSO products, pellet feed/concentrate possess "value-in-use" qualities which may attract a quality premium in pricing, including comparatively higher iron content, low impurities, and exothermic, energy-efficient qualities in the blast furnace steelmaking process. The price premiums and discounts applied across the value chain vary over time and in particular, higher value products such as lump and pellets are susceptible to changing market conditions.

Pellets

Pellets are the product of an agglomeration process that mixes very fine pellet feed with a binder (e.g., a slurry of bentonite), with the mixture rolled into “green” balls. The product is then fired on a grate or in a kiln to produce the final indurated product, consisting of “balls” with about 8-20 mm. Similar to lump, pellets can be charged straight into a blast furnace or into a direct reduction plant.

Pellets tend to have the highest value-in-use characteristics, and hence have generally commanded the highest pricing premium. Furthermore, pellets generally have a more stable chemical composition and lower levels of impurities compared to sinter or lump. Pelletizing plants in the Commonwealth of Independent States countries, Brazil, Canada, and Sweden are generally located near or are adjacent to mine sites or loading terminals.

Pellets suitable for the production of direct reduced iron generally contain a combined content of silica and alumina below 2%. These high valued products command up to 10% premium over traditional BF grade pellets, given the special chemical requirements.

19.3 Global Steel Demand

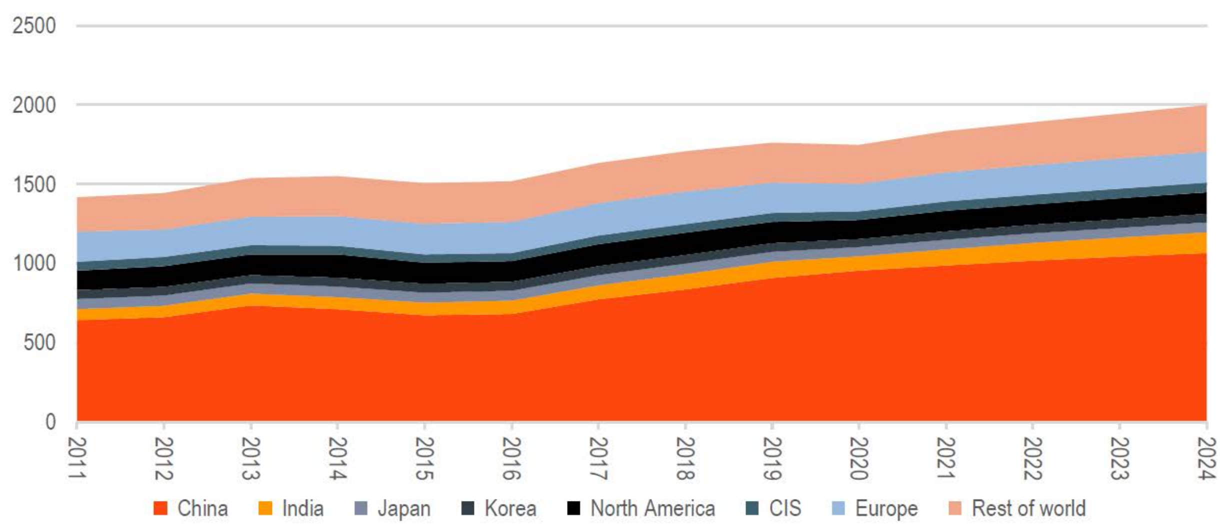
Iron ore is mainly used as a raw material for the manufacture of steel. Steel is one of the most widely used materials in the world given its applications in the construction, automobile, and consumer durables sectors. Therefore, the underlying trend of iron ore demand is primarily driven by that of steel production. Over the past decade, industrialization and urbanization mainly occurred in developing countries such as China. During the course of this development, China became the world’s largest steel producer and also the largest importer of iron ore. Increased urbanization rates in China, along with large investments in construction, power, and infrastructure over the past fifteen years, have driven Chinese steel demand and therefore its demand for iron ore.

Steel is one of the most commonly used metal alloys in the world due to its wide applications. In its various forms of alloys, Steel can be customised for different requirements of construction projects at a relatively low cost. It is also widely used in the automotive market due to its relative strength and low costs when compared with other materials. It is estimated to contribute to around 60% of the weight of an average car. Steel is also featured in a large variety of domestic household appliances and packaging materials. Thus, it is suggested that the consumption for finished steel tracks that of global economic growth and in particular, the pace of urbanization around the world that contributes to large scale construction activities.

Demand in steel markets have been revised slightly upward due to countries such as Japan, US, India, and others implementing stimulus measures with anticipated demand returning as lockdown restriction ease around the globe. Government stimulus packages are being rolled out with debt levels stretching beyond the global financial crisis of 2008. For instance, in Germany their stimulus package is currently USD 157 bn and economic aid an additional USD 90 bn.

AME estimates finished steel demand for 2020 was 1,748 Mt. This equates to a decline of 0.8% for the Year 2020, with a return to growth of 4.9% for the Year 2021. Steel demand will be supported by the infrastructure, residential buildings, mechanical equipment, and automotive (and other transport) industries. However, for developed economies this heavily relies on government stimulus packages in infrastructure projects while emerging economies such as China and India have twenty-year plans to bring its population out of poverty via urbanization policies and the construction of 'mega cities'.

Figure 19.4: Estimated Finished Steel Consumption by Key Country and Region, 2011-2024



Source: AME

Steel demand is highly correlated to individual country's gross domestic product ("GDP"), and with the recent government stimulus packages, these measures are supportive to the current economic climate. In particular, due to the unfortunate events of Covid-19, we are now in a fortunate position from an economic perspective. We are in an expansionary monetary economic cycle (due to low interest rates), along with Keynesian policies, due to recent government stimulus packages. Economic development means countries with high/(lower) steel demand tend to have high/(low) manufacturing, investment in fixed assets and urbanization.

The short-term changes in demand can be attributed to the opening of economies as lockdowns become less frequent. Longer term steel demand forecasts are determined by the unique economic development stage of each country. As can be appreciated, these two broad categories can be classified into developing economies and emerging economies.

For China steel demand growth rate will reduce, as population and urbanization decrease. While in India, steel demand growth rate will be high, caused by significant increase in government infrastructure investment. These projects include transport, offices, hospitals, and manufacturing. Additionally, supportive monetary policy by lowering the interest rate has also been favourable. While economically we are positioned for growth, risk of the vaccine rollout is unknown as governments will need to determine their trade-off between the economy and virus containment.

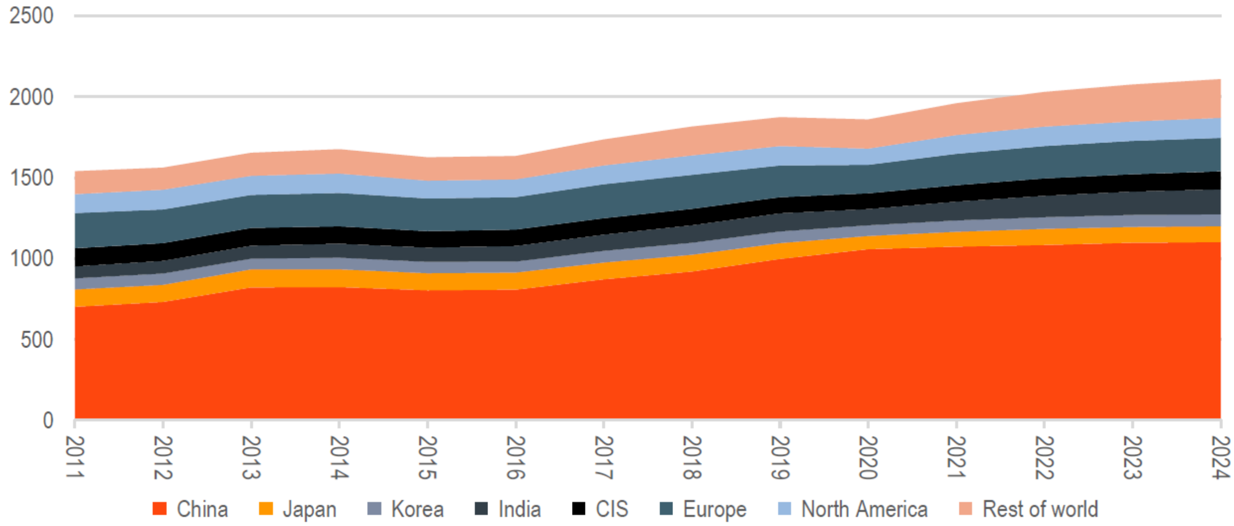
Continued growth in steel consumption is forecasted over the medium term as shown in Figure 19.4.

19.4 Global Steel Production

The steel industry largely operates on a regional basis, as freight costs are typically high in relation to the value of steel product, reducing its competitiveness if the product is shipped for a long distance. In addition, potential trade routes for steel can also be challenged by import quotas, anti-dumping duties and countervailing duty orders.

In 2020, global crude steel production fell 0.8% year-on-year to around 1,858 Mt, as steel output was disturbed by the ongoing impact of Covid-19. Despite this, China's crude steel production rose over 6% year on year to a record 1,056 Mt, as demand remained relatively strong despite the pandemic. Indian crude steel output was down around 10% at 100 Mt for the year, but still remain as the world's second largest steel producer. On the other hand, production from steelmakers in the Commonwealth of Independent States ("CIS") only dropped 2% to 98 Mtpa as producers managed to divert exports from other regions into a voracious China, while Europe saw production plummet by 11%. Estimates of steel production over the medium term are presented in Figure 19.5.

Figure 19.5: Estimated Crude Steel Production by Key Country and Region, 2011-2024



Source: AME

Chinese steel production increased by 95% from 2008 to 2019 (6.3% CAGR) driven by strong economic growth and significant infrastructure investment and construction activity. Steel production growth is forecast to moderate over the next five years as underlying GDP growth moderates and the economy transitions away from heavy fixed asset investment to manufacturing.

Despite the slowdown, China is expected to continue to dominate global steel production. For the first time, its production exceeded the long awaited 1 Btpa in 2020, and accounting for 54% of total global production. By 2024, China's share of global steel production is forecast to decline slightly to around 50% as India's crude steel production experiences robust growth. However, it will remain comfortably above 1 Bt level with estimated production of around 1.04 Bt in 2024.

Crude steel production is required to meet the needs of emerging economies of China, India, Indonesia and Vietnam. Notable supply increases were in China as production levels smash prior quarter numbers, and consequently driving up the price of iron ore. Countries with record crude steel production increases in the Fourth Quarter of 2020 include Turkey, Iran, and Vietnam. In particular, Vietnam has increased production for the year by approximately 50% as their new steelmaking facility of Hoa Phat Group hit the market.

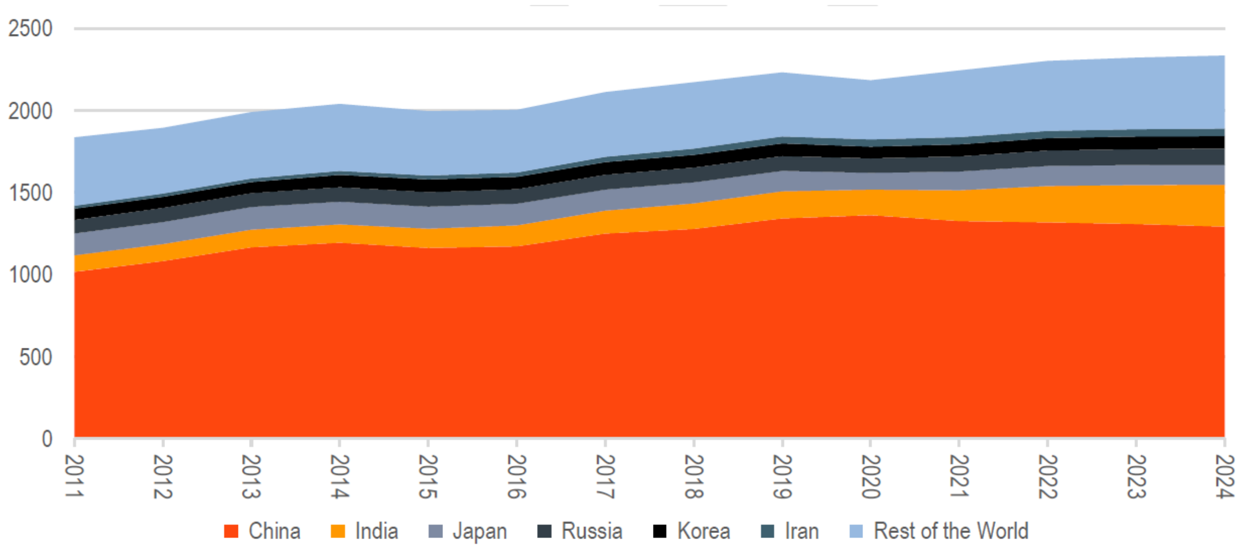
19.5 Iron Ore Demand

AME estimates global iron ore demand increased by 1.8% in 2020 to 2,244 Mdmt, before growing to 2,335 Mdmt by 2024. The economic impact of COVID-19 pandemic and the corresponding containment policies have severely impacted demand across the global economy with finished steel and iron ore no exception.

Policy responses from governments to the pandemic are likely to favor spending on fixed asset and infrastructure projects which will lead to recoveries in demand for steel. Deficit spending is likely to fund these stimulus programs given the historically low interest rates seen worldwide, a palatable outcome politically given the low cost of debt funding. Concentrated stimulus programs targeting infrastructure have been initiated in both China and the US with many economies in the world likely to follow, this poses a potential upside to medium term iron ore demand. The estimated global iron ore demand is presented in Figure 19.6.

Beyond the pandemic, China will remain the largest iron ore consumer over the medium term. China's iron ore demand is expected to decline to 1,292 Mdmt by 2024, as domestic policies drive an increase in EAF steelmaking.

Figure 19.6: Estimated Global Iron Ore Demand by Key Country and Region, 2011-2024



Source: AME

Key drivers for iron ore demand in the long term include the emerging economies of India, Brazil and Vietnam. Indian iron ore demand is driven by government policy to lift Indian crude steel production from 101 Mt in 2017 to 255 Mtpa by 2030. Steel development in India will focus on BF-BOF steelmaking and DRI, which are the major sources of iron ore demand. Vietnam's iron ore demand will be supported by ramp up of BF/BOF dominated steel expansions, while Brazil iron ore demand growth will benefit from its relative proximity to cheap iron ore sources. This will be offset by a decline in Chinese iron ore demand and an increase in scrap substitution.

The greatest shifting factor in iron ore demand will continue to be the use of EAF based processes and the corresponding shift to scrap and alternative metallics consumption over primary iron ore. As redevelopment occurs, more and more scrap will become available in the developing world, cheapening supplies, and undercutting what might be otherwise expensive and growing quantities of iron ore imports. From 2023 to 2032, BOF production is expected to grow at a Compound Annual Growth Rate ("CAGR") of 1.1%, while EAF production is expected to grow at a CAGR of 2.3%. BOF and EAF steelmaking are expected to account for 65% and 35% of total crude steel production by 2030, respectively. The main source of EAF steelmaking growth continues to be China, which is targeting a scrap consumption rate of 30% by 2025, but policy shifts in other developing countries could see this change in the medium to long term. This is despite the overall shift in steel production in India from EAF based to BF / BOF.

The upside for iron ore demand includes slower scrap consumption due to a lack of scrap in the system over the short term. Other risks for the demand outlook include India missing its steel production target or Chinese steel use declining faster than expected.

The key iron ore consuming regions are in Asia as they make up more than 60% of crude steel production. Consistent with the case for crude steel production, the three largest iron ore consuming countries are China (56% share in 2020), India (5%) and Japan (4%). Going forward, we anticipate little change in terms of the production share over the medium term. A notable change will be that as Indian steel production grows at a faster rate than its peer countries, it is anticipated to consume more iron ore. India is expected to have been the second-largest iron ore consuming country in 2020, overtaking Japan and taking up almost 8% share of global iron ore demand.

A slight anomaly is that the US only consumes around 1.4% of global iron ore, despite having a global steel production share of around 4% in 2020. This is attributable to the prevalence of EAF technology for steel production in the US, which uses scrap steel instead of iron ore as a raw material.

19.6 Iron Ore Supply

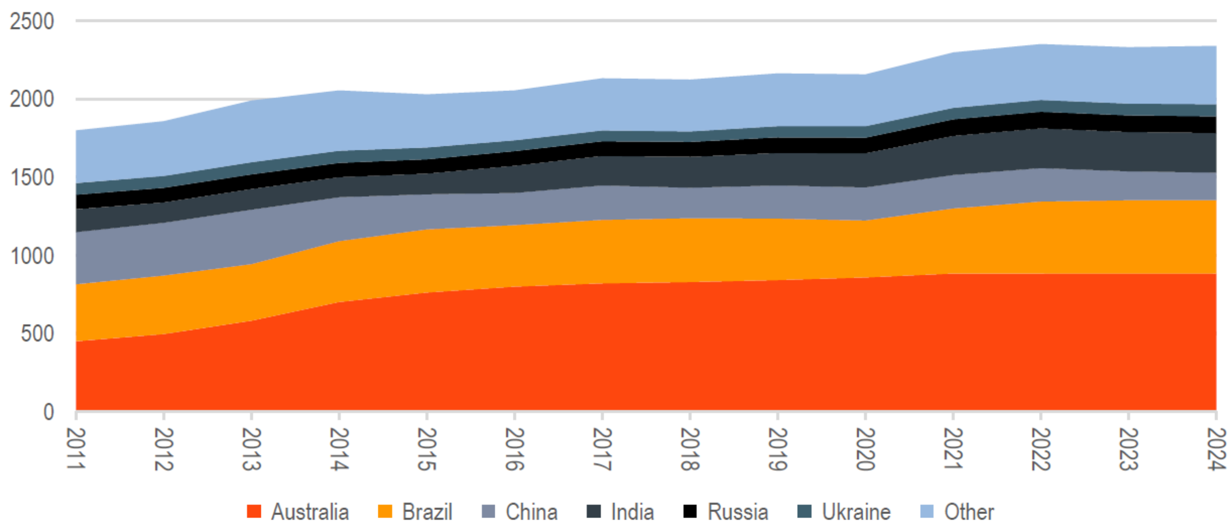
A large proportion of global iron ore supply is not situated in proximity to major steelmaking regions and consequently there is a large internationally traded seaborne iron ore market.

Global iron ore supply is estimated to have decreased by 0.3% in 2020 to 2,158 Mt (Figure 19.7). In the major producing region of Pilbara, mining was considered an essential service and continued to operate at capacity through the COVID-19 restrictions. In the first half of the year, Brazil miners suffered closures at certain sites with Vale halting production at the Conceição, Cauê and Periquito mines at the Itabira complex south-east of the country. Into 2021, global export supply will lift by 7% to 2,300 Mt.

Over the medium to long term, exporters from Australia and Brazil will continue to dominate the global supply. From a 2020 base, Australia will increase production to 941 Mt by 2030 and Brazil will increase production to 501 Mt. Low-cost resource bases and sophisticated infrastructure see the combined countries control 57% of the global export market share.

Despite COVID-19 being economically disastrous for Australia, the iron ore sector remained buoyant, benefiting from high seaborne prices, lack of competitors and increased exports to China. Shipments are back to weekly highs, with many major miners operating close to capacity and generating strong free cash flow at current iron ore prices. Australian operators also benefitted from lower operating costs due to a collapse in oil prices.

Figure 19.7: Estimated Global Iron Ore Supply by Key Country and Region, 2011-2024



Source: AME

The spread of the virus was a significant roadblock for Brazil's iron ore operations and resulted in global supply disruption. In Q3, Vale bounced back from the pandemic and regulatory hangover of the Brumadinho dam disaster and delivered the strongest output in almost two years. Despite this, the company made a downward revision to its production forecasts for 2020 and into 2021. Total supply in 2020 was down around 8% from the previous year to 364 Mt. Over the long term, Brazil will increase its global market share by to 29.3% in 2040, up to 565 Mt.

India has been economically devastated by the COVID-19 pandemic. It has caused shutdowns of iron ore mines, labour shortages and limited transportation. Increased exports to China and the onset of monsoon rains have created an iron ore supply shortage. In Q2 and Q3 2020, iron ore exports were 14.77 Mt and 10.54 Mt, respectively – up from 8.62 Mt and 6.74 Mt in the previous corresponding periods. Concerns surrounding the shortage of supply for the crucial steel ingredient have become increasingly prevalent as downstream activities gradually pick up and prices soar.

The key driver for iron ore supply is the major producers. In the event of the iron ore price falling, the major producers have demonstrated the ability to cut the cost of production and sustain margins, they may also increase or sustain production volume so as to grab market share from the higher-cost, marginal producers. Alternatively, they may slow down to decrease supply in response to low prices.

The major producers typically hold the assets with larger operating scale and lower costs. Vale in Brazil and Rio Tinto and BHP Billiton in Australia have iron ore operations with FOB cash costs as low as USD 10-25/t, attributable to economies of scale and established infrastructure systems. With almost all the major producers occupying the lower half of the cost curve, production growth from the majors can be maintained despite lower prices.

Rio Tinto is the world's second largest iron ore producer, after BHP, and is a leading producer and exporter of iron ore. The company supplies the global seaborne iron ore trade through product operations in the Pilbara region of Western Australia. The iron ore assets include an integrated portfolio of 16 mines, four (4) port terminals, related infrastructure and a 1,700 km rail network. Pilbara Blend products are the most recognised brand of iron ore, making up 70% of the product portfolio. The company was the largest iron ore exporter in 2020 as Vale experienced supply disruption. Shipments of 331 Mt (100% basis) were 1% higher than 2019 and production of 333 Mt (100% basis), 2% higher than 2019.

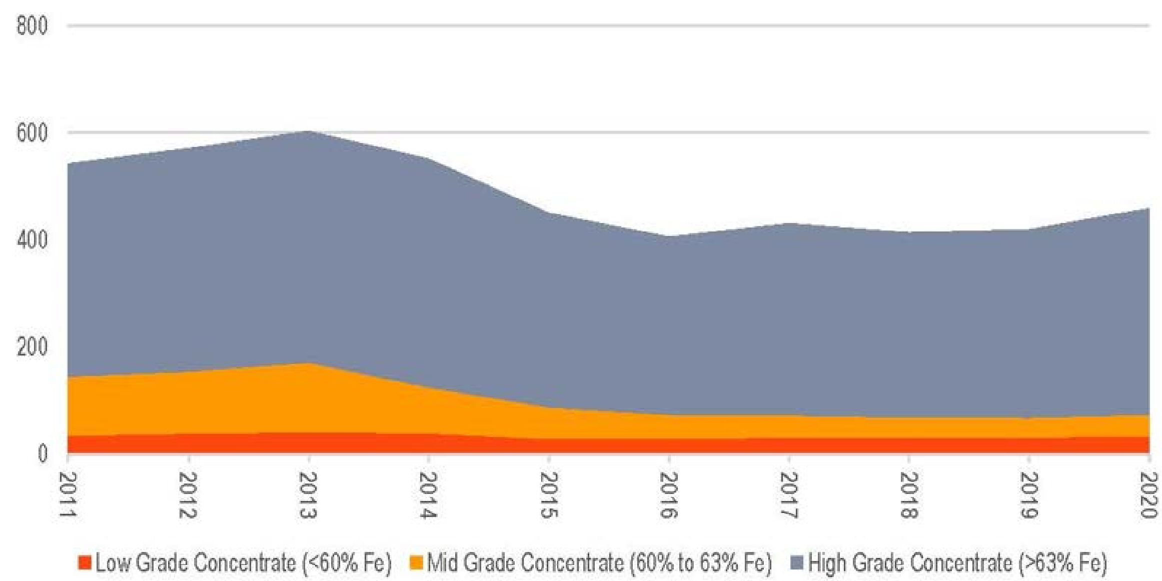
Vale is the largest producer of iron ore and pellets in the world. In the September quarter, the company delivered its strongest output in almost two years, bouncing back from the coronavirus pandemic and regulatory hangover of the Brumadinho Dam disaster. Despite this, the company announced in December that it was reducing its 2020 iron ore output forecast to 300-305 Mt, missing the previously lowered 2020 target by at least 310 Mt. The company expects its 2021 annual production to be between 315-335 Mt.

The availability of lumps is expected to decrease over the longer term, with future iron ore supply expected to be mainly composed of fines and pellets. Since the failure of the Samarco tailings dam in Brazil, and the recent failure in Vale's Southern System, the disposal of tailings generated by wet processing has become a major environmental issue. This has driven development of new dry processing technology for lower grade ores. It is expected over the long term that the highest growth could come from concentrate supply requiring extensive grinding and magnetic and flotation separation techniques as higher grade DSO haematite orebodies are depleted and lower grade magnetite dominant deposits become the dominant ore source.

Undeveloped, high-quality, large-scale deposits are now more commonly located in remote areas, away from existing rail and port facilities, or near infrastructure where there is limited capacity available. New iron ore projects for delivery in the longer term will have higher capital intensity, given the need to develop associated rail or road networks and port infrastructure. Examples are: the 40 Mtpa Mbalam-Nabeba Project in Cameroon and the 100 Mtpa Simandou Project in Guinea.

Production of low and mid-grade iron ore concentrates was traditionally dominated by China. As Chinese iron ore production has diminished, this has resulted in the iron ore concentrate market to be further focused on producing high grades as shown in Figure 19.8. Despite plans for high grade concentrate projects in Australia, most new productions of high-grade concentrate are expected to be focussed on the traditional producing regions of Canada, Europe and the CIS.

Figure 19.8: Estimated Grades of Iron Ore Concentrate, 2011-2020



Source: AME

From a country perspective, Australia, Brazil and China are expected to make up approximately 66% of global iron ore supply in 2020. We anticipate the production share to have few changes in the short to medium term. A notable change is that despite China experiencing flat production this year, it is anticipated to resume its decline as prices weaken and it experience higher estimated cash costs.

Australia

Despite COVID-19 being economically disastrous for some ports of Australia, the iron ore sector has remained buoyant, benefiting from higher seaborne prices and increased exports. China, the country's major trade partner, experienced unprecedented economic recovery from infrastructure stimulus and steel production – ramping up demand for iron ore. Shipments are back to weekly highs, with many major miners operating close to capacity and generating strong free cash flow at current iron ore prices.

Australia is the world's largest producer of iron ore, accounting for 39.4% of supply or 860 Mt. It is also the largest supplier to the seaborne market. The country also has the largest estimated reserves with 52bt-30% of the world's estimated 170 bt. Approximately 90% of the reserves are in Western Australia, with a higher concentration at Pilbara – the premier iron ore region globally. Iron ore mining is crucial to the Australian economy as it is the largest source of export revenue. More than 80% of these exports go to China to support their giant steel manufacturing operations. Other key markets include Japan, South Korea and Taiwan.

Brazil

The coronavirus pandemic was a significant roadblock for Brazil's iron ore operations and resulted in global supply disruption. The rapid spread of the virus in Itabira saw infection rates of around 12% of Vale's workers. In response, the company was required to send sick employees and those who had been in contact with them home – running on minimum staff numbers and reducing productivity rates. The iron ore volumes were also affected by mine shutdowns and heavier than usual rains that resulted in flooding.

Brazil is a distant second to Australia as a global producer of iron ore – accounting for 18% of the world or 364 Mt in 2020. The September quarter has seen a new record in iron ore exports to China – up 49% to 80Mt from the previous quarter. This represents an increase of 29% up from the previous corresponding period. Despite this, total supply in 2020 was down 8% from the previous year.

China

China is not self-sufficient in high-quality iron ore; domestic supply alone is unable to provide the quantity and quality its steel industry requires. China's steel-making giants are dependent on overseas suppliers such as Australia, which typically accounts for around 62% of iron ore imports. China is likely to increase the proportion of imported iron ore as its deposits become of increasingly lower grade the more they are depleted.

China is the third largest producer of iron ore, accounting for 9.9% of global supply or 213 Mt. Almost all of the country's iron ore reserves are used domestically but it is not nearly enough to meet its demand of 1,361 Mt for the same period. These deposits are of a very low grade and are consequently expensive to mine and refine. As the country's history of industrialization grows, so too will its supply of scrap steel which will result in an uptick of mini mills. Domestic iron ore production will decline over the long term down to 144 Mt by 2040.

Coming in as the seventh largest iron-producing country, Canada's export iron ore supply was 60.6 Mt for 2020 – up from 56.9 Mt in 2019. The country will have a general upward trend in supply. An increase to 138 Mt in 2040 represents an increase in global market share to 5.2%. This will be fuelled by an economic response plan to accelerate $2.2 billion in annual federal infrastructure.

West Africa's Simandou region is the world's largest high-grade iron ore deposit and unlocking potential production could trigger a market share battle. It must contain with a 110 km range of hills in southeastern Guinea. Despite a decade of little development progress by Rio Tinto and Aluminum Corporation of China (Chalco), Simandou is squarely back on the radar. The majority of the iron ore would be shipped to China for steel production. Chinese enterprises are particularly interested to reduce their reliance on Australian and Brazilian imports.

The iron ore industry has undergone a period of rationalization over the past six years, with production costs falling up to 50% from their peaks in 2012 and 2013. This cost improvement has allowed more producers to remain in the market longer, despite lower prices; however, this is not seen as sustainable over the longer term. Operating costs will once again increase in real terms, and together with other factors, these may limit the long-term growth potential of global iron ore supply.

Increasing capital intensity is to be expected as ore grades stabilise and then gradually decrease. Lower grades require increased plant size and throughput for the same metal unit output. As the world's most easily accessible and richest orebodies have been largely exploited, mining companies are being forced to dig deeper and are encountering higher strip ratios. Longer transport distances will also lead to greater production costs for newer projects developed in the forecast period. Increasing capital intensity is expected to lift the barriers to entry for new mines. Operating costs are also expected to rise moderately. While labour costs are forecast to increase in real terms, pressure from higher diesel prices are expected to be limited due to an anticipated flat growth profile for the crude oil price in the long term.

In addition to the operation cost perspective, infrastructure limitations and political risks may also provide further constraints.

Infrastructure constraints: Transport networks and the logistics of transporting iron ore from project to market are fundamental determinants to the viability of a project. While the more developed iron ore producers are facing infrastructure delays, transport networks in West Africa and other developing regions are in many cases still under-developed. AME considers that transport will be a material impediment to the development of many projects over the next decade.

For example, if a developing nation has ports infrastructure with relatively limited facilities, this may lead to high throughput time, high turnaround time for ships, slower loading rates and delays due to break-downs. There are also potential difficulties in loading capesize vessels for undeveloped ports, leading to a higher utilization of smaller vessels. These factors may then be translated into port congestions, higher incidence of demurrage costs and overall high costs, eroding margins for iron ore producers and limiting the supply growth rate.

Political Risk: There may be increased political risk associated with exploration and operations in developing areas such as Africa. Political risk is reflected in increased capital costs. This considers the fact that many new developments are remote from the target markets and have unstable government regimes. For politically unstable regimes, supply of key production inputs such as electricity and labour may have considerably higher volatility due to ongoing social issues, giving a cause for production suspension or delay.

19.7 Iron Ore Pricing

19.7.1 <u>Pricing Methodology</u>

Under the traditional "annual contract pricing system", iron ore was priced through annual negotiations between the world's largest steelmakers and their suppliers. A large steel company—usually from Japan or Europe, and more recently from China—would reach a pricing agreement with one of the three largest producers, BHP Billiton, Rio Tinto, or Vale, and the first agreed price would then be set as the point of reference for all subsequent iron ore contracts within a given timeframe. Prices were generally based on the Japanese Financial Year ("JFY") (April 1st of current calendar year to March 31st of following calendar year) during the first quarter of each year.

In JFY 2009, through CISA, the major steel makers in China took a more collaborative role in price negotiations. In the same year, the 40-year-old pricing system broke down as negotiations dragged on beyond the customary April settlement date amid a volatile spot pricing environment. From April 2010, iron ore pricing largely shifted from annual fixed pricing to quarterly pricing.

Spot Pricing

In recent years, spot pricing has been widely adopted among market players with reference to the Platts 62% Fe Fines IODEX ("Iron Ore Index"). Rio Tinto and BHP Billiton now largely negotiate iron ore sales on this basis. In deriving the unit price for the content of iron, the average of Platts 62% Fe IODEX assessments, which are on a CFR North China basis, over a particular historical reference period is taken. From this, freight costs over the same period are deducted. The resulting figure is then divided by 62 to derive the value of each percent of iron content, also called the dry metric tonne unit price. The unit price is multiplied by the iron content of each product to derive an FOB Australia price of fines; a premium is added for lump ores. Additionally, there are extra penalties factored into account for high levels of deleterious materials, as well as when the iron grade drops below the minimum specified in the benchmark.

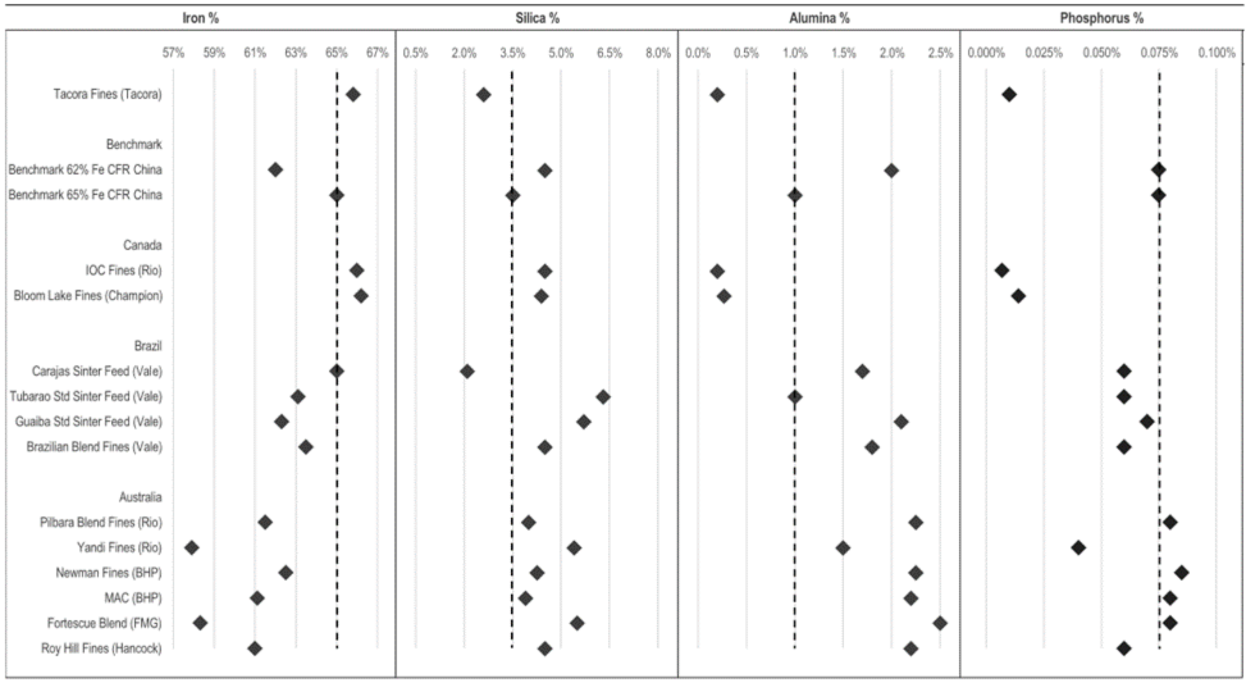
While the pricing method adopted has been largely identical in market settlements, the historical reference period of the IODEX used to determine prices varies from mill to mill. Depending on the levels of iron ore inventories kept at a steel mill and the frequency, mode, and pricing of steel sales, some have used IODEX assessments for the three months preceding the quarter or the three-month period commencing four months prior to the quarter. For example, January to March for determining June prices, or December to February assessments for June prices. Other quotation periods include the month of delivery, five days before and after the notice of readiness at the discharge port.

19.7.2 Value-In-Use Adjustments

In determining the price of a particular iron ore product, miners and steel mills consider four (4) fundamental factors. These are the iron content, the chemical composition/impurities, granulometric characteristics and freight costs. Ores with identical compositions, delivered from the same point of origin will theoretically have the same price on a delivered basis.

Value-in-use is a term used to describe the adjustments made against a benchmark price to account for differences in ore quality. Prices for iron ore products are generally set against the 62% Fe Fines Spot Price CFR North China benchmark prices and adjusted for value-in-use and freight differentials. The benchmark 62% Fe Fines Spot Price is typically considered to have the following quality parameters: 4.5% silica ("SiO_2"), 2% alumina ("Al_2O_3"), 0.075% phosphorous ("P"), 8% moisture and 0.02% sulphur ("S"). The costs incurred at a steel mill are influenced, to an extent, by differing ore chemistries. The premium and discount applied to the benchmark price for a specific ore is calculated based on the difference in iron content to benchmark and the impurity levels relative to trigger grades (i.e., silica over 5.5%). Key impurities considered are silica, alumina, phosphorus and sulphur. Figure 19.9 presents the estimated grades for various iron ore products.

Figure 19.9: Comparison of Selected Sinter Product Estimated Grades



Source: Company Reports, Presentations, AME

A high iron content feed is typically preferred by steelmakers as higher iron content reduces the effective transport costs on a Fe unit basis and increases the iron content yield of the sinter. The benchmark iron content for iron ore is considered to be 62% per dry metric tonne (“dmt”). Currently, the iron unit adjustment is estimated to be around USD 1.20/dmt per 1% of iron content above 62%. Higher iron content tends to attract a premium above the standard adjustment when the iron content is above 63.5%.

Generally, the standard range for silica is from 4.5% to 5.5% to match the desired silica level of sinter products. Therefore, silica levels above 5.5% are considered high and can raise the blast furnace slag volumes and the fuel rate (and, in turn, the coke consumption rate). Silica discounts are currently estimated to be around USD 3.10/dmt per 1% of silica content over 4.5%. Conversely, large amounts of very low silica content ores (less than 1%) often require the use of siliceous ore or olivine to restore normal sinter silica levels. Chinese domestic concentrates typically have a higher silica level of around 5%, and historically have been used to blend with imported fines in the sintering process. Declining Chinese domestic sintering concentrate has led to falling silica discount as Chinese steelmakers have greater tolerance to higher silica seaborne product as a substitute.

Alumina content generally improves the sinter ability of fine ores by enhancing balling characteristics. Higher levels of alumina allow the lime to silica ratio to be higher, and this increases the sinter strength and reduction disintegration index (“RDI”) of the sinter. The RDI is a quantitative measure of the degree of disintegration of the sinter that is likely to occur in the upper portion of the blast furnace after some reduction.

For Australian ores, sales agreements often stipulate penalties for every 1% in excess of anywhere between 2% to 3% alumina. Alumina discounts are currently estimated to be around USD 2.50/dmt per 1% of alumina content over 2%.

Sulphur can have the impact of making steel brittle and primarily enters the blast furnace through coke and coal injection, and to a lesser extent, iron ore. An increase in sulphur content increases the blast furnace sulphur load, the flux requirement, blast furnace slag volume and coke rate, and lifts hot metal desulphurization costs.

High phosphorus levels are detrimental to the quality of the steel and also increase the steelmaking slag volumes and reduce the ability of the furnace operator to recycle BOF slag. As such, phosphorus is especially undesirable in ferrous raw materials. Some Chinese steel makers are known to reject cargoes with phosphorus levels above 0.075% and sulphur above 0.07%. Those that are accepted are typically with material discount requests. The size of the discount will not necessarily be linked to the cost of dephosphorization or desulphurization, as there could be other market-related contributing factors.

Loss on ignition ("LOI") is the weight loss when volatile components bound in the mineral structure are driven off by heating. The main volatile is water-bound in goethite and limonite. The CO_2 bound in carbonate minerals (siderite, calcite, and dolomite) is the other main source of volatiles. Iron ore with a higher LOI impacts on the efficiency of the sintering process though increased fuel rates through the sintering process.

19.7.3 Price Forecasts

AME explains the current status of the iron ore market as the result of highly unusual events. Being by far the largest steel producer and importer of iron ore, China's economic expansion and demand for steel in 2020 compensated for the demand shortfall in the rest of the world. China experienced an iron demand boom in 2020 that drastically contrasts with a slump in the rest of the world. As the COVID-19 pandemic became demonstrably contained within the March quarter, the Chinese government stepped in to stimulate the economy. Significant investment in infrastructure projects substantially renewed strength in manufacturing activity and steel production, increasing demand for iron ore. The country experienced an accelerated run-rate in construction to make up for the loss at the outset of the year.

In addition, the coronavirus pandemic was a significant roadblock for Brazil's iron ore operations and resulted in global supply disruption. The rapid spread of the virus in Itabira saw infection rates of around 12% of Vale's workers. In response, the company was required to send sick employees and those who had been in contact with them home – running on minimum staff numbers and reducing production rates. The iron ore volumes were also affected by mine shutdowns and heavier than usual rains that resulted in flooding. The combination of supply disruption and unprecedented demand from China resulted in a price spike in December to USD 143.27/t.

The impact of COVID-19 on the rest of the world was prolonged until the first quarter of 2021 until control measures were expanded and vaccines become available. Most of the world registered negative growth for 2020, with the exception of China. A major economic recovery is expected in 2021-2022 with economic growth exceeding average levels due to pent up demand and major investments in infrastructures by governments to fuel the economic recoveries.

Several sources were consulted to propose a price forecast for iron ore Benchmarks of 62% Fe and 65% Fe Northern China CFR. They can be included in three different groups:

- Specialized Firms in Metals Markets

  It includes AME and two similar firms that focus on the demand and supply of steel and iron ores globally.

- Mining Analysist

  These are the analysist retained by the research departments of various financial institutions. Our survey identified nine analysts listed in Table 19.1 with their individual forecasts.

- Forward Quoted Prices

  The Bloomberg SGX (Singapore Exchange) presents daily the forward prices for both iron ore Benchmarks for the next three years. It must be noted that the contract volumes are relatively small beyond the next twelve months.

Table 19.1: Iron Ore – 62% CFR China Fines Estimates

Iron Ore - 62% CFR China fines estimates									
Analyst	**Date**	**Units**	**2021**	**2022**	**2023**	**2024**	**2025**	**LT**
UBS	25-Jan-21	US$/dmt	110.00	95.00	85.00	n.a.	n.a.	60.00
Credit Suisse	25-Jan-21	US$/dmt	105.00	n.a.	n.a.	n.a.	n.a.	75.00
BMO	25-Jan-21	US$/dmt	108.00	96.00	78.00	63.00	n.a.	60.00
DB	31-Jan-21	US$/dmt	120.00	90.00	n.a.	n.a.	n.a.	70.00
Jefferies	23-Jan-21	US$/dmt	125.00	100.00	90.00	85.00	80.00	75.00
RBC	26-Jan-21	US$/dmt	122.50	95.00	75.00	n.a.	n.a.	65.00
TD Securities	25-Jan-21	US$/dmt	147.50	100.00	n.a.	n.a.	n.a.	n.a.
BoA	29-Jan-21	US$/dmt	135.00	110.00	100.00	85.00	83.00	84.90
Scotiabank	11-Jan-21	US$/dmt	115.00	85.00	75.00	70.00	70.00	70.00
Average		**US$/dmt**	**120.89**	**96.38**	**83.83**	**75.75**	**77.67**	**69.99**

Our review of the different sources of information indicated that the forecasts from specialized firms and mining analysists underestimate the global economic recovery initiated in China in 2020 and expanding to the rest of the world in 2021 and 2022. Consequently, we used the average forward prices for 2021 and 2022, since it reflects better the global economic growth that will very likely result in a “super cycle” for metals and minerals similar to the period of 2010-2012. Our forecast for 2023 is the average of the forward prices and the estimates form mining analysts. Finally, the forecast for 2024 and beyond is based on a return to normal economic growth and demand for steel and iron ore in a relatively balanced market for steel and iron ore. There is a strong convergence of all forecasts beyond 2023. The various price forecast for iron ore is presented in Table 19.2.

Similarly, historical and forecast for the iron ore Benchmark 62% Fe are plotted on Figure 19.10. It shows that Benchmark prices are well supported in the range of $60-70/t of concentrate and can average $100-110/t over a complete price cycle.

A general widening of the 62% Fe fines and 65% Fe fines pricing spread has been observed since mid-2016; it appears to be driven by higher demand for ferrous products with high-value-in-use, such as high grade and pellets to reduce coking coal consumption. If based only on iron content, it would be expected that the 65% premium would be approximately 4.8%. However, the spread peaked at the end of 2018 at over 31% then back down to 12% by end of 2020.

AME expects this pricing spread to ease marginally in the short term, but remain elevated above long-term levels, as steel mills focus on productivity with an average of around 24%. The long-term shift to higher grade raw material is expected to be driven by ongoing drive to reduce the pollution through increased productivity and efficiency in integrated steel works.

Table 19.2: Iron Ore Price Forecast 62% Fe China / 65% Fe China (USD/t)

Iron Ore Price Forecast 62% Fe China / 65% Fe China (USD/t)						
	2021	**2022**	**2023**	**2024**	**2025**	**LT**
AME (Feb 2021)	93.8/108.8	74/91	70/87	72/89	NA	72/89
FIRM 1 (June 2020)	68/75	64/72	69/79	71/80	73/84	70/83
FIRM 2 (Feb 2021)	113/128	80/94	70/84	74/89	66/79	65/78
Mining Analysts (Jan 2021)	128.89/NA	96.38/NA	83.83/NA	75.75/NA	77.67/NA	69.99/NA
Forward Market Prices (March 15, 2021)	147.99/173.96	112.12/135.29	94.19/116.01	NA	NA	NA
Updated Feasibility Study	147.99/173.96	112.12/135.29	89/110	72/89	72/87	70/85

Figure 19.10: Historical and Forecast Iron Ore Price 2011-2030 (USD/t, Real 2020)

	Historical										Forecast							
	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028
62% Fe Tianjin Spot Price	168	129	136	97	55	58	71	70	93	108	148	112	89	72	72	70	70	70
65% Fe Tianjin Spot Price	182	138	144	111	67	66	86	92	103	122	174	135	110	89	87	85	85	85
Difference 62% vs 65%	14	9	8	14	12	8	15	22	10	14	26	23	21	17	15	15	15	15

19.7.4 Scully Product

Based on the typical specifications for the Scully Sinter product, as supplied by Tacora, AME was able to compare the key parameters to other high-grade fines products, as well as the standard Benchmarks.

Table 19.3: Comparison of Scully Sinter Specification with Peers and Benchmarks

Dry Weight %	Fortescue Blend	62% Fines CFR North China (Benchmark)	Pilbara Blend	65% Fines CFR North China (Benchmark)	Carajas Sinter Feed	Tacora Fines	Bloom Lake Sinter
Iron Content	58	62	61-62	65	65	65.9	66.2
Silica	5.5	4.5	3.5-4.5	3.5	2.1	2.6	4.4
Alumina	2.5	2	2.0-2.5	1	1.7	0.2	0.27
Phophorous	0.08	0.08	0.07-0.09	0.08	0.06	0.01	0.014
Sulphur	0.04	0.02					0.01

Source: AME, Tacora Resources, Company Reports

The iron ore content of the Scully Sinter product, at 65.3% is above the high-grade Benchmark iron content of 65%, and as such would be expected to attract a premium. This is also higher than the largest brand of high-grade sinter in the seaborne market, the Carajas Sinter Feed blend. However, this is less than other Labrador Trough producers, such as Bloom Lake, with 66.2% iron.

Scully's silica content is low at 2.6%, which is low against other Labrador Trough producers and high-grade Benchmarks of 4.4% and 3.5%, respectively. However, this is higher than the Carajas blend. With Chinese domestic ores typically having higher silica content, low silica iron ores are sought for blending.

Scully's alumina levels are very low at 0.2%, which is typical of Labrador Trough ores, and well below benchmark specifications and other high-grade ores. Additionally, with very low phosphorous levels, again below benchmark and other high-grade iron ores, make this advantageous for blending with lower grade ores which typically have higher levels of these deleterious elements.

The LOI of the Scully product is expected to be very low at 0.5%. This is advantageous, as it can increase the efficiency of the sintering process. This compares to a 2-5% LOI typically seen in Brazilian ores and up to 10% LOI seen from some lower grade ores from the Pilbara.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This Feasibility Study section will discuss reasonably available information on environmental, permitting, and social or community factors related to the Project. As this is a recently reactivated site that has operated for more than 50 years, there is a well-developed record of environmental matters for this mine.

20.1 Related Information

Tacora Resources Inc. ("Tacora") prepared and submitted an Environmental Assessment Registration ("EA Registration") to the Government of Newfoundland and Labrador on September 28, 2017 in accordance with the Newfoundland and Labrador Environmental Protection Act (NL EPA, Part 10). The Government placed the document on a public notice period, responded to public comments, and released the reactivation project from further environmental assessment on November 21, 2017.

The EA Registration included discussions regarding the physical features of the Project, natural habitat, potential resource conflicts, and socioeconomic influences of this site. These various factors were considered from reactivation, continued operation and eventual closure and rehabilitation of the mine site.

Following additional permitting requirements, the mine was reactivated, and has been in operation since January 2019, with its first ore to crusher on May 23, 2019, and first seaborne vessel shipment of iron or concentrate on August 26, 2019.

20.2 Environmental Studies

The 2017 EA Registration document described above is the most recent overall environmental study completed at this site. The release outlined several conditions that were required prior to restart of the concentrator and mining. Tacora has fulfilled these conditions. Another EA Registration document was also prepared and submitted by Cliffs Natural Resources ("Cliffs", the previous owners) in November 2015, however, that document was written with a focus on decommissioning and rehabilitation of the mine. This decommissioning project was released on February 29, 2016.

Ahead of reactivation, Tacora prepared four (4) major submissions for approval. This included three (3) plans and one (1) application to the Government of Newfoundland and Labrador in support of the facility which relate to environmental and other operational impacts of resuming operations at the Scully Mine.

These were:

- Reactivation Plan: a document that described Tacora's plan to restore the site from its current condition at the time to operational readiness.
- Development Plan ("DP"): a document that describes the resumed operations of the mine and associated facilities for the remaining mine life.
- Rehabilitation and Closure Plan ("RCP"): a document that outlines mine closure and site rehabilitation for future land use. This plan includes closure cost estimates that were used to document the necessary financial assurance to the Government of Newfoundland and Labrador.
- Operating Certificate of Approval ("CofA" AA18-015646) application: this document describes the environmental control, monitoring and reporting measures that Tacora follows to assure compliance with federal and provincial environmental regulations.

Based on the NL Department of Industry, Energy and Technology ("IET") 2020 Development Plan Guidelines and the Rehabilitation and Closure Plan Guidelines, a proponent is required to submit an updated DP and RCP every five years, or if there is a significant change to the Project. The DP was submitted in 2018, and the RCP in 2017, with an approval from IET for both plans, dated January 23, 2018. Conditions outlined in this approval have also been met. Based on this, the plans are due for an update in January 2023. The CofA was approved in 2018 and expires or is due for renewal on January 22, 2023.

In addition to the above noted major submissions, multiple minor permits and approvals were obtained prior to specific activities being undertaken at the site, related to reactivation and on-going operations. These permits and approvals are outlined in Section 20.5.

20.2.1 Air

The reactivation of the Scully Mine will result in airborne emissions from mining, processing ore, material handling operations and tailings disposal. These are all activities that have been historically associated with this mine site. Airborne emissions challenges at the site are known, documented, and must be managed to minimize any potential adverse environmental impact going forward.

The previous owners conducted air emissions testing (2008 through 2011) to determine pollutant emission rates for various aspects of the facility operations. These rates were then used as some of the input values into an air dispersion computer model that simulate potential ambient air quality impacts to the local area from facility operations over many years. Other model inputs include fugitive emissions from other mining related activity such as storage piles and vehicular traffic. The modeling report from 2014 indicated potential exceedances of ambient air quality standards for total suspended particulate matter ("TSP") and fine particulate matter ($PM_{2.5}$). While the computer simulation predicts possible ambient air quality standard exceedances, two actual ambient air monitoring stations located downwind of the site in the Town of Wabush have not recorded an actual exceedance attributable to facility operations in the multiple years that the monitors have been in service.

20.2.1.1 Air Pollution Control Systems

In December 2020, a plant-wide assessment of Air Pollution Control ("APC") systems serving Tacora's processing operations was completed. The assessment evaluated the current APC equipment throughout the process train, including the Crusher, Classifier, Dryer (bldg. scrubber, not stacks), High Tension Building and the Load Out Building. The assessment included a review of past APC inspection reports, regulatory documents and system drawings along with multiple site visits, flow measurements and discussions with Tacora staff. The assessment acts as a road map for the additional investigations and engineering to improve the APC systems.

The initial assessment identified five APC systems in poor condition that warranted consideration for replacement (Dryer wet scrubber, Classifier wet scrubber, High Tension #2 (HT-DC-002), High Tension #4 (HT-DC-004) and Loadout #2 dry dust collector. Follow-up assessments have identified additional scrubbers for replacement (High Tension 1 & 3, Dryer Stack Scrubbers 1-3, Crusher). Also, ductwork and source ventilation plant-wide have been determined to be inadequate to control indoor air quality requiring substantial improvements.

20.2.2 Upland Habitat

The area around the Scully mine has been heavily used and impacted by industrial mining operations for over fifty years. Most elements of the natural environment have therefore been temporarily or permanently altered by vegetation clearing, overburden/waste rock excavation, and ore mining.

The 2015 Decommissioning and Rehabilitation (Cliffs, Registration 1825) and 2017 Wabush Scully Mine Reactivation (Tacora Resources Inc., Registration 1931) environmental assessment registration documents contain descriptions of the upland environment around Scully Mine, but comprehensive terrestrial studies of this area have never been conducted. The major habitat loss for flora and fauna occurred decades ago when mining first began in this area. There is minimal habitat remaining for large mammals, forest specialist birds and mammals, amphibians, and forest-reliant vegetation. However, the area around Scully Mine still provides some suitable habitat for open habitat specialist songbirds, some species of small mammals, and generalist species like Red Fox.

There are no known Species at Risk using this area, and no rare plants have been documented from areas surrounding Scully Mine. However, surveys would be required to confirm these assertions.

Minimal clearing and grubbing are required as part of on-going operations, however, to avoid adverse effects on potential migratory birds and bird species of special conservation concern, all clearing activities will be conducted in accordance with accepted protocols related to avoidance of disturbing nesting sites. Tacora's no hunting, fishing, or trapping policy will be implemented throughout the construction and operation of the Project, therefore no other wildlife conflicts are anticipated.

20.2.3 Aquatic Habitat

The 2015 (Cliffs) and 2017 (Tacora) EA Registration documents are the most recent studies that discuss aquatic habitat. Over the years, the mine has implemented fisheries habitat compensation and improvement projects for offsetting impacts from various site activities.

The Project site is situated in a region with abundant aquatic resources including many small and large lakes, rivers, and associated streams and fish communities. The area affected by Tacora's operations has been subjected to industrial activity for over 50 years, and fish resources have been able to conduct some, or all their life cycle stages within suitable areas of the mine site.

Tacora is currently fully compliant with all fisheries compensation and offsetting required under the *Fisheries Act*. The Loon Lake Extension and Flora Riverine Compensation Channel projects, required as compensation for tailings deposition in Tacora's Tailings Impoundment Area, have been completed and monitoring concluded in 2014 and 2018, respectively. The Hay Riverine Compensation Channel Project, as required for the removal of the Hay Lake outlet for mining purposes, has also been completed and monitoring concluded in 2018. Tacora has been released from Letters of Credit associated with these projects. The Loon Lake Enhancement Project, as required for the removal of the Hay Lake for mining purposes, will undergo the last year of monitoring in 2021 and it is anticipated Tacora will be released from the Letter of Credit "(LOC") associated with this Project.

Tacora is also continuing all required monitoring of effluent discharges and water quality as required under the federal Metal and Diamond Mining Effluent Regulations ("MDMER"), and provincial (Certificate of Approval) criteria including acute and sub-lethal biological testing.

The reactivation of the mine and associated deposition of mine tailings to the end life of mine of 2047 will result in an expansion of the area affected by tailings and this will require an amendment to Schedule 2 of the MDMER under the *Fisheries Act*. This amendment will generate a requirement for an environmental assessment ("EA"). Continued tailings deposition to ELOM 2047 will impact several lakes and streams in the Flora Basins watershed and Tacora will be required to offset the fish habitat losses associated with these waterbodies. The expanded tailings deposition will affect approximately 360.6 units (100 m^2) of stream habitat and 50 ha of lake habitat. Tacora is currently exploring the technical and financial feasibility options to offset the habitat losses. Tacora will be required to complete a fish offset plan, acceptable to Fisheries and Oceans Canada ("DFO"), and issue Letters of Credit to cover the plan's implementation and monitoring costs as required under the *Fisheries Act*. Tacora will need to complete the fish habitat offsetting prior to any habitat losses associated with continued tailings deposition.

Tacora will implement a no fishing policy throughout the operation of the mine; therefore, no conflicts with recreational fisheries are anticipated.

20.2.4 Cultural and Historic Resources

There are no known historic and heritage resources within the Project area, however, there are no existing environmental studies that focus on cultural or historical resources. The site has already been heavily impacted and is located within an area that has been subject to on-going mining activity for the past five decades. It is therefore unlikely that the area contains, or that the Project will result in the disturbance or destruction of historic and heritage resources. Public access to the Project area is restricted and, land and resource uses, and other activities therefore do not currently take place on the site. No interactions with, or adverse effects upon, commercial, municipal, traditional or recreational activities in the area are therefore anticipated.

The 2017 EA Registration document completed by Tacora discussed positive socio-economic impacts of the reactivation of an existing brownfield industrial site that has operated for more than 50 years.

20.2.5 Water

The 2017 EA Registration, Development Plan and Rehabilitation & Closure Plan all describe water resource impacts related to reactivation, operations and eventual closure of the Tacora Mine. Surface water quality monitoring at the mine site was conducted for many years and there is a long, well documented history of water quality monitoring results.

20.2.6 Hazardous Materials Management

20.2.6.1 Soil and Groundwater Contamination

The facility has operated a fuel tank farm for many years, storing Bunker C fuel oil, diesel fuel and some small amount of gasoline. Over the years, there have been documented spills to the extent that Tacora has completed several environmental site assessments ("ESAs"), groundwater monitoring programs, a remedial options review and Remedial Action Plan ("RAP") for the fuel transfer area and former gas storage tank farm. In the Fall of 2020, Tacora initiated a multi-year assessment and remedial program to manage the mixed gas/diesel plume identified at the reference site. Further details are discussed below in Subsection 20.3.

20.2.6.2 Lead Based Paint, Asbestos Containing Materials, and PCBs

Due to the site's age, there are areas of facility's structures that contain lead-based paint, asbestos containing materials ("ACM") and a few remaining PCB filled fluorescent light ballasts. The province defines lead-based paint as having greater than 0.06% lead by weight, and ACM with asbestos concentrations greater than 1%. The most recent site survey was conducted in October 2019.

The previous owners confirmed to the Provincial government in a letter from May 2002 that as of 1997 the only remaining equipment on site with the potential to contain PCBs in concentrations greater than 50 ppm may be some fluorescent light ballasts.

Tacora has committed to the removal of ACMs on a monthly basis to reduce quantities remaining on site. The most recent removal occurred February 2021.

20.2.7 Tailings and Waste Rock

20.2.7.1 Tailings Impoundment Area

There are documented Dam Safety Inspections ("DSIs") of the Tailings Impoundment Area ("TIA") for 2012, 2013 and 2014, these inspections were conducted by Golder and Associates ("Golder"). The reports indicated that the dikes were stable and water levels (measured via piezometers) remained at levels that do not impact dike stability. As expected, water levels dropped measurably after plant operations were curtailed and the TIA continued to dewater.

Following the restart of operations, the DSIs were resumed in 2019 by Golder and in 2020 by Hatch, their findings indicated that the north, south, and saddle dikes were in satisfactory condition. With the reactivation of tailings slurry discharges from the South Dike, the existing standpipe piezometers reported increasing water levels as expected and consistent with the historic operations.

In December 2012, Cliffs also retained AECOM to complete an external audit of the dike design, suitability and operational condition of the existing TIA dikes.

There is a draft TIA Dam Safety Review prepared by AMEC in December 2016. This report was prepared in support of the intended mine closure and implementation of the Rehabilitation & Closure Plan for the TIA. As such, it emphasized ongoing revegetation efforts and surface water management to minimize potential erosion impacts on water quality.

In the Government of Canada Gazette, Regulations Amending the Metal Mining Effluent Regulations (now Metal and Diamond Mining Effluent Regulations), under the Fisheries Act, issued an Amendment on February 5, 2009, to designate waterbodies as TIAs for the Scully Mine. Within the amendment, it states that the tailings at Scully Mine are non-toxic and non-acid generating.

The geology supports this, whereas ore minerals being extracted at the Scully Mine are hematite, magnetite, and martite while the waste is predominantly quartz and hydrated iron oxides such as limonite and goethite. These ore and waste units are overlain by a glacial till which varies in thickness across the property. There are no significant amounts of ARD related minerals (e.g. sulfides) present in the Scully deposits. In addition, since the restart of the Scully Mine, effluent and water quality samples are collected and analyzed on a weekly, monthly and quarterly schedule as required under Tacora's Certificate of Approval ("CofA") and Environment and Climate Change Canada ("ECCC"), MDMER at final discharge points and reference and exposure sites. Metal levels for these samples have not exceeded the limits specified in these regulations.

The above supports the understanding that ARD / ML is not a concern for the Site. However, to further confirm this concept, and per the IET 2020 Rehabilitation and Closure Guidelines requirements, Tacora is committed to engaging an ARD / ML Qualified Professional to assess the need to complete a program and report to characterize and evaluate the potential for and mitigation of ARD / ML at Scully Mine, prior to the 2023 update of the Rehabilitation and Closure Plan.

20.2.7.2 Waste Rock Piles

AMEC completed a slope stability evaluation of waste rock piles ("WRP") in November 2016. The objectives of this evaluation were to provide a preliminary assessment of the stability of WRP, determine the location of safety berms on the current waste rock piles, and answer geotechnical questions in the Comments on Scully Mine / Scully Mine Rehabilitation and Closure Plan sent in April 2016 by the Department of Natural Resources ("DNR"), now the Department of Industry, Energy and Technology ("IET"). Based on AMEC's study, the WRPs are currently constructed at the angle of repose, in their current state the WRPs are considered marginally stable. While the WRPs have been in place for the life of the mine, there is evidence of tension cracks at the top of some slopes and of previous slope failure.

From an engineering standpoint, the recommendation for slope stabilization, and determination of slope factor of safety ("FoS"), depend on the future planned use for the area and a review of the consequences in the event of a slope failure. Currently there are no official guidelines for the required FoS for slope stability. The required FoS for seismic slope stability is generally recommended to be 1.1 or greater. From the slope stability analysis performed in the AMEC WRP study, it was found that a slope of approximately 2H:1V may provide a FoS of 1.4 to 1.5. However, this study was performed without a comprehensive geotechnical investigation. Depending on the future use of the land around the mine, and lack of geotechnical information, a FoS of two may be warranted, which would require a reduction of slopes to approximately 3H:1V. Upon recommendation by EOR, a Geotechnical investigation is planned for 2023, to provide a more accurate slope stability analysis.

20.3 Environmental Issues

The following environmental issues will not materially affect Tacora's ability to extract the mineral resources but are existing conditions that require continued monitoring and ongoing resolution.

20.3.1 Ambient Air Quality

As discussed above in Subsection 20.2.1, although the ambient air quality dispersion model predicted the site would have exceedances, the installed air monitoring stations have not shown any actual exceedances attributed to the mine, the regulatory expectation is that the computer model results are satisfactory.

Under the CofA, Tacora is currently required to complete stack emission testing and dispersion modelling every two years. Upon demonstrating that the mine meets the guidelines, the testing and modelling can be completed every four years. Tacora's EA Registration document commits to completing stack testing within 12 months of resuming operations. Stack testing has yet to be completed. A request was submitted to

NL DECC in December 2020 to postpone the first round of stack testing until Spring 2021. This is to allow improvement to the APC systems in advance of stack testing. The last round of stack testing was last completed at the Mine between September 2011 and January 2012. Results from this test program were used in a dispersion model, with results reported in January 2014. The modelling report identified that the Mine exceeded ground-level dust concentrations.

To meet the CofA requirements, source testing will be completed in 2021 and every subsequent two years. These results will be used for air dispersion modelling of the site for submission to the NL government. This submission will support the ongoing requirements for demonstration of compliance with provincial air emissions regulations.

Past dispersion modelling has also indicated exceedances of the air emission limits under Tacora's CofA. Indoor air quality testing by Tacora has identified exceedances of the NL Silica Code of Practice. The overall objective of the ongoing APC work is to:

- Control the workspace concentrations of dust and other pollutants generated by the various operations of the ore processing and material handling operations at the facility to comply with provincial workplace requirements.

- Control emissions to the atmosphere through improvements in air pollution control systems to comply with Tacora's CofA from the NL DECC.

Tacora has established a five-year plan to recapitalize their APC systems to meet these objectives. This includes engineer assessment, design and implementation of new dust collectors and ancillary systems in the High Tension Building (2021), Dryer building (2022), Loadout (2023), and Crusher & Classifier (2024).

The CofA stipulates that APC equipment shall be operated and maintained in a manner that ensures optimum performance. The regulator may require installing additional APC systems as necessary to comply with the Regulations.

20.3.2 Soil / Groundwater Contamination

As noted in Subsection 20.2.6.1, a multi-year assessment has been initiated. According to Tacora's consultant, the multi-year program's objective is to reduce the environmental risk to both ecological and human receptors in the study area, obtain site closure for this impacted area with the NL DECC and reduce the environmental liability associated with this impacted area for Tacora. The main aspects of the multi-year program will be: collection of sufficient site data each year to refine the remedial program for subsequent years, develop a conceptual site model and contribute to a site closure report that will very likely consist of a quantitative human health and ecological risk assessment and groundwater model at the later stages of the program.

Remedial work will focus on product removal and remediation of the plume's highest impacted areas in the initial years combined with yearly passive in-situ treatment of the plume to stabilize and enhance natural attenuation processes. In essence, a risk-managed approach that combines focused remediation of the highest risk potential area of the plume and management of residual impacts.

In the Fall of 2020, year one of the five-year program was initiated with the following work items completed: remediation of 475 t of impacted soil, installation of two recovery wells in the plume area, impacted groundwater/free product removed and treated from the recovery wells, groundwater monitoring program completed, and additional delineation in the plume area was completed. Year two of the five-year remedial and assessment program is currently planned to start in June 2021.

20.4 Operational Requirements

The environmental operating requirements are specified within the numerous permits, approvals and authorizations, including the major approvals; the Development Plan (ongoing operations), and the Rehabilitation & Closure Plan (post-closure activities), the CofA, the Waste Management System CofA (LB-WMS20-01023J), and site environmental management plans such as the Environmental Protection Plan (“EPP”), the Spill Contingency Plan, the MDMER Emergency Response Plan (“ERP”), the Waste Management Plan (WMP), and other minor approvals as outlined in Subsection 20.5.

20.4.1 Tailings and Waste Management

Tailings deposition will continue in the Flora Lake TIA and is described within Subsection 18.4 of this Study. The environmental considerations with respect to this part of the facility operations is largely that of ongoing vegetation efforts that improve basin stability and minimize the potential for both erosion and fugitive dust emissions. The vegetation work is an integral portion of the progressive rehabilitation work specified within the Rehabilitation and Closure Plan. In addition, the management of the tailings deposition to ensure that the natural drainage pathway of the Flora watershed (i.e., alignment and profile of Diversion Channel) allows unobstructed flows and facilitates ease of progressive construction for permanent engineered channel that meets the closure purpose.

Slope stability and additional vegetation of waste rock stockpiles is another aspect of the progressive rehabilitation work identified in the Rehabilitation and Closure Plan. To the extent supported by the mining plans, future waste rock placement can be incorporated into the existing stockpile locations to improve existing stockpile stability. Then additional vegetation work can be incorporated to reduce erosion potential and thus further improve slope stability.

Solid waste management and on-site disposal will continue under provincial regulatory requirements and the site specific CofA for landfill operations (LB-WMS20-01023J). The CofA includes descriptions of allowed and disallowed materials, management techniques and reporting requirements to ensure that materials are properly handled.

20.4.2 Site Monitoring

Water quality monitoring has been conducted to address requirements under MDMER and includes monitoring effluent quality and water quality in receiving waters and at reference locations on a schedule as specified in the regulations. This also includes sub-lethal toxicity testing annually for the Flora Lake FDP. Monitoring results are reported to ECCC through their online Mining Effluent Reporting System ("MERS"). Environmental Effects Monitoring ("EEM") is also conducted under the MDMER on a three-year cycle with the most recent cycle (Cycle 6) reported on November 2020.

Metal mining environmental effects monitoring data are submitted into the Environmental Effects Monitoring Electronic Reporting ("EEMER") and MERS as maintained by ECCC. Tacora has reviewed historical data in EEMER for exposure (Wabush Narrows, Long Lake, and Flora Outlet Arm) and reference sites (Walsh River and Jean River) for exceedances of MDMER authorized limits of arsenic, copper, lead, nickel, zinc and total suspended solids ("TSS"). The only exceedances in that database for the period 2004 to 2020 are for TSS (> 30 mg/L) at Flora Outlet Arm associated with seasonal spring runoffs that temporarily re-suspend solids during high flow rates. A TSS exceedance was also reported at Jean River in June 2019 and this was considered an anomaly.

Water and effluent quality monitoring are conducted under the mine's Certificate of Operations ("CofA"), including acute lethality biological testing. Monitoring results are reported to the NLDECC through their Environmental Database Management System ("EDMS").

Tacora will continue monitoring of effluent discharges and water quality as required under the federal (MDMER) and provincial (CofA) criteria, including acute and sub-lethal biological testing. Table 20.1 provides a summary of water and effluent quality sampling locations, as related to the applicable regulations.

Table 20.1: MDMER and CofA Water Quality Monitoring Stations

Sample Point	Commonly used Abbreviations	Site Type	Regulatory Requirement										
			Federal MDMER					Provincial WRMD	Provincial CofA				EEM
			Weekly Effluent Monitoring & Flow Measurement	Monthly ALT	Quarterly EC	Quarterly Sublethal Toxicity	4x/y. WQM	Real Time Streamflow and Water Quality	Weekly EMP	Monthly ALT	Monthly TPH Testing	4x/y WCA	
Tailings Line Emergency Dump Basin 1	TLED-B1 or TED	Final Discharge Point	X	X	X		X		X	X	X	X	X
East Pit 2 Dewatering	ED2	Final Discharge Point	X	X	X		X		X	X	X	X	X
West Pit Extension Settling Basin	WES	Final Discharge Point	X	X	X		X		X	X	X	X	X
Knoll Lake Inflow	KLYD	Final Discharge Point	X	X	X		X		X	X	X	X	X
Flora Lake Discharge	FLYD	Final Discharge Point	X	X	X	X	X		X	X	X	X	X
Jean River	JR	Reference Area					X					X	X
Walsh River *also known as Long Lake Outlet	WR	Reference Area					X					X	X
West Pit Dewatering Settling Basin (Downstream, Long Lake) *also known as Long Lake	LL	Exposure Area					X					X	X
Wabush Narrows	WNAR	Exposure Area					X					X	X
Flora Outlet Arm *also known as Flora Creek	FOA	Exposure Area					X	X				X	X
West Pit Deep Well	WPDW	EEM Site only										X	X
Tailings Line	TDES	EEM Site only										X	X
Virot Lake *also known as Virot Lake Outlet	VL	EEM Site only										X	X

ALT = *Acute lethality testing (for Tacora these include rainbow trout and Daphnia magna)*

CofA = *Certificate of Approval*

EC = *Effluent characterization*

EMP = *Effluent monitoring program*

EEM = *Environmental effects monitoring (required by both MDMER & CofA)*

MDMER = *Metal and Diamond Mining Effluent Regulations*

WCA = *Water chemistry analysis*

WQM = *Water quality monitoring*

WRMD = *Water Resources Management Division*

In addition, Tacora operates two ambient air monitoring stations in the town of Wabush – one located near the Provincial Building and another near JRS Middle School. Three parameters relevant to Tacora's operations are monitored between the two stations: (1) TSP; (2) particulate matter less than 2.5 microns in diameter ($PM_{2.5}$); and (3) sulfur dioxide (SO_2). TSP is used as the primary nuisance dust measurement and $PM_{2.5}$ is used as the health-based dust measurement. The measurement of SO_2 is used to determine the impact of sulfur-based fuel combustion on ambient air quality. The Provincial Building station monitors TSP, $PM_{2.5}$ and SO_2 whereas the station near JRS Middle School measures TSP and $PM_{2.5}$. Tacora installed a new SO_2 analyzer at the Provincial Building station in November 2020. Both air monitoring stations are calibrated quarterly by Tacora and monitored in real time by the Newfoundland and Labrador Department of Environment and Climate Change - Pollution Prevention Division. The department also conducts quarterly performance audits.

20.4.3 Water Management

Surface water drainage patterns will not materially change during the site reactivation period. Any reactivation related grading changes will be temporary in nature and the surface watersheds will remain as with previous operations. At the plant site, surface water drains into the settling basins south of the plant, where it is treated and eventually released into Knoll Lake. There are no planned modifications to any watershed associated with the plant site.

Water treatment associated with tailings management consists of natural (unaided) settling of solids in Flora Lake, approved under MDMER for tailings management. Flora South acts as a settling basin before water is passed onto Flora North. Water quality in Flora Lake, as measured at Flora Lake Discharge (Final Discharge Point under MDMER), which then flows to the exposure site Flora Lake Outlet Arm, is consistently in compliance with the metal criteria in the federal MDMER and provincial Effluent discharge Criteria in Table 5 of Certificate of Approval AA18-015646 and the acute lethality criteria in the MDMER and the CofA. However, TSS measurements typically exceed criteria during springtime monitoring events.

TSS exceedances in the Spring are attributable to on-going spring thaw and run-off resulting in higher-than-normal water levels increasing the TSS levels in all areas. The large amounts of snow melt during this period attributes to the amount of runoff. In addition, the forest fire of 2013, which surrounded Flora Lake, destroyed vegetation that would have previously aided in reducing the flow rate and natural filtration of any run-off. TSS levels are expected to diminish during the spring runoff period as the tailing's revegetation program proceeds and the vegetation matures and inhibits the erosion due to high runoff.

Pit dewatering is an ongoing part of this facility that has the potential to impact surface water quality at the associated discharge points. There are no anticipated concerns with pollutants during reactivation as all discharges will be pumped water from the pits with no mineral processing activity. During operations, pit dewatering discharge locations are routinely and frequently sampled for water quality as part of the MDMER, CofA, and NLDECC compliance.

20.4.3.1 Hydrometric and Water Quality Stations Agreement

Tacora entered into a Memorandum of Agreement ("MOA") with the NL DECC whereas hydrometric and water quality stations are to be installed, operated and maintained at site to monitor real-time data. There is currently one station installed and being monitored, located at Flora Creek below the Trans Labrador Highway.

20.4.3.2 Water Use Licences

In addition to the above noted water quality monitoring requirements, Tacora has also obtained two Water Use Licences, one for water withdrawal and use from Little Wabush Lake (WUL-18-9503), and one for water withdrawal and dewatering from West Extension Pit, West Pit, South Pit and East Pit (West) near Labrador City and the Town of Wabush (WUL-18-9504). Within these licences, there are conditions, commitments, and annual reporting requirements to be met. Additional detail on these approvals is provided in Subsection 20.5.1.

20.5 Required Permits and Status

20.5.1 Permits

The Tacora Mine has several regulatory permits that must be maintained, updated or renewed to operate. These requirements and associated status are shown in

Table 20.2.

Table 20.2: Tacora Mine Permits

Issuing Department	Title	Date Issued	Expiry Date	Status
Federal				
Environment and Climate Change Canada (ECCC)	Amendment to the Metal and Diamond Mining Effluent Regulations Designating Flora Lake and Three Streams as a Tailings Impoundment Area (TIA)	Feb 5, 2009	N/A	Amendment process required, baseline study work to begin summer of 2021
Fisheries and Oceans Canada (DFO)	Fisheries Act Authorization (FAA) for the Vern-Hay Project	Amended Jan 15, 2018	N/A	
Natural Resources Canada	Licence to Store, Manufacture or Handle Explosives (Licence U300903/E)	Sept 24, 2020	Sept 30, 2021	Shared licence with Orica
Provincial				
Newfoundland and Labrador **Department of Environment and Climate Change (NLDECC)**	Environmental Assessment Registration	Nov 21, 2017	N/A	Project Released
	Water Use Licence – Industrial (Mining), (WUL-18-9503)	Jan 18, 2018	Jan 18, 2023	
	Water Use Licence – Industrial (Mining-Pit Dewatering), (WUL-18-9504)	Jan 18, 2018	Jan 18, 2023	
	Certificate of Approval (CofA), (#AA18-015646)	Jan 22, 2018	Jan 22, 2023	
	Memorandum of Agreement (MoA) Hydrometric and Water Quality Stations – Tacora Mine	Feb 02, 2018	Mar 21, 2022	
	Permit to Alter a Body of Water	Dec 22, 2020	Dec 22, 2021	
Newfoundland and Labrador Department of Industry, Energy and Technology (IET)	Mill License (ML-TRI-01)	Oct 19, 2018	Oct 31, 2023	
	Development Plan	Jan 23, 2018	Update required every 5 years	
	Reclamation and Closure Plan	Jan 23, 2018	Update required every 5 years	
Digital Government and Service NL	Certificate of Approval (CofA), Waste Management (LB-WMS20-01023J)	Jan 1, 2021	Dec 31, 2021	Requested transfer of existing CofA to TACORA on November 27, 2017
	Blasters Safety Certificate	TBD	TBD	Contractor to obtain approvals
	Magazine License	TBD	TBD	Contractor to obtain approvals
	Approval for Storage and Handling of Gasoline and Associated Products	TBD	TBD	Renewal and transfer of previous approvals is required
	Fuel Tank Registration	TBD	TBD	As above
	Approval for Used Oil Storage Tank System	TBD	TBD	As above
	Material Lift Certificates	TBD	TBD	As above
	Boiler and Pressure Vessels	TBD	TBD	As above
Municipal				
Town of Wabush	Building Permit (Tailings Drainage Culvert)	Dec 1, 2020	Dec 1, 2021	

In addition, as noted in Subsection 20.2.3, an expansion to the tailings impoundment area will require an amendment to Schedule 2 of the MDMER, which will generate the requirement for a series of associated regulatory approvals. Please refer to the anticipated list of approvals required as part of this amendment in Table 20.3 below.

Table 20.3: Anticipated Approvals Required for Tailings Expansion

Issuing Department	Title
Federal	
Environment and Climate Change Canada (ECCC)	Amendment to the Metal and Diamond Mining Effluent Regulations Designating Flora Lake and Three Streams as a Tailings Impoundment Area (TIA)
	Alternatives Assessment
Fisheries and Oceans Canada (DFO)	Fisheries Act Authorization (FAA)
	Fish Habitat Offsetting Plan
	Experimental Licence (for Baseline data collection on fish populations and future monitoring activities)
Impact Assessment Agency of Canada (formerly Canadian Environmental Assessment Agency or CEAA)	Federal Environmental Assessment
Transport Canada	Navigable Waters Assessment Act – Notice of Works
Provincial	
Newfoundland and Labrador Department of Environment and Climate Change (NLDECC)	Environmental Assessment Registration
	Water Use Licence – Industrial (Mining)
	Certificate of Approval (CofA), (#AA18-015646) Amendment
	Permit to Alter a Body of Water (for instream works)
Newfoundland and Labrador Department of Industry, Energy and Technology (IET)	Development Plan Amendment
	Rehabilitation and Closure Plan Amendment
	Quarry Permit
	Commercial Cutting Permit
Municipal	
Town of Wabush	Building Permit (General Development Application)

20.5.2 Bonds

There are three (3) financial assurance requirements that Tacora had to put forward regarding environmental remediation protection; one for mine reclamation and two related to fisheries habitat restoration. These requirements have been met by issuing appropriate LOC or escrow account to the required regulatory authorities.

Table 20.4: Financial Assurance Requirement

Program	Regulatory Authority	Why	Amount & Status
Rehabilitation & Closure Plan	IET	Closure (without reactivation)	CAD 36.751 MM escrow account established July 18, 2017
		Mine Closure	Before resuming operations, escrow account balance must increase to CAD 41.738 MM
Authorization for Permanent Alteration, Disruption or Destruction (HADD) of Fish Habitat, under the *Fisheries Act*	Fisheries and Oceans Canada (DFO)	Flora Riverine Monitoring Program	CAD 97k LOC issued July 17, 2017
			Closed / Returned January 10, 2020
Fisheries Act Authorization #06-04-001 amendment 2		Hay Riverine & Loon Lake Enhancement	CAD 111.5k LOC issued July 17, 2017
			Reduced to $49,675 November 26, 2019

20.6 Social and Community

Tacora has been in regular contact with the local communities as well as with the Indigenous governments and organizations having a stated interest to the area. These community and stakeholder consultation activities have included frequent meetings with Mayors and Councils, local businesses, Indigenous leaders, local political representatives, local interest groups, provincial and federal regulators.

Tacora Resources has developed a good communication strategy with both the Towns of Labrador City and Wabush, mostly through virtual means during the Covid pandemic. Tacora is part of Labrador West group that has met on several occasions (virtually) to discuss Covid concerns in the Labrador West region. We keep them informed on our Covid restrictions, requirements, and procedures, particularly around any out-of-province workers we may have on site.

Tacora has a budgeted fund to which community groups may apply for funding assistance. All requests are assessed under a criteria matrix and each request is evaluated on an individual basis. In 2020, Tacora donated to such groups as the Labrador West Minor Hockey, Labrador West Food Bank, Wabush Figure Skating Club, and other non-profit organizations.

Tacora Resources also has a membership in the Labrador West Chamber of Commerce.

20.6.1 Municipalities

Tacora Resources has entered into a 3-year agreement with the Town of Wabush, whereby Tacora pays a grant to the Town in the amount of $1.6 million per year. The agreement expires in 2022. In addition, Tacora has agreed, in partnership with the Town of Labrador City and the Iron Ore Company of Canada to pay the Town of Wabush a sum of $200,000 for three years, from 2019 to 2021. These types of agreements are typical in the region, as the communities depend on the mining industry to support municipal capabilities such as emergency response.

Tacora and the Town of Wabush have additionally formed an Emergency Response Agreement as of 2020, whereby the Town of Wabush would provide services and support to the mine site in an emergency situation.

20.6.2 Indigenous Communities

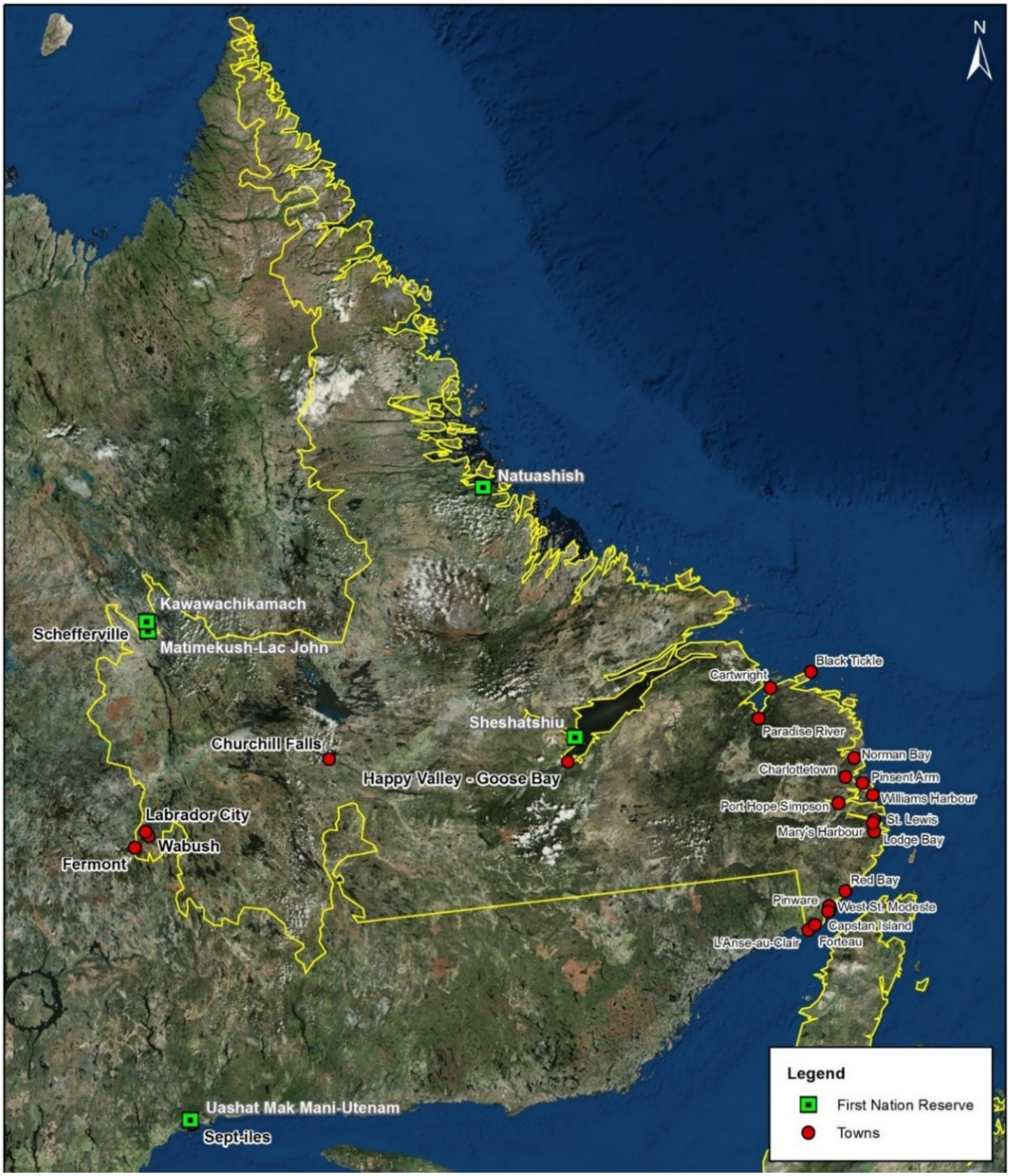
The Quebec-Labrador Peninsula area has five distinct Indigenous groups claiming traditional and Aboriginal rights to all or part of the Project area. The communities served by these groups and organizations are shown in Figure 20.1. Several of the Indigenous governments and organizations have overlapping territorial or land claims. However, the land claims of Quebec Indigenous groups in Labrador have not been accepted for negotiation by the Government of Newfoundland and Labrador.

The Innu Nation of Labrador (“Innu Nation”), representing Innu in the communities of Natuashish and Sheshatshiu, claims Aboriginal rights and title to most of Labrador. Their land claim was accepted for negotiation by the Government of Canada and the Government of Newfoundland and Labrador, with formal negotiations beginning in 1991. Tacora Resources has successfully negotiated an Impact Benefits Agreement (“IBA”) with the Innu Nation on March 21, 2018. The IBA outlines guidelines around hiring, procurement, and royalty payments. Tacora has agreed that the Innu Nation will be given prior notification of hiring opportunities and certain procurement opportunities on which Innu Partnership companies can submit quotes and bids. Full details of the IBA are available on Tacora’s website.

A second group with asserted land claims to the Labrador west region is the Innu Takuaikan Uashat Mak Mani-Utenam (“Uashat or ITUM”). The ITUM live in two settlements within their reserve, Uashat and Maliotenam, both on the Quebec North Shore, near Sept-Iles. Although the ITUM’s land claim has not yet been accepted by the Newfoundland and Labrador government, Tacora has attempted to initiate negotiations with ITUM to identify areas of mutual benefit for both parties, however, no formal negotiations have been scheduled to date. Tacora continues to offer the opportunity to ITUM to come to the table to finalize an IBA.

The third group that claims historical land use rights within Labrador west is the Innu Nation of Matimekush-Lac John (“MLJ”). The people were traditionally members of the Uashat located adjacent to Sept-Iles, this band voluntarily moved to the Schefferville region in the early 1950s when the Quebec North Shore & Labrador (“QNS&L”) Railroad was completed. Tacora is in the process of negotiating a Memorandum of Understanding with MLJ to set a framework for ongoing discussions and possible IBA negotiation.

Figure 20.1: First Nation Reserve Locations



Also located in the Schefferville, QC area is the Naskapi Nation of Kawawachikamach ("Naskapi"). They were originally a small nomadic tribe, settling in Fort Chimo in the mid-1800s, before moving to Schefferville in the 1950s. The Naskapi relocated to the present site of Kawawachikamach, approximately 16 km north of Schefferville in the 1980s following the James Bay Settlement. To date Tacora has not entered any Memorandum of Understanding ("MOU") or IBA discussions with the Naskapi Nation.

The final Indigenous group that Tacora is working with is the NunatuKavut Community Council ("NCC"). Also identified as NunatuKavut, the members of this group self-identify as Southern Inuit, as per their 2010 land claim assertion document entitled Unveiling NunatuKavut. NCC states that its 6,000 members live in 23 Labrador communities, seventeen of which are on the southeast coast from Paradise River to L'Anse au Clair. It also states that members reside in six other communities in central and western Labrador, including Happy Valley-Goose Bay and Labrador City. NunatuKavut has filed a comprehensive land claim with the province of Newfoundland and Labrador as well as with the federal Government of Canada. Tacora has met with NCC's leadership and is beginning discussions about areas of mutual benefit.

20.7 Mine Closure

The RCP was last updated in 2017 and used the approach developed by the previous owner of Scully Mine, Cliffs, in their 2014 Plan. The 2017 RCP was adjusted to reflect the requirements in the draft 2010 Mining Act Guidelines, and meetings held with the Department of Natural Resources (now the Department of IET), Mineral Development Division. The 2017 RCP, approved in 2018, will be updated in 2023, per the requirement to submit an updated RCP every five years, or if there is a significant change to the Project. The 2023 RCP will also be adjusted to reflect the requirements in the 2020 Rehabilitation and Closure Plan Guidelines. The rehabilitative measures presented have been developed at a conceptual level for the purpose of preparing the RCP and to provide the basis for estimating the cost of closure.

20.7.1 Remediation and Reclamation

Closure of the site will consist of the following elements:

- Removal and appropriate disposal of all hazardous chemicals, reagents and materials from both the mine and surface facilities that could otherwise present a risk of future environmental harm.

- Demolition and removal of all above-grade buildings, foundations (to 300 mm below grade) and other infrastructure (e.g., overhead piping, electrical cables) no longer required once the mine has closed.

- Shipping and sale of salvageable material if prevailing salvage markets and scrap prices and associated economics permit; otherwise, these materials will be removed for approved disposal off site.

- Disposal of all non-salvageable, non-hazardous demolition debris into an approved on-site or near site waste disposal site.

- Cleanup of all surface yards including removal and appropriate disposal of all materials.

- Assessment of soil contamination in the area of the surface facilities and implementation of appropriate management measures (i.e., remediation or Human Health and Ecological Risk Assessment) to address contaminated soils identified.

- Removal of fencing, recontouring of roadways and restoration of natural drainage patterns wherever practical.

- Decommissioning of seven tunnels.

- Re-vegetation of the plant and non-flooded mine site footprints, where practical (as dictated by soil availability). If soil availability is limited, other means such as hydroseeding will be used.

- Progressive reclamation of inactive tailings areas involving re-vegetation of exposed tailings beaches, dike structures and all disturbed areas. Regrading of dike slopes where required to reduce erosion gullies and promote establishment of vegetation (particularly on the over-steepened bench slopes of the North and South Dikes). There will be a construction of a permanent engineered Diversion Channel with riprap erosion protection to ensure conveyance and stable pathway for the drainage of the Flora watershed passing along the extents of the tailings beach and discharge into North Flora Lake. Construction of engineered ditches along strategic locations of swales formed on the tailings beach to manage potential erosion from concentrated runoff flows.

- Natural flooding of the open pits to enhance environmental stability and re-establish natural drainage patterns.

- Berming of the open pits to reduce accessibility.

- Berming of the waste dump crests and vegetating the dump slopes that are visible from the towns of Wabush and Labrador City as well as Duley Lake.

- Monitoring activities and programs to evaluate site erosion, pit wall stability, surface water and groundwater quality, pit infilling, site access control and treated and revegetated areas, including implementation of required corrective measures to deal with environmental concerns that may arise in the post-closure time period.

20.7.2 <u>Estimated Reclamation Costs</u>

The cost to decommission and reclaim the open pits, waste rock dumps, ore handling and processing facilities, tailings management area and all of the related infrastructure associated with the Scully Mine site is estimated to be CAD 90,283,206. The cost estimate was determined by reviewing and updating the approved 2017 cost estimate, with some changes to reflect updated site quantities, additional research, recent contractor estimates, and updated cost estimates based on RS Means Costworks. Where possible costs based on current knowledge for activities were used. This estimate includes costs for an ongoing environmental monitoring program which will extend beyond closure. Additional long-term monitoring costs are included for maintenance and geotechnical inspections.

The following criteria and assumptions have been considered in completing the cost estimates:

- The cost estimate was derived largely from updating the cost estimate in the 2017 Rehabilitation and Closure Plan, which was based on updating the costs estimates completed in 2011 and 2014.

- An inflation rate of 3.6% was applied using the Canadian consumer price index for the Province of Newfoundland and Labrador.

- The volumes and quantities for buildings and associated infrastructure were obtained from a study carried out by Applied Technical Resources Inc. ("ATR Inc.") of Wheeling, West Virginia entitled "Asset Retirement Obligation Study" of Scully Mine. ATR Inc. conducted a survey of each building and provided estimated costs for asbestos removal, building and equipment demolition and removal and for the breakup and removal of all above and at grade concrete, with backfilling and cover. The estimates prepared by ATR Inc. were in USD and converted to CAD using the exchange rate on February 27, 2003 (USD 1 = CAD 1.39). The current estimates use the ATR Inc. estimates, corrected for inflation using the Canadian consumer price index.

- All costs are in Canadian funds for all labour, materials and equipment.

- All costs are in year 2021 dollars with no allowance made for escalation.

- The estimate assumes that all surface buildings and infrastructure will be demolished after the facilities have been cleared of process materials and after all potentially hazardous materials have been removed. Material and equipment will be disposed of at the on-site waste disposal facility or disposed of offsite. Only materials and equipment that have been decontaminated and/or have had all potentially hazardous products removed (i.e., removal of hydrocarbons, asbestos, insulation, reagents and chemicals, etc.) would be placed into the on-site landfill in accordance with the Waste Management Plan.

- The estimate assumes that experienced third-party contractors would be retained to demolish most of the surface buildings. The contractors would also be responsible for the removal and disposal of all potentially hazardous materials such as chemicals, reagents, glycol, lubricants, hydrocarbons, hydraulic fluids, refractory materials, instruments containing nuclear source material, PCB containing light ballasts, etc. The contractor will be tasked with the removal of any asbestos siding and/or insulation materials, demolition and removal of the buildings and their contents including the demolition and removal of all above and at grade concrete foundations (to 300 mm below grade). Mobilization and demobilization costs have been included in the cost estimate.

- Costs have been included for the clean out of all hydrocarbon storage tanks and for the demolition and disposal of these facilities following cleaning.

- Costs have been included for the removal from site of remaining chemicals, hydrocarbons, etc. that cannot be returned to the original suppliers.

- Costs have been included for the demolition and removal of all above ground utilities, such as power distribution lines and poles, pipelines and trestles and site rail lines.

- Reclamation cost estimates for the remaining tailings materials not already reclaimed have been based on actual experience developed by Scully Mine in the successful reclamation of tailings in the North Basin of Flora Lake.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Expenditures

It was decided to classify capital expenditures ("CAPEX") in two sections:

- Upgrade CAPEX: expenditures required to purchase equipment, repair, and upgrade the existing facilities and execute pre-production to reach a stable production of 6 Mt of concentrate per year.
- Sustaining CAPEX: expenditures required to maintain the production level of 6 Mt of concentrate per year through equipment replacement and major repairs or purchase of additional equipment and expansion of the Tailings facilities.

21.1.1 Upgrade Capital Expenditures

When evaluating the re-start of the Scully Mine in 2018, an initial upgrade capex of CAD 210.1 m was estimated inclusive of contingency, but exclusive of applicable taxes, escalation, risk and management reserves. At the end of 2020, upgrade capex totaled CAD 156M and the estimate-to-complete (ETC) was CAD 95M for an estimate-at-completion ("EAC") of CAD 251M for an overrun of CAD 41M or 19.5%. Some of packages were delayed and cancelled and new packages were added as the requirements changed and discoveries were made. The accuracy of the ETC is ± 15% given the methodology and level of confidence with respect to the project definition and it meets the requirement of AACE-Class III-Feasibility Study Estimate. Table 21.1 presents the CAPEX summary by Work Package.

Table 21.1: CAPEX Estimate Summary per EWP

EWP # and Description	FEAS 2018 Total Costs (k CAD)	Closed Capital Cost (k CAD)	ETC (k CAD)
01 - Dryer Re-Built, #2	3,437	3,278	0
02 - Mill Gear Replacement, #6	1,021	781	0
03 - 1B Belt Magnet	313	229	0
04 - Fresh Water Header	218	35	0
05 - Mill Feed Chute	879	863	0
06 - Mill screen oversize and Spiral Feed Pump	389	-	0
07 - Manganese Reduction	8,360	4,529	0
08 - Primary Crusher Rock Hammer	1,995	1,499	0
10 - Added Loadout Bin Capacity	14,707	15,963	981
11 - Plant Structural Repairs	405	2	0
12 - Dryer PLC Update	1,212	3,292	0
13 - Electrical Refurbishment	4,757	4,778	0
14 - New Instrumentation	298	511	0
15 - Rod Deck Conversion	630	181	0
16 - Plant PLC Upgrade	2,020	3,181	0
17 - Railcars and Tripack	4,030	2,487	0
18 - Rail Track	287	53	0
19 - Mining	105,732	80,366	0
2021 Updated Feasibility			
21 - MCC Upgrade			2,595
22 - Mill Lubrication Project			1,262
23 - Mill Gear Replacement on L1 &L4			2,538
24 - Mill Recirc Line Project			1,137
25 -1B/2B Belt Replacement			429
26 - Scavenger Spiral Expansion			1,865
27 - Modernize Instrumentation and Control			851
28 - Tails Line Improvement			3,872
29 – Tailings Earthwork			385
30 - Mining Equipment			17,570
31 – Commissioning Bunker C			670
33 – 2022 Mill VFD, Gearing, other Upgrades			15,000
34 - 2022 SFPPN Port Upgrades			15,880
Subtotal: Direct Costs	**150,690**	**122,029**	**65,035**
Indirects (excluding escalation and risk)	**59,382**	**33,972**	**19,126**
Contingency			**10,867**
Total	**210,072**	**156,001**	**95,028**

Note: Numbers may not add due to rounding, Currency for CLOSED CAPEX is 1.25 USD TO CAD

The ETC for the remaining Upgrade CAPEX includes all of the packages direct and indirect costs with the associated contingency of 5% for the mobile equipment and 15% for the other categories.

Table 21.2 presents a division of responsibility per package. Although MTO's and budgeted expenditures were not entirely developed by G Mining Services ("GMS"), thorough analyses of MTO's developed by others and budgetary proposals obtained by Vendors, Contractors and Tacora Resources Inc. ("Tacora") were reviewed and finalized.

Table 21.2: Division of Responsibility

EWP # and Description	Responsibility		
	Scope of Work / Engineering	MTO development	CAPEX Estimate
10 - Added Loadout Bin Capacity	Tacora	Tacora	Tacora / GMS
20 - Plant Mill Maintenance Tactics	Tacora	Tacora	Tacora / GMS
21 - Mill Main Motor / MCC 5KV Upgrade	Tacora	Tacora	Tacora / GMS
22 - Mill Lubrication Project	Tacora	Tacora	Tacora / GMS
23 - Mill Gear Replacement on L1 &L4	Tacora	Tacora	Tacora / GMS
24 - Mill Recirc Line Project	Tacora	Tacora	Tacora / GMS
25 -1b/2b Belt Replacement	Tacora	Tacora	Tacora / GMS
26 - Scavenger Spiral Expansion	Tacora	Tacora	Tacora / GMS
27 - Modernize Instrumentation	Tacora	Tacora	Tacora / GMS
28 -Tails Line Improvement	Tacora / Hatch	Tacora / Hatch	Tacora / Hatch
29 - Tailings Earthworks	Hatch	Hatch	Tacora / Hatch
30 - Mining Equipment	GMS	GMS	GMS
31 - Commissioning Bunker C	Tacora	Tacora	Tacora / GMS
32 - Progressive Rehabilitation	SEM	SEM	SEM / Tacora
33 - 2022 Mill VFD, Gearing, other Upgrades	Tacora	Tacora	Tacora / GMS
34 - 2022 SFPPN Port Upgrades	Tacora	Tacora	Tacora / GMS
900 – INDIRECTS	Tacora / GMS	Tacora / GMS	Tacora / GMS

21.1.2 Packages Description

Only the remaining packages are discussed at this time.

03 - 1B Belt Magnet

The belt magnet was installed (mechanical scope completed Sept 2020). The electrical scope and commissioning are still outstanding. The E&I commissioning is not covered in the 2021 CAPEX budget as Tacora does not require significant funding for the remaining work.

08 - Primary Crusher Rock Hammer

Two rock breakers were procured. The first one was installed (May 2019). The other is in the warehouse and foundation has been engineered. The installation is not covered in the 2021 CAPEX budget as Tacora does not require significant funding to complete the package.

10 - Added loadout bin capacity

This project was partially completed. The silos were installed. The remaining work involves the completion of the conveyor catwalk and the tie-in to the current tripper system. Funding is allocated in the 2021 CAPEX.

20 - Plant Mill Maintenance Tactics

This area is focused on the efficiency of the mill planned maintenance and scheduling. The two primary areas of improvement are to reduce time required for Mill Preventive Maintenance (“PM”) and minimize crude end outages.

In 2021, a strategic relationship was built with a new vendor with a commitment to execute mill PMs in 6 days starting in April. Tacora does not request additional CAPEX to achieve those PM tactics.

21 - Mill Main Motors and MCC upgrade

In the 2021, the scope of work for the mill motors is the replacement of 5 KV starters and relays and optimization of the motor installation procedure.

The two 600 V MCC's in the boiler room along with the four (4) associated main breakers for emergency power transfer are original (1960's) equipment and need to be replaced and automated. The boiler room is essential, especially during the winter months and it cannot be without power for longer than 15-30 minutes before systems start to freeze.

22 - Mill Lubrication Project

The poor reliability of the current lube systems and the additional impact on the bearings and gearing performance lead to downtime. The main issues with the lube systems are the lack of instrumentation, poor bath design, and the lack of filter which cause dirty oil, leaks, and frequently empty baths.

The first new lube system was installed in January and Tacora will continue to replace the obsolete lube systems. The newly designed lube system is simple and effective; it utilizes the gearing reservoir, properly engineered instrumentation and controls to ensure better uptime and reliable lubrication for the gears.

23 - Mill Gear Replacement on L1 and L4

Excessive gear vibration on Lines 1 and 4 shows that there is poor bull gear, intermediate gear, and bearing alignment on these mills. This has caused a reduction in throughput on these mills in order to minimize vibration and avoid catastrophic failure, but it has also caused increased downtime to repair and temporary fix the gearing problem.

In March, the intermediate gear, high speed pinion and low speed pinion are being replaced, new pinion bearings being installed, new trunnion bearing being installed, trunnion sole plate foundation being repaired and new sole plates being installed on Mill 1.

Similarly, Mill 4 is getting trunnion foundation repaired and an entire new gear assembly installed. With these gear improvements and the new lubrications system, a significant decrease in downtime associated with gearing should be experienced and the imposed throughput restriction removed.

24 - Mill Recirc Line Project

The mill recirculation lines are bringing the mill screen oversize back into the mill to achieve further grinding. This is achieved with a pump and 8" line that re-enters the mill through the discharge end. Due to the coarse nature of the material and the size of the pipe, a lot of water is required to keep the material flowing. Additionally, this causes a lot of wear and tear on the pump which needs to be rebuilt every 1,400 hours.

In January, the recirculation pump on Mill 4 was replaced with a more robust test pump and operating at slower speed to reduce wear. This mill is also in the process of being fitted with a smaller re-entry line so that we can pump at the same velocity but reduce the water in the mill.

The scope of this work is to supply materials and equipment to perform the required demolition and installation of six (6) new slurry Warman pumps (Weir MCU150).

25 - 1B/2B Belt Replacement

The 1B and 2B conveyor systems are non-redundant and are critical pieces of equipment to keep all mills fed. Significant internal belt damage to both the 1B & 2B conveyors has been observed. New belts are on order and are scheduled for installation in late June. Follow up with preventive maintenance practices and inspections will significantly eliminate unplanned downtime.

26 - Scavenger Spiral Expansion

The gravity circuit has a very basic flowsheet with no scavenging circuit. Fine material is recirculated and mixed with coarse material in the existing gravity circuit. The separation efficiency is decreased when processing both streams together.

In order to improve recoveries, testing was completed to show the benefits of processing the fine ore through a scavenger circuit of dedicated spirals.

The scope of this work is to supply materials and equipment, to perform required demolition and installation of scavenger spirals.

27 - Modernize Instrumentation

New Weightometers

The scope of work is to purchase and install 10 new weightometers to improve the accuracy of the process controls and mass balancing across the site.

Spiral Feed Flowmeters

When the plant restarted, new density transmitters were supplied to control the feed to the spirals; however it has been determined that the old ultrasonic flowmeters are unreliable and inaccurate. Within the new process control scheme, it is important to understand both flow and density to ensure good performance in the spirals. The scope of work is to purchase and install 12 flowmeters at the spirals feed.

Mill Feed Flowmeters/Control Valves

Currently there is no flow measurement or control on the water added to the grinding mills feed. Adding this control will avoid plugging the mill feed chute if insufficient water is supplied or having an excess of dilution and poor grinding if too much water is added. With a reduction in water and increase in mill density, an increase in gravity circuit recovery is expected.

Six flow meters and six control valves will be installed at the mill feed.

28 - Tails Line Improvement

The scope of work is to purchase and install 5.5 km of pipeline to reach the expanded tailing facility.

29 - Tailings Earthwork

The scope of work is to manage the earthwork for dike extension at the tailings impoundment.

30 - Mining Equipment

The CAPEX is to achieve the desired mining capacity to extract ore and waste tonnages specified in the mine plan. The purchase of three additional 240 t Haul Truck will be required, bringing the number of trucks to a total of 13 units.

31 - Commissioning Bunker C

Currently there is one functional Bunker C tank. A second tank will enable operations to continue when the first tank is unavailable.

33 - 2022 Mill VFD, Gearing, Other Upgrades

The scope of work is to continue to increase availability and throughput with important maintenance of the variable speed drive and gear replacement on the mills over 2022.

34 - 2022 SFPPN Port Upgrades

The scope of this work is to complete upgrades of the concentrate handling systems at SSPPN *(Société ferroviaire et portuaire de Pointe-Noire)* in 2022.

21.1.3 CAPEX Estimate

The CAPEX estimate was coordinated by GMS with input from Tacora, Hatch and SME.

The following sources of information and data were used for the CAPEX estimate development.

Labour Costs

Most of the Labour Costs were based on Contractor quotation. Labour rates inclusive of the following:

- Base salaries
- Fringe benefits
- All applicable social charges, fees, contributions & premiums
- Personal Protective Equipment (PPE)
- Union delegates
- Workers rotational transportation, as appropriate
- Room and board, as appropriate
- Workers and staff local transportation
- Construction equipment costs c/w rental, operations and maintenance
- Tools & consumables c/w freight & logistics costs
- Site direct supervision
- Sub-contractors' overhead & profit.

Equipment and Material Costs

Most of the pricing received by Tacora were firm price from suppliers and included engineering and vendor representatives.

PM, E, P and CM Services

The cost estimate for engineering services was developed by Tacora and included the following:

- Salaries, fringes, uplifts, recruitment, overhead, etc.
- Expenses (i.e. business and rotational travelling, including in-transit costs, etc.).
- IT services.

The cost estimate for project management, procurement and construction management services are included with Tacora Owner's costs.

Site Indirect Costs

Site indirects are included with Tacora Owner's costs; they consist of the following:

- Power distribution through tie-ins to the existing reticulation.
- Construction water through the existing plant water distribution.
- Existing offices on site including office supply and IT equipment.
- Communications.
- Fuel.
- Existing infrastructures for the management of sewage, construction waste and garbage.
- Maintenance of existing access and in-plant roads, parking and walkways.
- Access control and monitoring.
- Existing lay down and storage areas, as well as warehousing, complete with materials management and materials handling equipment.
- Site security.
- Existing light vehicles.
- Existing first aid and medical installations.
- Site Surveying.
- NDE and QA/QC testing, including laboratory services.
- General and final clean-up.

Owner's Costs

Owner's costs are included with Tacora OPEX budget.

Contingency

The Project contingency was evaluated using an individual contingency per estimate item.

21.2 Sustaining Capital Expenditures

The Sustaining CAPEX is grouped in three large sectors: Mine, Plant, Tailings. The CAPEX estimates per sector are shown in Table 1.8 for a total of CAD 618,231,844.

Table 21.3: Sustaining Capex Estimates

	CAD (000)
Mine	
Addition + Replacement	214,883
Major Repairs	224,810
Plant (Conc. + Infrastructures)	72,000
Tailings	
Earthwork	32,748
Pump & Pipelines	31,385
Contingency	42,405
Total	**618,232**

21.2.1 Mine

Equipment requirement was estimated to achieve the life of mine ("LOM") production schedule based on achievable equipment productivities comparable to similar operations. From this equipment requirement, a purchase schedule was established along with a replacement schedule based on expected equipment life. The cost of equipment is based on quotes obtained from vendors. The LOM schedule of equipment purchase is listed in Table 21.4 and the corresponding investment of CAD 214,883,000 is detailed in Table 21.5. Similarly, a useful life and a repair and replacement schedule was estimated for the major components of large mine equipment and the corresponding expenditures CAD 224,810,000 are added to Table 21.5.

It is calculated that the residual book value of the Major Mine Equipment at the end of the mine life would be CAD 72,853,000. This value would likely be realized only if additional reserves are found and the mine life is extended. If this equipment had to be sold in the used equipment market, it is estimated that the net realized value would be less than 25% of the remaining book value. Due to the uncertainty of the mine equipment salvage value and its relatively small NPV in 27 years, its contribution was excluded from the project economics.

Table 21.4: Equipment Purchase Schedule

Equipment Requirement	Total	2022	2023-2032	2033-2038	2039-2041	2042-2044	2045-2046
Major Equipment							
Electric Prod Drill	3	-	1	1	-	1	-
Electric Hydraulic Shovel (24 m³)	-	-	-	-	-	-	-
Diesel Hydraulic Shovel (24 m³)	1	-	-	1	-	-	-
Diesel Hydraulic Shovel (22 m³)	1	-	-	-	-	1	-
Wheel Loader (20 m³)	-	-	-	-	-	-	-
Wheel Loader (13.5 m³)	-	-	-	-	-	-	-
Mining Truck (240 t)	15	-	1	14	-	-	-
Mining Truck (100 t)	-	-	-	-	-	-	-
Track Dozer (600 HP)	6	-	2	2	-	2	-
Track Dozer (350 HP)	8	-	4	2	-	2	-
Motor Grader 16 ft and 18 ft	8	-	4	2	-	1	1
Wheel Dozer 500 HP	1	-	1	-	-	-	-
Water/Sand Truck	4	1	1	1	-	1	-
Support Equipment							
Excavator (49 t)	2	-	1	-	-	1	-
Rock Breaker	5	-	2	1	1	-	1
Wheel Loader (4.7 m³)	1	-	-	1	-	-	-
Wheel Loader 425 HP	9	-	4	3	1	1	-
Wheel Loader 165 HP	3	-	1	1	-	1	-
Skid Steer Loader	10	-	5	2	2	-	1
Boom Truck 28 t	2	-	1	1	-	-	-
Forklift Diesel 4 t	1	-	-	1	-	-	-
Manlift 60 ft	3	-	1	1	1	-	-
Cable Reel Wheel Loader	1	-	1	-	-	-	-
Mechanic Service Truck	3	-	1	1	-	1	-
Welding Truck	4	-	2	1	-	1	-
Tire Handler Forklift	1	-	1	-	-	-	-
Fuel & Lube truck (7.5 kL) ADT	3	-	1	1	-	1	-
Tow Haul Truck	-	-	-	-	-	-	-
Trailer Lowboy 100 t	-	-	-	-	-	-	-

Equipment Requirement	Total	2022	2023-2032	2033-2038	2039-2041	2042-2044	2045-2046
Pick-up Truck	20	-	10	5	5	-	-
Diesel Powered Air Heaters	4	-	2	2	-	-	-
Dewatering							
Long Hole Dewatering	-	-	-	-	-	-	-
In-Pit Dewatering	5	-	5	-	-	-	-

Table 21.5: Equipment Sustaining Capital

Equipment Requirement	Total	2022	2023-2032	2033-2038	2039-2041	2042-2044	2045-2046
Major Equipment							
Electric Prod Drill	25.1	-	8.4	8.4	-	8.4	-
Electric Hydraulic Shovel (24 m^3)	-	-	-	-	-	-	-
Diesel Hydraulic Shovel (24 m^3)	14.1	-	-	14.1	-	-	-
Diesel Hydraulic Shovel (22 m^3)	9.8	-	-	-	-	9.8	-
Wheel Loader (20 m^3)	-	-	-	-	-	-	-
Wheel Loader (13.5 m^3)	-	-	-	-	-	-	-
Mining Truck (240 t)	95.6	-	6.4	89.2	-	-	-
Mining Truck (100 t)	-	-	-	-	-	-	-
Track Dozer (600 HP)	11.0	-	3.7	3.7	-	3.7	-
Track Dozer (350 HP)	7.1	-	3.5	1.8	-	1.8	-
Motor Grader 16 ft and 18 ft	15.2	-	7.6	3.8	-	1.9	1.9
Wheel Dozer 500 HP	6.6	-	6.6	-	-	-	-
Water/Sand Truck	7.6	1.9	1.9	1.9	-	1.9	-
Support Equipment							
Excavator (49 t)	1.2	-	0.6	-	-	0.6	-
Rock Breaker	1.1	-	0.4	0.2	0.2	-	0.2
Wheel Loader (4.7 m^3)	1.7	-	-	1.7	-	-	-
Wheel Loader 425 HP	6.4	-	2.9	2.1	0.7	0.7	-
Wheel Loader 165 HP	1.4	-	0.5	0.5	-	0.5	-
Skid Steer Loader	0.9	-	0.5	0.2	0.2	-	0.1
Boom Truck 28 t	0.7	-	0.4	0.4	-	-	-
Forklift Diesel 4 t	0.1	-	-	0.1	-	-	-
Manlift 60 ft	0.4	-	0.1	0.1	0.1	-	-
Cable Reel Wheel Loader	1.6	-	1.6	-	-	-	-
Mechanic Service Truck	1.0	-	0.3	0.3	-	0.3	-
Welding Truck	0.6	-	0.3	0.2	-	0.2	-
Tire Handler Forklift	0.2	-	0.2	-	-	-	-
Fuel & Lube truck (7.5 kL) ADT	3.5	-	1.2	1.2	-	1.2	-
Tow Haul Truck	-	-	-	-	-	-	-

Equipment Requirement	Total	2022	2023-2032	2033-2038	2039-2041	2042-2044	2045-2046
Trailer Lowboy 100 t	-	-	-	-	-	-	-
Pick-up Truck	1.0	-	0.5	0.2	0.2	-	-
Diesel Powered Air Heaters	0.04	-	0.0	0.0	-	-	-
Dewatering							
Long Hole Dewatering	-	-	-	-	-	-	-
In-Pit Dewatering	1.0	-	0.5	-	-	-	-
Total							
Sub-Total Equipment Purchase	214.9	2.5	47.9	130.0	1.5	30.8	2.2
Sub Total Major Comp.Repairs	224.8	9.3	96.8	48.7	32.5	20.9	16.6
Total Mine Equipment Sustaining Capital	439.7	11.8	144.7	178.7	34.0	51.7	18.8

21.2.2 Plant

The Plant sector includes mostly the concentrator but also all other infrastructures of the Scully Mine. No detailed estimate for the Plant was available and discussion with Tacora personnel resulted in an allocation of CAD 3,450,000 (including 15% contingency) per year from 2022 to 2045 assuming that no investment would be made in the last two years of operations.

21.2.3 Tailings

Section 18.1.1 discusses the tailings storage requirements during operations. Hatch detailed the nine deposition phases designed to meet the needed volume. Typically, each phase will initially require earthwork to raise and extend dykes after preparing foundations and tree clearing. Then pipelines and spigot lines will be added to reach the planned discharge locations. As discharge distance and elevation increase, additional pumps and booster stations will be added as needed.

The estimated expenditures for additional tailings disposal volume are summarized as follows:

Table 21.6: Estimated Expenditures for Additional Tailings Disposal Volume Summary

	Direct (CAD 000)	Indirects (CAD 000)	Contingency (CAD 000)	Total
Tailings – Earthwork	30,141	2,607	4,912	37,660
Tailings – Pipelines	25,413	5,972	4,708	36,093

21.3 Scully Operating Costs

The operating costs at the Scully Mine were included in three areas:

- Mining
- Processing
- General Services and Administration.

The operating costs for the Project have been estimated at a Feasibility Study level with an accuracy of ±15%. Operating costs are based on year-end 2020 prices and wages and include procurement and logistics costs. No contingency was added to operating costs.

21.3.1 Mining Operating Costs

Mining expenses were estimated by GMS from its data bank and specific consumables costs at Scully. All explosives and diesel costs are charged to mining operations.

The mine operating costs are estimated from first principles for all mine activities. Equipment hours required to meet the LOM plan are based on productivity factors or equipment simulations. The delivered fuel price to site used in estimating mining costs is CAD 0.88/L and the consumption is calculated at 0.63 l/t mined. The mine wage scale established for the Project has operators earning between CAD 48/h and CAD 57/h, including benefits and bonus allowances. The major equipment hourly operating costs are presented in Table 21.7 exclusive of operating labour and major components repairs.

Table 21.7: Major Equipment Hourly Operating Cost

Machines Hourly Cost (CAD/h)	Maint. Labour	Fuel / Electricity	Parts & Repairs	Lube	GET	Tires	Total	Major Components
Electric Production Drill	43.85	13.15	68.58	5.50	10.45	-	**141.53**	48.45
Electric Hydraulic Shovel (24 m³)	40.93	39.29	88.00	3.63	150.00	-	**321.85**	236.46
Diesel Hydraulic Shovel (24 m³)	29.23	257.15	95.46	8.76	150.00	-	**540.60**	228.64
Diesel Hydraulic Shovel (22 m³)	24.56	191.33	85.42	6.73	45.00	-	**353.04**	176.07
Wheel Loader (21 m³)	51.45	149.58	84.28	17.69	160.28	63.20	**526.48**	73.95
Wheel Loader (13.5 m³)	29.23	72.89	80.00	7.32	32.00	37.60	**259.04**	21.09
Mining Truck (240 t) K	14.03	194.41	37.06	4.62	18.00	52.50	**320.62**	59.81
Mining Truck (100 t) K	11.69	74.71	25.41	2.60	4.00	24.00	**142.41**	18.92
Track Dozer (899 HP)	13.45	66.96	28.63	1.69	28.00	-	**138.73**	61.15
Track Dozer (600 HP)	11.69	31.54	22.00	1.50	19.00	-	**85.73**	30.10
Motor Grader 18ft	22.22	22.16	25.23	2.40	12.70	9.12	**93.82**	11.81
Water/Sand Truck	21.63	182.22	45.00	12.00	8.00	42.00	**310.85**	25.03
Wheel Dozer 500HP	14.03	40.17	31.20	5.90	19.87	20.00	**131.17**	25.03

For the major mining equipment, the parts and repair costs are based on a life cycle costing strategy.

The average mining cost during operations is estimated at CAD 2.70/t mined, including re-handling costs. This operating cost estimate excludes capital repairs, which are treated as sustaining capital. If considering the major components repairs, the mining cost would be CAD 2.96/t mined.

Table 21.8 details the mine operating costs by activity. Haulage is the major mining cost activity, representing 32,5% of total costs, followed by blasting (14.3%), loading (7.4%) and road maintenance (6.9%). Loading and haulage for stockpile re-handling is also captured as a separate activity.

Salaries and diesel fuel are the dominant cost by element representing respectively 25.1% and 19.7% each of total costs, followed by maintenance parts (13.2%) and bulk explosives (9.1%).

Table 21.8: Mine Operating Cost Summary by Activity

Activity	Total Cost (M CAD)	Cost (CAD/t mined)	% of OPEX
Mine Operations Admin	58.5	0.07	2%
Mine Maintenance Admin.	107.7	0.12	5%
Mine Geology & Engineering	42.4	0.05	2%
Grade Control	128.1	0.15	5%
Drilling	91.1	0.10	4%
Blasting	334.2	0.38	14%
Loading	186.2	0.21	8%
Hauling	758.6	0.87	32%
Dump Maintenance	131.1	0.15	6%
Road Maintenance	165.1	0.19	7%
Dewatering	33.5	0.04	1%
Overburden Mining Contract	111.7	0.13	5%
Support Equipment	113.1	0.13	5%
Electrical Cable Handling	25.6	0.03	1%
Rehandling	58.3	0.07	2%
Total Mining Cost	**2,345.1**	**2.70**	**100%**

21.3.2 Processing Operating Costs

Processing costs of the Scully concentrator depend to a large extent on the specific flowsheet developed for the Scully ore and the high quality targeted for the final concentrate. Description of the flowsheet and corresponding mass balance can be reviewed in Section 17. When comparing the operating costs of the Scully concentrator to other concentrators, Scully benefits from the following:

- Coarser grind.
- Low crushing and grinding indices.
- Very low cost of power.
- No reagent required.

The current concentrate capacity is estimated at 385 tph per grinding line with an availability of 89.5% for an ore feed capacity of 18.1 Mtpy. It is expected that the concentrator capacity could approach 19 Mtpy once optimalization is completed, the equipment upgrade finalized, and process control fully implemented.

Processing cost were based on the following parameters proposed by Tacora and reviewed by GMS for reasonableness:

- Management and supervision costs.
- Operating and maintenance labor rates.
- Electrical power consumption and costs.
- Bunker C consumption and costs for boilers and dryers.
- Contract maintenance during shutdowns.
- Laboratory operations.
- External consultants.

The costliest consumable is the Bunker C for the boilers and dryers which require 4.442 l/t of concentrate. At current price of CAD 0.64/l, it represents an expenditure of CAD 17M/y. All site electrical power is included in the processing costs. At full capacity, demand is 50 MW and the demand monthly charge is CAD 1.68/kw adding up to CAD 1M/y. Overall power consumption is 25.44 kw-hr/t of ore processed at CAD 0.026/kw-h for CAD 0.66/t of ore.

Processing costs for an average year are estimated as follows:

Table 21.9: Processing Costs per Year

Fixed Costs	CAD (000)	CAD/t Processed	CAD/t Concentrate
Tacora Labour	24,353	1.39	4.06
Contractors	7,200	0.41	1.20
Vehicles	1.523	0.09	0.25
Variable Costs	**CAD (000)**	**CAD/t Processed**	**CAD/t Concentrate**
Power		0.66	1.93
Bunker C		0.97	3.02
Parts & Equipment		1.19	3.47
Total		**4.71**	**13.93**

The total unit costs will vary from year-to-year depending on the tonnages processed to deliver the 6 Mt of concentrate per year.

21.3.3 General Services & Administration Operating Costs

General Services and Administration (G&A) costs include the following activities:

- Senior Management
- Insurance
- Accounting & Management Systems;
- Security
- Health & Safety
- Environment
- Human Resources & Training
- IT and Communications
- Housing and Camps
- Offices and Building Maintenance.

In Tacora's situation, some management time is required to plan and supervise activities performed by others such as rail transportation and port storage and ship loading, marketing and ocean freight.

Based on our data bank of comparable mining operations, the G&A cost estimates were increased to CAD 15,500,000/y or 9.0% of direct costs. This would be best-in-class considering the size of the operation, its relative remoteness, and the increasing pressure on the ESG front to add management, organization, standards and third-party audits and allocate more funds to local communities.

Corporate costs were not included in the G&A costs.

22 ECONOMIC ANALYSES

The economic assessment of the Scully Mine has a start time of January 1st, 2021 and is based on price projections in the U.S. currency and cost estimates in the Canadian currency. An exchange rate of CAD 1.25 per USD was assumed to convert particular components of the cost estimates and USD market price projections into CAD. No provision was made for the effects of inflation: all costs are based on year-end 2020 (constant dollars).

The evaluation was carried out on an all-equity investment basis and assumed no initial debt and financial leverage. It must be noted that commercial production was declared in 2019 and most of the planned investment to repair and upgrade the facilities, mainly the process plants were completed by year-end 2020. The remaining expenditures in 2021-2022 are for items added subsequently to the initial 2018 Feasibility Study. The current reserves are sufficient for the next 27 years (2021-2047).

Table 1.11 presents all revenues, transportation and port charges, royalties, OPEX and CAPEX, working capital requirements and closure cost on an annual basis. These are all input data required to determine annual cash flow streams, as discussed in previous sections.

22.1 Revenues

As explained in Subsection 19.7, spot pricing for iron ore has been widely adopted by market players with reference to the Platts 62% Fe Fines-CFR North China basis. The North China destination is justified by China being the world largest steel producer (54% of global production) and the world largest consumer (56%) and importer of iron ore. The 62% Iron Fines spot price is adjusted to take into account iron content, chemical composition / impurities and granulometry. A high iron content feed is typically preferred by steelmakers as higher iron content reduces the effective transport costs on a Fe unit basis and increases the iron content yield of the sinter. The importance of iron content in the concentrate has resulted in an additional iron ore reference price, the Platts 65% Fe Fines-CFR North China. A general widening of the 62% Fe fines and 65% Fe fines pricing spread has been observed since mid-2016; it appears to be driven by higher demand for ferrous products with high-value-in-use, such as high grade and pellets to reduce coking coal consumption.

Table 1.12 presents the calculations required to estimate the net revenue FOB Scully Mine from the assumed reference price of 62% Fe Fines CFR Northern China. The selected reference price is discussed in Subsection 19.7.3. The current premium to take into account the high iron content of the Scully concentrate is in the order of USD 27/t of concentrate in the prevailing market conditions. It is expected that the premium will come down to USD 17/t of concentrate once the steel and iron ore markets return to historical growth in demand. Cargill is responsible for the marketing of the Scully concentrates and is remunerated from a portion of the concentrate premium; their fees are currently USD 7.50/t and would reduce to USD 5/t under normal market conditions.

Table 22.1: Annual Cash Flow

Annual Cash Flow

Tacora Scully Mine

				2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Period number																	
Operations	01-Jan-21	30-Sep-47															
Closure	01-Oct-47	30-Sep-48		-	-	-	-	-	-	-	-	-	-	-	-	-	-
Production Highlights																	
Tonnage Mined	kt		868,807	28,228	32,849	32,000	32,000	32,381	32,000	32,000	32,000	32,000	32,000	32,000	32,000	32,000	32,000
Tonnage Processed	kt		478,943	15,588	18,084	17,573	18,328	18,530	18,239	18,715	19,292	18,236	18,043	18,334	18,802	18,475	18,232
Iron concentrate - dry	k dmt		159,100	5,124	6,000	6,009	6,000	6,002	6,000	5,999	5,987	6,000	5,999	5,995	5,946	5,980	5,948
Cash flow waterfall (k CAD)		CAD/t of conc.															
Income																	
Revenue (FOB Pointe Noire) (after 1 % deduction for losses)		87.10	13,856,858	928,541	840,807	642,621	506,533	495,599	482,105	482,007	481,056	482,105	482,024	481,714	477,742	480,527	477,924
Royalties MFC		-5.75	(915,517)	(63,405)	(57,039)	(43,024)	(33,403)	(32,630)	(31,676)	(31,670)	(31,604)	(31,676)	(31,671)	(31,650)	(31,378)	(31,568)	(31,390)
Other Royalties		-1.18	(188,422)	(6,068)	(7,106)	(7,116)	(7,106)	(7,109)	(7,106)	(7,104)	(7,090)	(7,106)	(7,105)	(7,100)	(7,041)	(7,083)	(7,044)
Total		80.16	12,752,920	859,068	776,662	592,480	466,024	455,860	443,323	443,233	442,361	443,323	443,249	442,965	439,322	441,877	439,490
Operational Expenditure																	
Mining	100%	-14.74	(2,345,076)	(88,745)	(89,882)	(86,750)	(87,869)	(86,481)	(86,226)	(87,509)	(94,078)	(96,421)	(98,435)	(88,892)	(87,821)	(84,040)	(84,415)
Processing	100%	-14.31	(2,276,398)	(103,567)	(93,215)	(82,979)	(86,331)	(86,468)	(85,113)	(87,333)	(90,027)	(85,097)	(84,199)	(85,555)	(87,739)	(86,210)	(85,076)
General & Administration	100%	-2.61	(415,493)	(16,368)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)
Concentrate Rail Transport and Port		-20.27	(3,224,736)	(145,947)	(156,214)	(131,772)	(119,823)	(118,885)	(117,689)	(117,672)	(117,503)	(117,689)	(117,675)	(117,620)	(116,914)	(117,409)	(116,946)
Total Opex	100%	-51.93	(8,261,704)	(354,627)	(354,811)	(317,001)	(309,522)	(307,334)	(304,529)	(308,014)	(317,107)	(314,707)	(315,809)	(307,566)	(307,974)	(303,159)	(301,938)
Operating Cost (CAD/t ore)			17.25	22.75	19.62	18.04	16.89	16.59	16.70	16.46	16.44	17.26	17.50	16.78	16.38	16.41	16.56
Working capital adjustments (during ops.)		-0.25	(40,289)	-41,461	-16,716	-3,231	-688	1,343	175	218	-925	283	738	-	-1,883	-103	1,001
Operating cash flow		27.98	4,450,927	462,980	405,135	272,248	155,815	149,868	138,969	135,437	124,329	128,899	128,178	135,398	129,466	138,615	138,553
Upgrade Capital Costs																	
Upgrade Capital Costs			(84,161)	(31,780)	(52,381)	-	-	-	-	-	-	-	-	-	-	-	-
Upgrade Capital Costs-Contingency			(10,867)	(3,916)	(6,951)	-	-	-	-	-	-	-	-	-	-	-	-
Pre-production revenue			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sub-total Upgrade Capital Costs	100%		(95,028)	(35,696)	(59,332)	-	-	-	-	-	-	-	-	-	-	-	-
Total upgrade capital cost			(95,028)	(35,696)	(59,332)	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital Costs																	
Mining Equipment and Major Repairs			(439,694)	-	(11,772)	(5,981)	(16,472)	(19,510)	(11,346)	(7,648)	(18,302)	(15,862)	(13,286)	(18,920)	(17,387)	(13,776)	(18,441)
Concentrator and Infrastructure			(72,000)	-	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)
Tailings Earthworks			(32,748)	-	(228)	(731)	(6,321)	-	-	(1,183)	(1,843)	(9,267)	-	-	(1,115)	(1,225)	(2,925)
Tails Pipeline			(31,385)	-	(1,107)	-	(6,726)	(1,081)	(1,081)	(1,081)	(1,081)	(1,081)	(6,726)	(648)	(648)	(648)	(648)
Grand Total Contingency			(42,405)	-	(1,239)	(859)	(3,231)	(1,588)	(1,179)	(1,172)	(1,804)	(2,795)	(2,123)	(1,493)	(1,584)	(1,420)	(1,908)
Total sustaining capital			(618,232)	-	(17,345)	(10,570)	(35,749)	(25,178)	(16,606)	(14,083)	(26,029)	(32,005)	(25,135)	(24,062)	(23,735)	(20,070)	(26,923)
Closure & Rehabilitation Costs																	
Closure & reclamation expenditures			(90,283)	(432)	(648)	(2,913)	(347)	(5,597)	-	(8,864)	-	(1,021)	(2,045)	-	-	-	(1,776)
Closure & reclamation cash flows			(90,283)	(432)	(648)	(2,913)	(347)	(5,597)	-	(8,864)	-	(1,021)	(2,045)	-	-	-	(1,776)
Cash flow Before Tax			3,647,384	426,852	327,810	258,765	119,719	119,094	122,364	112,489	98,300	95,873	100,997	111,337	105,731	118,545	109,855
Tax																	
Provincial income tax			(517,682)	(46,976)	(57,440)	(35,255)	(18,244)	(16,328)	(16,122)	(14,778)	(14,832)	(15,015)	(14,579)	(16,152)	(15,628)	(16,874)	(16,481)
Federal income tax			(517,682)	(46,976)	(57,440)	(35,255)	(18,244)	(16,328)	(16,122)	(14,778)	(14,832)	(15,015)	(14,579)	(16,152)	(15,628)	(16,874)	(16,481)
Total			(1,035,363)	(93,952)	(114,880)	(70,511)	(36,488)	(32,656)	(32,244)	(29,557)	(29,664)	(30,030)	(29,157)	(32,303)	(31,256)	(33,748)	(32,963)
Net Cash Flow			2,612,021	332,899	212,930	188,254	83,231	86,437	90,120	82,932	68,636	65,844	71,840	79,033	74,475	84,797	76,892

Annual Cash Flow

Tacora Scully Mine

				2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048
Period number				2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048
Operations		01-Jan-21	30-Sep-47														-
Closure		01-Oct-47	30-Sep-48	-	-	-	-	-	-	-	-	-	-	-	-		
Production Highlights																	
Tonnage Mined	*kt*			36,000	36,000	35,129	32,308	32,631	32,000	33,944	36,000	32,000	32,000	35,801	32,015	17,520	-
Tonnage Processed	*kt*			18,657	18,121	17,733	17,539	17,511	17,133	18,064	19,080	18,785	16,441	17,528	16,810	11,070	-
Iron concentrate - dry	*k dmt*			5,942	6,000	6,000	6,000	6,000	6,000	6,000	5,966	6,000	5,915	6,000	6,000	4,287	-
Cash flow waterfall (k CAD)			CAD/t of conc.														
Income																	
Revenue (FOB Pointe Noire) (after 1 % deduction for losses)			87.10	477,479	482,105	482,105	482,105	482,105	482,105	482,105	479,412	482,105	475,261	482,105	482,105	344,454	-
Royalties MFC			-5.75	(31,360)	(31,676)	(31,676)	(31,676)	(31,676)	(31,676)	(31,676)	(31,492)	(31,676)	(31,208)	(31,676)	(31,676)	(22,586)	-
Other Royalties			-1.18	(7,038)	(7,106)	(7,106)	(7,106)	(7,106)	(7,106)	(7,106)	(7,066)	(7,106)	(7,005)	(7,106)	(7,106)	(5,077)	-
Total			80.16	439,081	443,323	443,323	443,323	443,323	443,323	443,323	440,854	443,323	437,048	443,323	443,323	316,791	-
Operational Expenditure																	
Mining		100%	-14.74	(90,068)	(89,984)	(89,747)	(87,870)	(89,150)	(85,734)	(82,908)	(85,277)	(81,785)	(98,610)	(91,733)	(78,109)	(46,538)	-
Processing		100%	-14.31	(87,060)	(84,561)	(82,751)	(81,846)	(81,714)	(79,949)	(84,296)	(89,038)	(87,657)	(76,722)	(81,792)	(78,444)	(51,658)	-
General & Administration		100%	-2.61	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(15,500)	(11,625)	-
Concentrate Rail Transport and Port			-20.27	(116,867)	(117,689)	(117,689)	(117,689)	(117,689)	(117,689)	(117,689)	(117,211)	(117,689)	(116,473)	(117,689)	(117,689)	(85,224)	-
Total Opex		100%	-51.93	(309,496)	(307,734)	(305,687)	(302,905)	(304,053)	(298,872)	(300,394)	(307,026)	(302,632)	(307,305)	(306,714)	(289,743)	(195,045)	-
Operating Cost (CAD/t ore)				16.59	16.98	17.24	17.27	17.36	17.44	16.63	16.09	16.11	18.69	17.50	17.24	17.62	-
Working capital adjustments (during ops.)			-0.25	3,299	- 3,521	1,540	- 2,066	1,139	- 1,724	2,376	930	389			6,100	12,500	
Operating cash flow			27.98	132,884	132,068	139,176	138,352	140,409	142,727	145,305	134,758	141,080	129,743	136,609	159,680	134,246	-
Upgrade Capital Costs																	
Upgrade Capital Costs				-	-	-	-	-	-	-	-	-	-	-	-	-	-
Upgrade Capital Costs-Contingency				-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pre-production revenue				-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sub-total Upgrade Capital Costs		100%		-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total upgrade capital cost				-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital Costs																	
Mining Equipment and Major Repairs				(10,491)	(80,269)	(27,218)	(28,478)	(7,564)	(18,417)	(7,989)	(8,781)	(28,798)	(14,151)	(13,839)	(4,997)	-	-
Concentrator and Infrastructure				(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	-	-	-
Tailings Earthworks				-	-	(1,115)	-	(1,115)	-	(1,225)	-	(1,089)	-	(2,046)	(1,319)	-	-
Tails Pipeline				(648)	(648)	(648)	(648)	(648)	(648)	-	-	-	-	(4,940)	-	-	-
Grand Total Contingency				(1,072)	(4,561)	(2,075)	(1,971)	(1,093)	(1,468)	(1,033)	(889)	(2,053)	(1,158)	(2,190)	(448)	-	-
Total sustaining capital				(15,211)	(88,478)	(34,057)	(34,097)	(13,420)	(23,534)	(13,248)	(12,670)	(34,941)	(18,309)	(26,015)	(6,763)	-	-
Closure & Rehabilitation Costs																	
Closure & reclamation expenditures				(1,984)	-	-	-	-	-	-	-	-	-	-	-	(20,933)	(43,725)
Closure & reclamation cash flows				(1,984)	-	-	-	-	-	-	-	-	-	-	-	(20,933)	(43,725)
Cash flow Before Tax				115,690	43,590	105,119	104,255	126,989	119,194	132,058	122,088	106,140	111,434	110,593	152,918	113,313	(43,725)
Tax																	
Provincial income tax				(15,385)	(15,529)	(14,695)	(15,278)	(15,619)	(16,982)	(17,198)	(16,371)	(17,400)	(15,653)	(16,773)	(19,605)	(6,489)	-
Federal income tax				(15,385)	(15,529)	(14,695)	(15,278)	(15,619)	(16,982)	(17,198)	(16,371)	(17,400)	(15,653)	(16,773)	(19,605)	(6,489)	-
Total				(30,771)	(31,059)	(29,390)	(30,556)	(31,239)	(33,964)	(34,395)	(32,743)	(34,800)	(31,306)	(33,546)	(39,209)	(12,978)	-
Net Cash Flow				84,919	12,532	75,729	73,698	95,750	85,230	97,662	89,345	71,340	80,128	77,047	113,708	100,335	(43,725)

Table 22.2: Revenue Estimates (USD/t of Concentrate)

		2021				2022	2023	2024	2025	LT
Benchmark		**1Q**	**2Q**	**3Q**	**4Q**					
62% Fe China		164.83	154.20	142.35	130.57	112.12	89.00	72	72	70
65% Fe Premium		26.00	26.30	26.41	25.19	23.17	21.00	17	15	15
Other Premium		0.98	0.82	0.76	0.71	2.00	2.00	2	2	2
Cargill Marketing		-7.50	-7.53	-7.54	-7.23	-7.04	-6.50	-5.50	-5.00	-5.00
Revenue (CFR Client)		184.31	173.79	161.98	149.24	130.25	105.50	85.50	84.00	82.00
Freight		-19.09	-17.46	-19.43	-19.62	-17.01	-19.08	-17.28	-17.28	-17.07
Net Revenue (FOB Pointe Noire)		165.22	156.33	142.55	129.62	113.24	86.42	68.22	66.72	64.93
Rail Transport - QNSL		-18.27	-15.87	-14.92	-13.96	-13.25	-11.38	-10.01	-10.01	-9.85
Railcar Maintenance		-0.53	-0.37	-0.35	-0.32	-0.27	-0.27	-0.27	-0.27	-0.27
Port SFPPN		-9.93	-7.29	-5.85	-5.64	-6.51	-5.56	-5.56	-5.56	-5.56
Royalties	**MFC**	-11.24	-10.62	-9.64	-8.76	-7.61	-5.73	-4.45	-4.35	-4.22
	Others	-0.34	-0.34	-0.34	-0.34	-0.84	-0.84	-0.84	-0.84	-0.84
Net Revenue (FOB Scully)		**124.91**	**121.84**	**111.45**	**100.60**	**84.76**	**62.64**	**47.09**	**45.69**	**44.19**

Notes: Exchange Rate: 1.25 CAD/USD
Mass Recovery: 34.65%
Concentrate Losses at Port: 1%
Net Revenue (FOB) Pointe Noire will be reduced to 99%

It is estimated that the Scully concentrates will be sold mainly in China and other Asian countries (60%) and less in Europe and the Middle East (40%). The ocean freight costs will vary significantly depending on the location of the client and the global shipping markets. The freight cost estimates provided to us fluctuate in the USD 17-20/t range; a 1% humidity content is assumed in the calculations.

22.2 Rail Transportation and Port Charges

Tacora has life-of-mine ("LOM") agreements with third parties to ensure rail transportation of the concentrate from Scully to the Pointe Noire port and for the port storage and ship loading activities. Cost for the Quebec North Shore and Labrador Railway ("QNS&L") portion of the rail transportation from the Wabush Junction to the Arnaud Junction has a base component plus a reference price participation. The cost for the *Société ferroviaire et portuaire de Pointe-Noire ("SFPPN")* covers the rail transportation from the Arnaud Junction to Pointe Noire and all port activities; it has a fixed monthly charge and a variable component based on concentrate tonnage handled. It is also assumed that a 1% concentrate loss occurs at Pointe Noire. The financial model simply assumed revenues of 99% of the Net Price at Pointe Noire.

The total costs for rail transportation and port services are currently in the range of USD 20-30/t of concentrate because of the exceptionally high reference concentrate price. Under long term reference price market, the rail transportation and port costs would be USD 16/t. It must be noted that the agreements with QNS&L and SFPPN are in CAD; the costs indicated in Table 1.12 previously were converted to USD on the basis of CAD 1.25/USD.

22.3 Royalties

Certain royalties are payable on the basis of concentrate tonnage shipped from Pointe Noire.

A first royalty is based on the terms of the Nalco-Javelin (Minerals Land) Act of 1957 which requires payment of CAD 0.22/long ton (CAD 0.2187/t) of concentrate to the Government of Newfoundland & Labrador in lieu of any other provincial mining taxation.

A second royalty is the result of IBA negotiations with the First Nations and will involve payments of CAD 0.50/t of concentrate once the Scully Mine generates a positive cash flow.

A third royalty is payable to Knoll Lake, a corporation controlled by Tacora, which amounts to a net cost to Tacora of CAD 0.117/t of concentrate.

Finally, the more substantial royalty is payable to MFC Bancorp on the basis of 7% of net revenues at Pointe Noire minus certain deductions. The cost to Tacora is currently in the high range of USD 9-11/t of concentrate but would decline to USD 4-5/t under long term market conditions.

After deductions of royalties, Tacora is currently benefiting from revenues FOB Scully of USD 100-125/t of concentrate (CAD 125-155/t) shipped. Over the long-term, based on modest growth in steel production and iron ore demand, net revenues FOB Scully are expected to settle down to USD 44/t of concentrate (CAD 55/t).

22.4 Scully OPEX and CAPEX

Tacora is in the process of completing repairs and replacement of existing processing equipment, purchasing of mining equipment and other investments in the infrastructures required to produce 6 Mtpy of iron ore concentrates on a sustainable basis. These remaining expenditures total CAD 95M starting in 2021 and would be completed by the end of 2022. Details are discussed in Subsection 21.1.1 under Upgrade Capital Expenditures and summarized in Table 22.3.

Table 22.3: Upgrade CAPEX

	2021 CAD (000)	2022 CAD (000)	Total CAD (000)
Mine Equipment	8,513	9,058	17,570
Plant (Concentrator + Infrastructure)	16,584	15,000	31,584
Port Upgrades	-	15,880	15,880
Indirects + Contingency	10,600	19,394	29,994
Total	**35,697**	**59,332**	**95,028**

Once sustainable operations are reached, Sustaining Capital Expenditures of CAD 618.2M were estimated over the 26 years of continuous production of the 6 Mt of iron ore concentrates per year. Details are provided in Subsection 21.2 and a summary is presented in Table 22.4. A yearly schedule of expenditures is listed in Table 1.12.

It is calculated that the residual book value of the Major Mine Equipment at the end of the mine life would be CAD 72,853,000. This value would likely be realized only if additional reserves are found and the mine life is extended. If this equipment had to be sold in the used equipment market, it is estimated that the net realized value would be less than 25% of the remaining book value. Due to the uncertainty of the mine equipment salvage value and its relatively small NPV in 27 years, its contribution was excluded from the project economics.

Table 22.4: Sustaining CAPEX

	CAD (000)
Mine	
Addition + Replacement	214,883
Major Repairs	224,810
Plant (Conc. + Infrastructures)	72,000
Tailings	
Earthwork	32,748
Pump & Pipelines	31,385
Contingency	42,405
Total	**618,231**

During operations, all transportation of products from the Scully Mine to the Pointe Noire port and all shipping from Pointe Noire to overseas clients are performed by third parties under LOM agreements including price formulas. Tacora is responsible for operations and costs at the Scully site. These operating costs (“OPEX”) are distributed in three sectors as described in Subsection 21.3: Mining, Processing and G&A. Yearly costs are detailed in Table 1.12 and LOM average costs are as shown in Table 1.9.

Table 22.5: OPEX

	$/t of concentrate		$/t milled		$/t mined	
	CAD	USD	CAD	USD	CAD	USD
Mine	14.74	11.79	4.90	3.92	2.70	2.16
Process	14.31	11.45	4.75	3.80	-	-
G&A	2.61	2.09	0.87	0.69	-	-
Total	**31.66**	**25.33**	**10.52**	**8.41**	**2.70**	**2.16**

22.5 Working Capital

In order to evaluate the investment required in the Working Capital, the basic information was obtained from the Tacora Corporate Financial model which provide an estimated statement of short-term assets and liabilities, including the following:

Table 22.6: Short Term Assets & Liabilities

Current Assets	Current Liabilities
Receivables	Payables
Product Inventory	Wages and Benefits
Parts & Consumables Inventory	Royalties
Pre-Payments	Other Current Liabilities
Other Current Assets	

Cash position and deferred taxes were excluded from the working capital.

The working capital will increase significantly in the initial years of production as receivables and inventories grow with production and the liabilities decrease with the end of upgrades and construction at the Scully Mine.

The investments initially required in the Working Capital are as follows:

Table 22.7: Working Capital Requirement – Early years

	CAD (000)
2021	41,461
2022	16,716
2023	3,231

Fluctuations in the working capital afterwards remain in the ±CAD 2M until 2046-2047 when the working capital is liquidated.

22.6 Rehabilitation and Closure Costs

Subsection 20.7 of the Report lists the activities involved in the reclamation and closure of the Scully site. Some activities such as progressive grading and revegetation of the tailings pond and waste dumps will be scheduled during operations with expenditures of CAD 31,984,000 including contingency. However, the greater amount of expenditures for closure will occur in 2047-2048 at the end of operations and are budgeted as follows:

Table 22.8: Rehabilitation and Closure Costs

	CAD (000)
Progressive	31,984
Closure 2047	14,575
Closure 2048	43,724
Total	**90,283**

Some activities will involve post closure monitoring and their budgets are included in the 2048 expenditures for simplicity.

22.7 Taxation

Tacora is exempt from the current regime of mining taxation in Newfoundland & Labrador. On the other hand, it is required to pay the Nalco-Javelin royalty described in Subsection 22.3 in lieu of mining taxes.

However, Tacora is subject to federal and provincial income tax at a rate of 15% of taxable income at each level of government (total 30%). Taxable income is calculated as the remaining amount after deducting from revenues all OPEX and capital cost allowances based on schedules specified by the tax authorities. Taxes are payables in instalments monthly or quarterly and tax losses can be recovered from taxes in future years or from taxes paid in the prior three years.

Tacora has extensive tax pools of CAD 310M at the end of 2020.

Table 22.9: Income Tax Pools (December 2020)

Tax Pool	Amount CAD (000)	Deductibility
Accumulated Tax Losses	155,612	100% of tax payable
CDE	1,097	30% declining balance
Asset Class 14.1	37,566	5% declining balance
Asset Class 1	1,218	4% declining balance
Asset Class 41	114,019	25% declining balance
Total	**309,512**	

Remaining Upgrade CAPEX of CAD 95,028,000 and Sustaining CAPEX of CAD 618,232,000 for the LOM period of 2021 to 2047 are assigned to the following tax pools.

Table 22.10: Additional Tax Pools

Tax Pool	Amount CAD (000)	Deductibility
Asset Class 41	687,639	25% declining balance
Asset Class 14.1	25,621	5% declining balance

The detailed tax calculations for the Base Case are presented in Table 22.11.

22.8 Valuation – Base Case

The valuation of the Scully Mine is based on the cashflows estimated over the 27 years of Life-of-Mine (LOM). The start of the valuation is based on the situation prevailing on January 1st, 2021. Prior expenditures to bring the Mine out of receivership and to upgrade facilities are excluded from the analysis; however tax pools available at the end of 2020 were used in the tax calculations for following years.

The net cashflows before and after taxes over the LOM are presented in Table 22.11 and Table 22.12 and detailed in Table 22.13. Over the period 2021-2023, the Scully Mine will benefit from the price super-cycle being experienced in the iron ore industry; net cashflow is heavily leveraged to Benchmark prices. It is also the period when income taxes will be at their highest levels between CAD 71 M to 115 M per year.

Table 22.11: Taxation

Taxation

Tacora Scully Mine

		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Period start		01-Jan-21	01-Jan-22	01-Jan-23	01-Jan-24	01-Jan-25	01-Jan-26	01-Jan-27	01-Jan-28	01-Jan-29	01-Jan-30	01-Jan-31	01-Jan-32	01-Jan-33	01-Jan-34	01-Jan-35
Period end	01-Jan-21	31-Dec-21	31-Dec-22	31-Dec-23	31-Dec-24	31-Dec-25	31-Dec-26	31-Dec-27	31-Dec-28	31-Dec-29	31-Dec-30	31-Dec-31	31-Dec-32	31-Dec-33	31-Dec-34	31-Dec-35
Operations	27	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
Closure	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Capital Expenditures																
Upgrade Capital Costs																
Upgrade Capex Conc. (exept SFPPN port upgrades)	61,882	31,780	30,102	-	-	-	-	-	-	-	-	-	-	-	-	-
Upgrade Capex Conc. (exept SFPPN port upgrades) Contingency	7,525	3,916	3,610	-	-	-	-	-	-	-	-	-	-	-	-	-
Upgrade Capex Conc. (SFPPN port upgrades)	22,279	-	22,279	-	-	-	-	-	-	-	-	-	-	-	-	-
Upgrade Capex Conc. (SFPPN port upgrades) Contingency	3,342	-	3,342	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Upgrade Capex	95,028	35,696	59,332	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining CAPEX																
Mining Equipment and Major Repairs	439,694	-	11,772	5,981	16,472	19,510	11,346	7,648	18,302	15,862	13,286	18,920	17,387	13,776	18,441	10,491
Mining Equipment and Major Repairs Contingency	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Concentrator and Infrastructure	72,000	-	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000
Concentrator and Infrastructure Contingency	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tailings Earthworks	32,748	-	228	731	6,321	-	-	1,183	1,843	9,267	-	-	1,115	1,225	2,925	-
Tailings Earthworks Contingency	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tails Pipeline	31,385	-	1,107	-	6,726	1,081	1,081	1,081	1,081	1,081	6,726	648	648	648	648	648
Grand Total Contingency	42,405	-	1,239	859	3,231	1,588	1,179	1,172	1,804	2,795	2,123	1,493	1,584	1,420	1,908	1,072
Total Sustaining Capex	618,232	-	17,345	10,570	35,749	25,178	16,606	14,083	26,029	32,005	25,135	24,062	23,735	20,070	26,923	15,211
Tax Pool Additions																
Class 41	687,640	35,696	51,057	10,570	35,749	25,178	16,606	14,083	26,029	32,005	25,135	24,062	23,735	20,070	26,923	15,211
Class 14.1	25,621	-	25,621	-	-	-	-	-	-	-	-	-	-	-	-	-
Class 1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
CDE	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total CapEx	713,260	35,696	76,677	10,570	35,749	25,178	16,606	14,083	26,029	32,005	25,135	24,062	23,735	20,070	26,923	15,211
Provincial & Federal Income Tax																
Income Tax																
Net revenues	12,752,920	859,068	776,662	592,480	466,024	455,860	443,323	443,233	442,361	443,323	443,249	442,965	439,322	441,877	439,490	439,081
Less: Operating costs including royalties and IBA	(8,261,704)	(354,627)	(354,811)	(317,001)	(309,522)	(307,334)	(304,529)	(308,014)	(317,107)	(314,707)	(315,809)	(307,566)	(307,974)	(303,159)	(301,938)	(309,496)
Less: Reclamation costs	(90,283)	(432)	(648)	(2,913)	(347)	(5,597)	-	(8,864)	-	(1,021)	(2,045)	-	-	-	(1,776)	(1,984)
Taxable income before depreciation and NOLs carryforward	4,400,933	504,009	421,203	272,567	156,155	142,929	138,794	126,355	125,254	127,596	125,395	135,398	131,349	138,718	135,776	127,602
Tax depreciation - Class 41	(745,347)	(32,967)	(35,569)	(34,380)	(31,575)	(31,297)	(28,696)	(25,358)	(24,032)	(25,279)	(26,101)	(25,726)	(25,269)	(24,427)	(24,195)	(23,413)
Tax depreciation - Class 14.1	(46,853)	(1,878)	(2,425)	(2,944)	(2,797)	(2,657)	(2,524)	(2,398)	(2,278)	(2,164)	(2,056)	(1,953)	(1,856)	(1,763)	(1,675)	(1,591)
Tax depreciation - Class 1	(813)	(49)	(47)	(45)	(43)	(41)	(40)	(38)	(37)	(35)	(34)	(32)	(31)	(30)	(29)	(28)
Tax depreciation - CDE	(1,097)	(329)	(230)	(161)	(113)	(79)	(55)	(39)	(27)	(19)	(13)	(9)	(7)	(5)	(3)	(2)
Taxable income - Before NOLs carryforward	3,606,822	468,786	382,932	235,036	121,627	108,855	107,479	98,522	98,879	100,099	97,191	107,678	104,186	112,493	109,875	102,568
Less: Net operating losses carryforward	(199,337)	(155,612)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Taxable income - After NOLs carryforward	3,451,210	313,174	382,932	235,036	121,627	108,855	107,479	98,522	98,879	100,099	97,191	107,678	104,186	112,493	109,875	102,568
Tax base federal	3,451,210	313,174	382,932	235,036	121,627	108,855	107,479	98,522	98,879	100,099	97,191	107,678	104,186	112,493	109,875	102,568
Federal income tax rate		15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%
Cash tax - basic	517,682	46,976	57,440	35,255	18,244	16,328	16,122	14,778	14,832	15,015	14,579	16,152	15,628	16,874	16,481	15,385
Total Federal tax	517,682	46,976	57,440	35,255	18,244	16,328	16,122	14,778	14,832	15,015	14,579	16,152	15,628	16,874	16,481	15,385
Tax base provincial	3,451,210	313,174	382,932	235,036	121,627	108,855	107,479	98,522	98,879	100,099	97,191	107,678	104,186	112,493	109,875	102,568
Newfoundland income tax rate		15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%
Total provincial tax	517,682	46,976	57,440	35,255	18,244	16,328	16,122	14,778	14,832	15,015	14,579	16,152	15,628	16,874	16,481	15,385
Tax Summary																
Provincial income tax	517,682	46,976	57,440	35,255	18,244	16,328	16,122	14,778	14,832	15,015	14,579	16,152	15,628	16,874	16,481	15,385
Federal income tax	517,682	46,976	57,440	35,255	18,244	16,328	16,122	14,778	14,832	15,015	14,579	16,152	15,628	16,874	16,481	15,385
Total	1,035,363	93,952	114,880	70,511	36,488	32,656	32,244	29,557	29,664	30,030	29,157	32,303	31,256	33,748	32,963	30,771

Taxation

Tacora Scully Mine

		16	17	18	19	20	21	22	23	23	23	23	23	23
Period start		01-Jan-36	01-Jan-37	01-Jan-38	01-Jan-39	01-Jan-40	01-Jan-41	01-Jan-42	01-Jan-43	01-Jan-44	01-Jan-45	01-Jan-46	01-Jan-47	01-Jan-48
Period end	01-Jan-21	31-Dec-36	31-Dec-37	31-Dec-38	31-Dec-39	31-Dec-40	31-Dec-41	31-Dec-42	31-Dec-43	31-Dec-44	31-Dec-45	31-Dec-46	31-Dec-47	31-Dec-48
Operations	27	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.7	-
Closure	1	-	-	-	-	-	-	-	-	-	-	-	0.3	0.7
Capital Expenditures														
Upgrade Capital Costs														
Upgrade Capex Conc. (exept SFPPN port upgrades)	61,882	-	-	-	-	-	-	-	-	-	-	-	-	-
Upgrade Capex Conc. (exept SFPPN port upgrades) Contingency	7,525	-	-	-	-	-	-	-	-	-	-	-	-	-
Upgrade Capex Conc. (SFPPN port upgrades)	22,279	-	-	-	-	-	-	-	-	-	-	-	-	-
Upgrade Capex Conc. (SFPPN port upgrades) Contingency	3,342	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Upgrade Capex	95,028	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining CAPEX														
Mining Equipment and Major Repairs	439,694	80,269	27,218	28,478	7,564	18,417	7,989	8,781	28,798	14,151	13,839	4,997	-	-
Mining Equipment and Major Repairs Contingency	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Concentrator and Infrastructure	72,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	-	-	-
Concentrator and Infrastructure Contingency	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tailings Earthworks	32,748	-	1,115	-	1,115	-	1,225	-	1,089	-	2,046	1,319	-	-
Tailings Earthworks Contingency	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tails Pipeline	31,385	648	648	648	648	648	-	-	-	-	4,940	-	-	-
Grand Total Contingency	42,405	4,561	2,075	1,971	1,093	1,468	1,033	889	2,053	1,158	2,190	448	-	-
Total Sustaining Capex	618,232	88,478	34,057	34,097	13,420	23,534	13,248	12,670	34,941	18,309	26,015	6,763	-	-
Tax Pool Additions														
Class 41	687,640	88,478	34,057	34,097	13,420	23,534	13,248	12,670	34,941	18,309	26,015	6,763	-	-
Class 14.1	25,621	-	-	-	-	-	-	-	-	-	-	-	-	-
Class 1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
CDE	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total CapEx	713,260	88,478	34,057	34,097	13,420	23,534	13,248	12,670	34,941	18,309	26,015	6,763	-	-
Provincial & Federal Income Tax														
Income Tax														
Net revenues	12,752,920	443,323	443,323	443,323	443,323	443,323	443,323	440,854	443,323	437,048	443,323	443,323	316,791	-
Less: Operating costs including royalties and IBA	(8,261,704)	(307,734)	(305,687)	(302,905)	(304,053)	(298,872)	(300,394)	(307,026)	(302,632)	(307,305)	(306,714)	(289,743)	(195,045)	-
Less: Reclamation costs	(90,283)	-	-	-	-	-	-	-	-	-	-	-	(20,933)	(43,725)
Taxable income before depreciation and NOLs carryforward	4,400,933	135,589	137,636	140,418	139,270	144,451	142,929	133,828	140,691	129,743	136,609	153,580	100,813	(43,725)
Tax depreciation - Class 41	(745,347)	(30,521)	(38,207)	(37,175)	(33,821)	(29,985)	(27,086)	(23,554)	(23,617)	(24,369)	(23,817)	(21,960)	(12,951)	-
Tax depreciation - Class 14.1	(46,853)	(1,511)	(1,436)	(1,364)	(1,296)	(1,231)	(1,169)	(1,111)	(1,055)	(1,003)	(953)	(905)	(860)	-
Tax depreciation - Class 1	(813)	(26)	(25)	(24)	(23)	(22)	(22)	(21)	(20)	(19)	(18)	(18)	(17)	-
Tax depreciation - CDE	(1,097)	(2)	(1)	(1)	(1)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	-
Taxable income - Before NOLs carryforward	3,606,822	103,529	97,966	101,854	104,130	113,213	114,652	109,142	115,999	104,352	111,821	130,698	86,985	(43,725)
Less: Net operating losses carryforward	(199,337)	-	-	-	-	-	-	-	-	-	-	-	(43,725)	43,725
Taxable income - After NOLs carryforward	3,451,210	103,529	97,966	101,854	104,130	113,213	114,652	109,142	115,999	104,352	111,821	130,698	43,261	-
Tax base federal	3,451,210	103,529	97,966	101,854	104,130	113,213	114,652	109,142	115,999	104,352	111,821	130,698	43,261	-
Federal income tax rate		15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%
Cash tax - basic	517,682	15,529	14,695	15,278	15,619	16,982	17,198	16,371	17,400	15,653	16,773	19,605	6,489	-
Total Federal tax	517,682	15,529	14,695	15,278	15,619	16,982	17,198	16,371	17,400	15,653	16,773	19,605	6,489	-
Tax base provincial	3,451,210	103,529	97,966	101,854	104,130	113,213	114,652	109,142	115,999	104,352	111,821	130,698	43,261	-
Newfoundland income tax rate		15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%
Total provincial tax	517,682	15,529	14,695	15,278	15,619	16,982	17,198	16,371	17,400	15,653	16,773	19,605	6,489	-
Tax Summary														
Provincial income tax	517,682	15,529	14,695	15,278	15,619	16,982	17,198	16,371	17,400	15,653	16,773	19,605	6,489	-
Federal income tax	517,682	15,529	14,695	15,278	15,619	16,982	17,198	16,371	17,400	15,653	16,773	19,605	6,489	-
Total	1,035,363	31,059	29,390	30,556	31,239	33,964	34,395	32,743	34,800	31,306	33,546	39,209	12,978	-

Once Benchmark prices return to their long-term level, yearly cashflows net of taxes will vary from CAD 65M to CAD 97M and income taxes will fluctuate between CAD 29M to 36M. The Net Present Values (NPV) at various discount factors are presented in Table 1.15 and the detailed calculations are presented in Table 22.13.

Table 22.12: NPV Results – Base Case

Discount	NPV – Before Taxes		NPV – After Taxes	
	(CAD M)	(USD M)	(CAD M)	(USD M)
0%	3,647	2,918	2,612	2,090
8%	1,872	1,498	1,345	1,076
10%	1,675	1,340	1,205	964

Financial markets would probably use the After-Tax NPV with a discount factor of 8-10% to assign a value to the asset. The market value of the Scully mine would then be about CAD 1,275M or USD 1,020M. However, this analysis is very conservative since it assumed a relatively low iron ore price for the last 24 years of operations; in reality, spikes in iron ore prices should be expected from time to time due to market imbalances.

Figure 22.1: Yearly Before-Tax Cash Flows

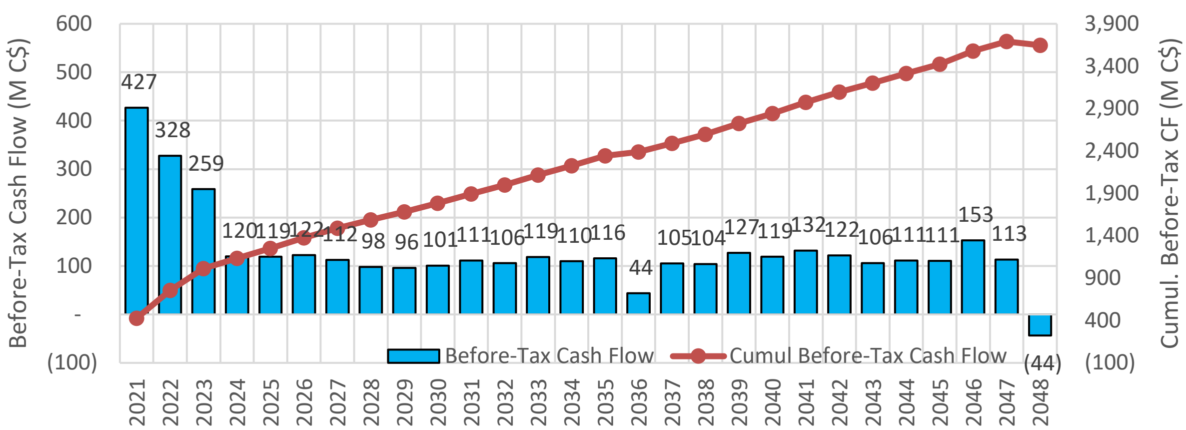


Figure 22.2: Yearly After-Tax Cash Flows

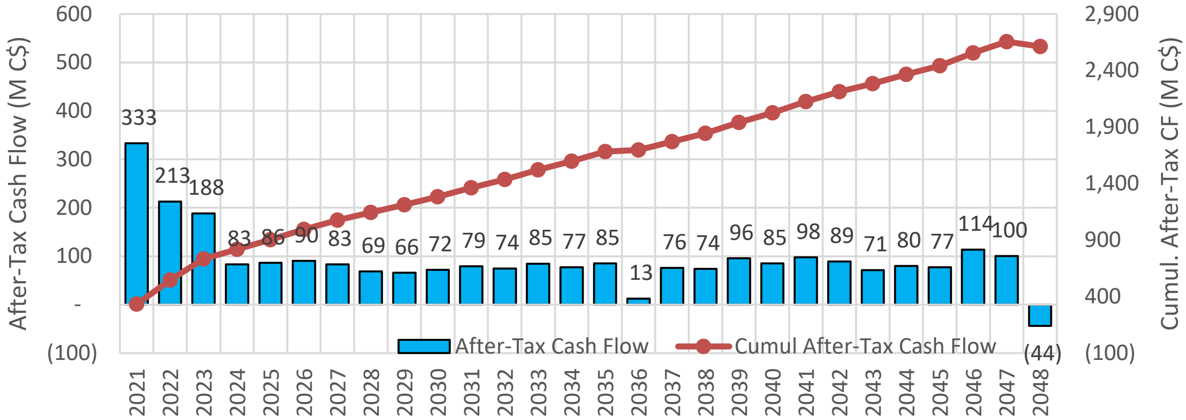


Table 22.13: NPV Results

Annual Cash Flow

Tacora Scully Mine

					2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Period number					2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Cash flow waterfall summary (k CAD)																		
Revenue				12,752,920	859,068	776,662	592,480	466,024	455,860	443,323	443,233	442,361	443,323	443,249	442,965	439,322	441,877	439,490
Total operating costs				(8,261,704)	(354,627)	(354,811)	(317,001)	(309,522)	(307,334)	(304,529)	(308,014)	(317,107)	(314,707)	(315,809)	(307,566)	(307,974)	(303,159)	(301,938)
EBITDA				4,491,216	504,441	421,851	275,479	156,502	148,526	138,794	135,219	125,254	128,616	127,440	135,398	131,349	138,718	137,552
Upgrade capital				(84,161)	(31,780)	(52,381)	-	-	-	-	-	-	-	-	-	-	-	-
Upgrade-Contingency				(10,867)	(3,916)	(6,951)	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining capital				(618,232)	-	(17,345)	(10,570)	(35,749)	(25,178)	(16,606)	(14,083)	(26,029)	(32,005)	(25,135)	(24,062)	(23,735)	(20,070)	(26,923)
Salvage value				-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Change in working capital				(40,289)	(41,461)	(16,716)	(3,231)	(688)	1,343	175	218	(925)	283	738	-	(1,883)	(103)	1,001
Taxes				(1,035,363)	(93,952)	(114,880)	(70,511)	(36,488)	(32,656)	(32,244)	(29,557)	(29,664)	(30,030)	(29,157)	(32,303)	(31,256)	(33,748)	(32,963)
Closure costs				(90,283)	(432)	(648)	(2,913)	(347)	(5,597)	-	(8,864)	-	(1,021)	(2,045)	-	-	-	(1,776)
Free cash flow				2,612,021	332,899	212,930	188,254	83,231	86,437	90,120	82,932	68,636	65,844	71,840	79,033	74,475	84,797	76,892
Returns After-Tax (k CAD)																		
Equity																		
Initial equity	k CAD			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Additional equity	k CAD			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Distributions	k CAD			2,612,021	332,899	212,930	188,254	83,231	86,437	90,120	82,932	68,636	65,844	71,840	79,033	74,475	84,797	76,892
Net cash flow to equity	k CAD			2,612,021	332,899	212,930	188,254	83,231	86,437	90,120	82,932	68,636	65,844	71,840	79,033	74,475	84,797	76,892
Evaluation cash flow	k CAD			(0)	332,899	212,930	188,254	83,231	86,437	90,120	82,932	68,636	65,844	71,840	79,033	74,475	84,797	76,892
Cumulative cash flow	k CAD				332,899	545,829	734,084	817,315	903,752	993,872	1,076,805	1,145,440	1,211,284	1,283,124	1,362,158	1,436,632	1,521,430	1,598,322
Payback Period	years			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Discount rate	% p.a	8.0%	10.0%															
NPV	k CAD	1,344,950	1,205,285	1,344,932	320,434	189,775	155,354	63,591	61,142	59,025	50,294	38,537	34,227	34,578	35,222	30,729	32,393	27,197
Returns Before-Tax (k CAD)																		
Equity																		
Initial equity	k CAD			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Additional equity	k CAD			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Distributions	k CAD			2,612,021	332,899	212,930	188,254	83,231	86,437	90,120	82,932	68,636	65,844	71,840	79,033	74,475	84,797	76,892
Taxes	k CAD			1,035,363	93,952	114,880	70,511	36,488	32,656	32,244	29,557	29,664	30,030	29,157	32,303	31,256	33,748	32,963
Net cash flow to equity	k CAD			3,647,384	426,852	327,810	258,765	119,719	119,094	122,364	112,489	98,300	95,873	100,997	111,337	105,731	118,545	109,855
Evaluation cash flow	k CAD			(0)	426,852	327,810	258,765	119,719	119,094	122,364	112,489	98,300	95,873	100,997	111,337	105,731	118,545	109,855
Cumulative cash flow	k CAD				426,852	754,661	1,013,426	1,133,145	1,252,239	1,374,603	1,487,092	1,585,392	1,681,265	1,782,262	1,893,599	1,999,330	2,117,875	2,227,730
Payback Period	years			-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Discount rate	% p.a	8.0%	10.0%															
NPV	k CAD	1,871,812	1,675,367	1,871,786	410,868	292,162	213,542	91,469	84,242	80,144	68,218	55,192	49,837	48,611	49,618	43,625	45,285	38,856

Annual Cash Flow

Tacora Scully Mine

Cash flow waterfall summary (k CAD)

Period number	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048
Revenue	439,081	443,323	443,323	443,323	443,323	443,323	443,323	440,854	443,323	437,048	443,323	443,323	316,791	-
Total operating costs	(309,496)	(307,734)	(305,687)	(302,905)	(304,053)	(298,872)	(300,394)	(307,026)	(302,632)	(307,305)	(306,714)	(289,743)	(195,045)	-
EBITDA	129,585	135,589	137,636	140,418	139,270	144,451	142,929	133,828	140,691	129,743	136,609	153,580	121,746	-
Upgrade capital	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Upgrade-Contingency	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining capital	(15,211)	(88,478)	(34,057)	(34,097)	(13,420)	(23,534)	(13,248)	(12,670)	(34,941)	(18,309)	(26,015)	(6,763)	-	-
Salvage value	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Change in working capital	3,299	(3,521)	1,540	(2,066)	1,139	(1,724)	2,376	930	389	-	-	6,100	12,500	-
Taxes	(30,771)	(31,059)	(29,390)	(30,556)	(31,239)	(33,964)	(34,395)	(32,743)	(34,800)	(31,306)	(33,546)	(39,209)	(12,978)	-
Closure costs	(1,984)	-	-	-	-	-	-	-	-	-	-	-	(20,933)	(43,725)
Free cash flow	84,919	12,532	75,729	73,698	95,750	85,230	97,662	89,345	71,340	80,128	77,047	113,708	100,335	(43,725)

Returns After-Tax (k CAD)

				2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048
Equity																	
Initial equity	k CAD			-	-	-	-	-	-	-	-	-	-	-	-	-	-
Additional equity	k CAD			-	-	-	-	-	-	-	-	-	-	-	-	-	-
Distributions	k CAD			84,919	12,532	75,729	73,698	95,750	85,230	97,662	89,345	71,340	80,128	77,047	113,708	100,335	(43,725)
Net cash flow to equity	k CAD			84,919	12,532	75,729	73,698	95,750	85,230	97,662	89,345	71,340	80,128	77,047	113,708	100,335	(43,725)
Evaluation cash flow	k CAD			84,919	12,532	75,729	73,698	95,750	85,230	97,662	89,345	71,340	80,128	77,047	113,708	100,335	(43,725)
Cumulative cash flow	k CAD			1,683,241	1,695,773	1,771,501	1,845,200	1,940,950	2,026,180	2,123,842	2,213,187	2,195,182	2,293,315	2,370,362	2,484,071	2,584,405	2,540,681
Payback Period	years			-	-	-	-	-	-	-	-	-	-	-	-	-	-
Discount rate	% p.a	8.0%	10.0%														
NPV	k CAD	1,344,950	1,205,285	27,811	3,800	21,259	19,156	23,045	18,991	20,147	17,066	12,618	13,121	11,681	15,962	13,041	(5,262)

Returns Before-Tax (k CAD)

				2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048
Equity																	
Initial equity	k CAD			-	-	-	-	-	-	-	-	-	-	-	-	-	-
Additional equity	k CAD			-	-	-	-	-	-	-	-	-	-	-	-	-	-
Distributions	k CAD			84,919	12,532	75,729	73,698	95,750	85,230	97,662	89,345	71,340	80,128	77,047	113,708	100,335	(43,725)
Taxes	k CAD			30,771	31,059	29,390	30,556	31,239	33,964	34,395	32,743	34,800	31,306	33,546	39,209	12,978	-
Net cash flow to equity	k CAD			115,690	43,590	105,119	104,255	126,989	119,194	132,058	122,088	106,140	111,434	110,593	152,918	113,313	(43,725)
Evaluation cash flow	k CAD			115,690	43,590	105,119	104,255	126,989	119,194	132,058	122,088	106,140	111,434	110,593	152,918	113,313	(43,725)
Cumulative cash flow	k CAD			2,343,420	2,387,010	2,492,129	2,596,383	2,723,372	2,842,566	2,974,623	3,096,711	3,080,763	3,208,145	3,318,738	3,471,656	3,584,969	3,541,244
Payback Period	years			-	-	-	-	-	-	-	-	-	-	-	-	-	-
Discount rate	% p.a	8.0%	10.0%														
NPV	k CAD	1,871,812	1,675,367	37,889	13,217	29,509	27,099	30,563	26,559	27,243	23,321	18,773	18,247	16,766	21,465	14,728	(5,262)

22.9 Sensitivity Analyses

There are many scenarios that can be tested to estimate the impact of variable parameters on the value of the Scully Mine. The following parameters were selected:

- Net Revenues – Pointe Noire
- CAD/USD exchange rate
- Operating Costs at Scully Mine
- Sustaining Capital

The ± 10% and ± 20% variations of these parameters were computed to study the resulting fluctuations in NPV values. In all cases, the data for 2021 were fixed and not affected by sensitivities. Net Revenues – Pointe Noire were selected to study the impact of fluctuations in iron ore price; these fluctuations can be due to variations in Benchmark prices, quality premiums, location of clients and related ocean freight. Costs for rail transportation and the MFC royalties are partially indexed to the Net Revenues - Pointe Noire.

Variations in the After-Tax NPV's for the four sensitivities are presented in Table 22.14, Table 22.15, Table 22.16 and Table 22.17 and are graphed in Figure 1.5.

Table 22.14: Sensitivity to Net Revenues – Pointe Noire

	NPV – After Taxes (CAD M)				
Discount	-20%	-10%	Base	+10%	+20%
0%	1,084	1,850	2,612	3,374	4,136
8%	700	1,023	1,345	1,667	1,990
10%	656	931	1,205	1,480	1,754

Table 22.15: Sensitivity to Exchange Rate

	NPV – After Taxes (CAD M)				
Discount	-20%	-10%	Base	+10%	+20%
0%	924	1,771	2,612	3,453	4,294
8%	635	989	1,345	1,701	2,056
10%	599	901	1,205	1,508	1,811

Table 22.16: Sensitivity to Scully Operating Costs

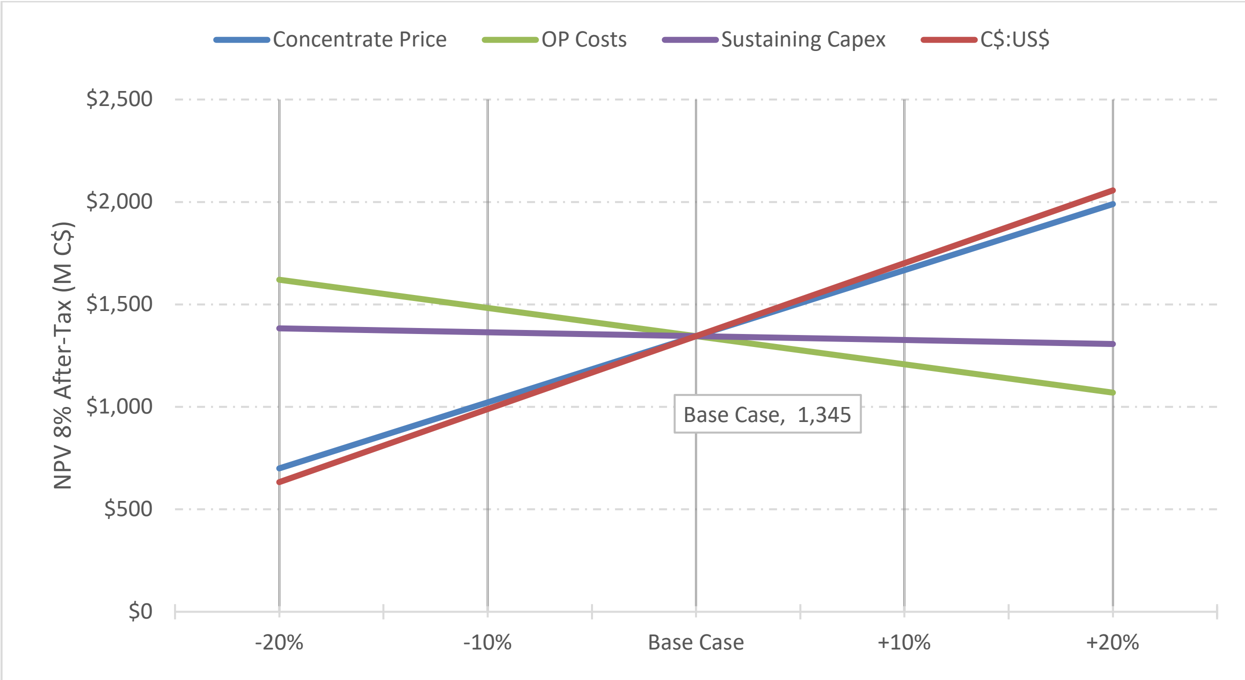
	NPV – After Taxes (CAD M)				
Discount	-20%	-10%	Base	+10%	+20%
0%	3,288	2,950	2,612	2,274	1,936
8%	1,620	1,483	1,345	1,207	1,069
10%	1,437	1,321	1,205	1,089	973

Table 22.17: Sensitivity to Sustaining Capital

	NPV – After Taxes (CAD M)				
Discount	-20%	-10%	Base	+10%	+20%
0%	2,702	2,657	2,612	2,567	2,522
8%	1,383	1,364	1,345	1,326	1,307
10%	1,237	1,221	1,205	1,189	1,173

When looking at results in Canadian dollars, the after-tax NPV's are particularly sensitive to fluctuations in revenues that can be caused by Benchmark prices, premiums, client location, ocean freight and exchange rate; a 20% variation in Net Revenues at Pointe Noire and exchange rate will result in an NPV (8%) differential of CAD 645M and CAD 711M, respectively. The next parameter of importance is the operating cost at the Scully mine with an NPV (8%) differential of CAD 276M for a 20% variation. Finally, a 20% variation in sustaining capital cause only an NPV (8%) reaction of CAD 32M.

Figure 22.3: Sensitivity – 8% NPV – After Tax



23 ADJACENT PROPERTIES

The Scully Mine is situated in an active iron ore mining district, as illustrated in Figure 23.1.

23.1 Quebec Iron Ore – Bloom Lake

The Bloom Lake Mine is located 22 km west of Wabush. Consolidated Thompson Iron Mines Limited began the construction of the mining infrastructures in 2008 and commenced mining operations in 2010 with the Phase 1 concentrator for a production of 7.4 Mtpy of iron concentrate. As part of an expansion plan to increase the mine production, the design and construction of a second concentrator was initiated to increase nominal capacity to about 15 Mt of concentrate per annum.

The mine was sold to Cliffs in 2011, which continued the Phase 2 construction project and conducted mining operations until they were suspended in December 2014, due to financial distress caused by a sharp decrease of iron ore prices. In January 2015, Cliffs sought creditor protection, resulting in the mine being put on a strict care and maintenance program, and placed into creditor protection.

In April of 2016, Québec Iron Ore Inc. (“QIO”), a wholly owned subsidiary of Champion Iron Limited (“Champion”) acquired the Bloom Lake assets and resumed operations in 2018. Production in the last nine months of 2020 was 5,989,700 wmt of concentrate at 66.3% Fe from 15,360,900 wmt of crude ore at 30.6% Fe.

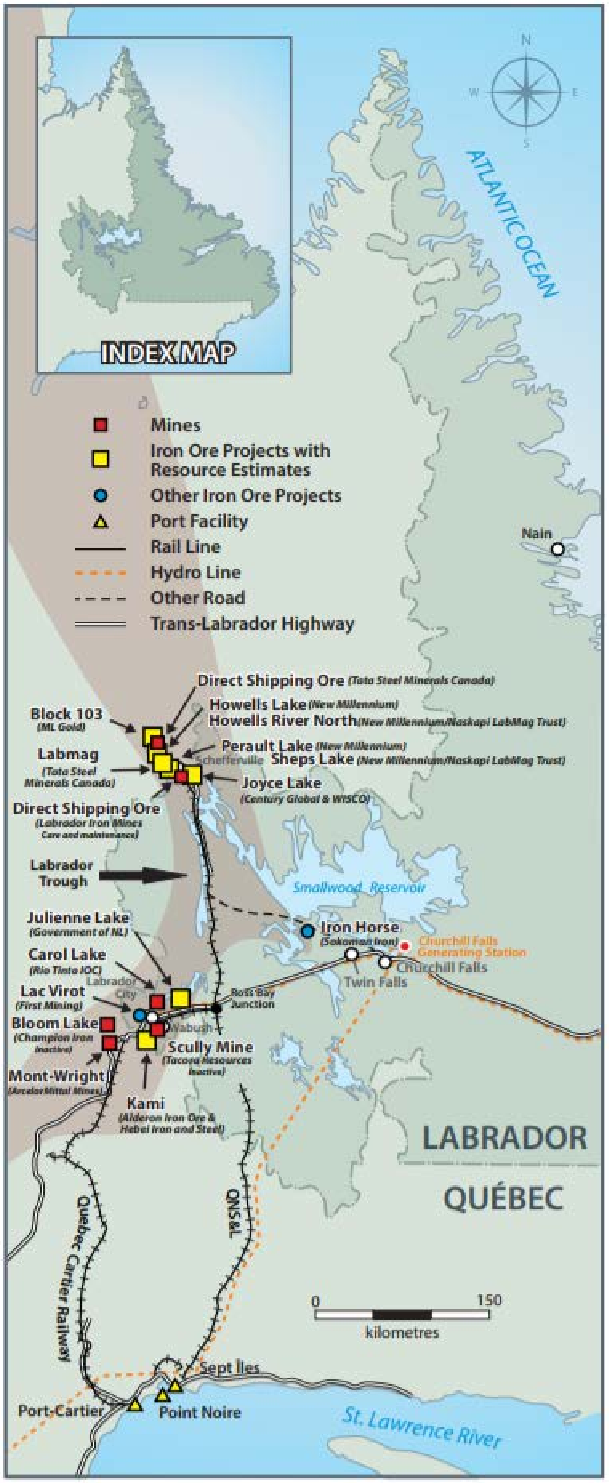
23.2 ArcelorMittal – Mont-Wright & Fire Lake

ArcelorMittal owns two mines in the area. The Mont-Wright Mine is approximately 35 km south-west of Wabush, while the Fire Lake Project is about 68 km further south. Both mines have a combined production of 26 Mt of concentrate per year.

23.3 Iron Ore Company of Canada

Approximately 15 km north of the Scully Mine is the Iron Ore Company of Canada (“IOC”); its major shareholder is the international mining group Rio Tinto. The mine currently has a production rate of 22 Mt of concentrate per year and its latest open pit Moss (formerly Wabush 3), began producing ore in the second half of 2018. Ore is sent to a concentrator for upgrading to 65-67% iron. Production in 2020 was 17.7 Mt of concentrate including 9.6 Mt of pellets. Guidance for 2021 is between 17.9 Mt to 20.4 Mt of concentrates. Mineral reserves are estimated at 1.3 Bt of crude ore at 38% Fe.

Figure 23.1: Project Location



24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Plan

24.1.1 Execution Strategy

The execution strategy for the process improvement ("Pathway") of the Scully Mine consists of a self-managed execution with a combination of lump sum contracts. The Tacora self-performed organization has been an integrated team of managers, staff, tradesmen and operators supplemented with outside resources.

This strategy selected because of the following considerations:

- The fast track nature of the project with the objective to have the Scully Mine facilities fully ramped up by December 2021.
- Brownfield environment where construction and operations activities would coexist.
- The scope of work only required a light project structure.
- Limited staffing of the Tacora team during the execution phase.

The Self-Performed strategy aims to minimize total project cost and optimize utilization of the current and new-hire Tacora personnel, providing a seamless transition from process improvement commissioning to sustained operations.

The following provides an overview of the execution strategy for the two major areas:

- **Process Plant:** Lump sum contracts for the majority of the improvement projects would be managed directly by the Tacora site management team.
- **Mine:** mine development, operation, and procurement of mining equipment and dewatering, would be executed by Tacora operating personnel.

The self-performing execution strategy was selected to optimize the Project cost by reducing overhead inherent to standard outsourced execution platforms. It was also appropriate considering the small scope of work. It enabled expeditious team formation by eliminating the process of formal requests for proposal of Engineering, Procurement, and Construction Management ("EPCM") services.

24.1.2 Detailed Engineering and Procurement

Early in the detailed engineering phase, Technical Services issued Requests for Information (“RFI’s”) for critical equipment and materials as well as for lump sum packages to be processed by procurement so that Requests for Quotation (“RFQ’s”) were issued shortly thereafter.

To meet the project schedule, partial commitments to vendors was necessary to obtain and incorporate vendors’ data (namely drawings, specifications, structural and electrical loads, etc.) in detailed engineering.

24.1.3 Construction

Construction activities benefited from the existing installations on site.

- Site access were monitored and controlled using the existing gates and fencing.
- The existing medical facilities were used for all issues pertaining to health and injuries.
- The existing services contracts on site were maintained (road maintenance, waste management, regular maintenance of facilities).
- The existing power distribution was used for construction.
- Existing overhead cranes and hoists were certified and used by contractors, as necessary.
- Existing plant warehouse and laydown areas, and material handling equipment, were available for construction materials and equipment handling and storage.
- Existing heavy and light vehicles workshops were made available for the maintenance and repair of vehicles.
- Existing plant roads and walkways were used for in-plant movement.

Secure perimeter demarcation addressed access and permitting requirements as well as Environment, Safety & Health (“ES&H”) management protocols between construction activities and parallel operations activities.

Construction included demolition and dismantling, concrete placement, steel erection, plate work installation, mechanical equipment and piping installation along with the installation of electrical, instrumentation and control.

Section 21.1.1 lists the improvement projects and related capital expenditures completed in 2019-2020 and the remaining projects to be finalized in 2021 and 2022.

24.1.4 Commissioning and Ramp-up

Commissioning is being conducted by the Tacora team. Each project has a Project Manager (“PM”) and a sponsor. The PM is responsible for the project implementation, commissioning, and handover to operations

The commissioning activities flowing from mechanical completion include the development of performance testing criteria combined and test programs to be applied during the Dry Commissioning and Wet Commissioning phases. Tacora’s operations are responsible for the start-up of the plant improvement projects as well as for ramp up.

24.2 Plant Maintenance Program

An important part of the improved plant availability is based on the reduced downtime of the production lines. During past operations, it was frequent that a line was stopped before the planned shutdown. This was caused by a combination of equipment degradation and misaligned preventive maintenance (“PM”) schedules and manpower shortage.

Tacora has developed new PM schedules that combine the maintenance of all equipment that is part of the same line. For example, each mill line will be set on a 2,900 hours running schedule after which it will be shut down and maintenance on all equipment on the line will be performed. In particular, the relining of the mill feed chute was considered as a major contributor to the duration of the shutdown, and can be reduced to improve utilization. With the planned modifications during the upgrade projects, this equipment will require significantly less maintenance. In addition, Tacora will develop work procedures based on their experience in other similar plants to safely and efficiently perform these tasks. The same philosophy will be applied to all critical lines or equipment of the plant. Tacora is using the Ellipse asset management system and will implement best practices around predictive maintenance.

To address the issue of manpower availability, Tacora has included in the agreement reached with the United Steelworkers a clause that allows for, when required, bringing in contractors as supplemental manpower to perform maintenance tasks. In addition, the compensation structure within this agreement is linked with safety, quality production and shipment of concentrate. It is believed that this will encourage employees to bring forward problems and improvements. Tacora also promotes a work culture that empowers decision making and continuous improvements. The flat management structure is organized in a way that managers are close to the operating and maintenance personnel which allows for a quick follow-up on suggestions made or issues raised.

24.3 Risk Management

There was a risk review of the current operational status. The objective of the review was to identify any risks and opportunities as the operation was ramping up in addition to removing old risks which had been mitigated. Each item was evaluated for its severity, likelihood and potential mitigation measures or follow-up actions.

24.3.1 Methodology

The procedure consisted in listing the risks identified by the team and ranking the risks using the Probability and Impact Diagrams ("PID") shown in Figure 24.1. Both risks and opportunities (or positive risks) were identified and ranked during the session.

Mitigation actions were identified for each risk whenever possible.

Figure 24.1: Probability Impact Diagram

Probability Impact Diagram (Risks)

				Consequence				
				Insignificant 1	Minor 2	Moderate 3	Major 4	Catastrophic 5
Likelihood	A	Almost Certain	95%	A1	A2	A3	A4	A5
	B	Likely	80%	B1	B2	B3	B4	B5
	C	Moderate	50%	C1	C2	C3	C4	C5
	D	Unlikely	20%	D1	D2	D3	D4	D5
	E	Rare	5%	E1	E2	E3	E4	E5

Low	Moderate	High	Extreme

Probability Impact Diagram (Opportunities)

				Opportunity Potential				
				Insignificant 1	Minor 2	Moderate 3	Major 4	Outstanding 5
Likelihood	A	Almost Certain	95%	A1 (O)	A2 (O)	A3 (O)	A4 (O)	A5 (O)
	B	Likely	80%	B1 (O)	B2 (O)	B3 (O)	B4 (O)	B5 (O)
	C	Moderate	50%	C1 (O)	C2 (O)	C3 (O)	C4 (O)	C5 (O)
	D	Unlikely	20%	D1 (O)	D2 (O)	D3 (O)	D4 (O)	D5 (O)
	E	Rare	5%	E1 (O)	E2 (O)	E3 (O)	E4 (O)	E5 (O)

Low	Moderate	High	Significant

The combination of risk consequence and risk likelihood results in an overall risk ranking as per the colored section of the risk matrix. The risk consequence and opportunity potential were evaluated as per Table 24.1.

Table 24.1: Consequence and Opportunity Potential

Technical Consequences Scale

	Consequences				
	1 - Insignificant	**2 - Minor**	**3 - Moderate**	**4 - Major**	**5 - Catastrophic**
Safety and Health	**First Aid Case**	**Minor Injury,** Medical Treatment Case with/or Restricted Work Case.	**Serious Injury** or Lost Work Case	**Major or Multiple Injuries -** permanent injury or disability	**Single or multiple Fatalities**
Environment	**No impact** on baseline environment. Localised to point source. No recovery	**Localised** within site boundaries. Recovery measurable within 1 month of impact.	**Moderate** harm with possible side effect. Recovery in 1 year.	**Significant** harm with possible wider effect. Recovery in 1 year.	**Significant** harm with widespread effect. Recovery longer than 1 year.
Production / Schedule	< 1 week	1 week - 1 month	1 - 3 months	3 - 6 months	> 6 months
Reputation	Localised temporary impact	Localised, short term impact	Localised, long term impact but manageable	Localised, long term impact with unmanageable outcomes	Long term regional impact
Business Impact	Impact that can be absorbed through normal activity.	An adverse event which can be absorbed with some management effort.	As serious event which requires additional management effort.	A critical event which requires extraordinary management effort.	Disaster with potential to lease to the collapse of the project.

Financial Consequences Scale

	Consequences				
	1 - Insignificant	**2 - Minor**	**3 - Moderate**	**4 - Major**	**5 - Catastrophic**
CAPEX	<$100k	$100k - $2m	$2m - $10m	$10m - $30m	>$30m
Financial NPV (USD) (Capital, Operational, Revenue)	<$100k	$100k - $2m	$2m - $10m	$10m - $30m	>$30m
Property Damage (USD)	<$10k	$10k - $200k	$200k - $2m	$2m - $20m	>$20m
OPEX ($45/tonne)	< $0.1 / tonne	$0.1 - $1 / tonne	$1 - $5 / tonne	$5 - $10 / tonne	> $10 / tonne
Schedule	< 1 month	1 - 3 months	3 - 6 months	6 - 12 months	> 12 months

24.3.2 Results

A total of 24 risks and 7 opportunities were identified.

Ideally, the mitigated risks would all be low or moderate, but all projects are bound to include certain risks which have a "high" or "major" impact level, even after mitigation.

The risk maps in Figure 24.2 show the number of risks per severity before and after mitigation.

Figure 24.2: Risk Map

Risk Map, Before Mitigation

Likelihood		Consequence: Insignificant	Minor	Moderate	Major	Catastrophic
Almost Certain	95%	0	2	0	0	0
Likely	80%	0	0	8	0	1
Moderate	50%	0	2	1	1	0
Unlikely	20%	0	0	4	1	2
Rare	5%	0	0	1	1	0

Number of Risks per Severity	
Extreme	4
High	13
Moderate	7
Low	0
Total:	24

Risk Map, After Mitigation

Likelihood		Consequence: Insignificant	Minor	Moderate	Major	Catastrophic
Almost Certain	95%	0	0	0	0	0
Likely	80%	0	0	3	0	0
Moderate	50%	1	1	0	0	0
Unlikely	20%	2	8	3	1	0
Rare	5%	0	2	2	0	1

Number of Risks per Severity	
Extreme	0
High	5
Moderate	6
Low	13
Total:	24

Opportunity Map

Likelihood		Consequence: Insignificant	Minor	Moderate	Major	Outstanding
Almost Certain	95%	0	0	0	0	1
Likely	80%	0	1	0	0	0
Moderate	50%	0	0	1	1	1
Unlikely	20%	0	1	1	0	0
Rare	5%	0	0	0	0	0

Number of Risks per Severity	
Outstanding	3
Major	2
Moderate	1
Low	1
Total:	7

Table 24.2 lists the “Extreme” risks and their associated mitigation plan.

Table 24.2:Extreme Risks

#	Assessment	Description
1	B5	Plant Structural Repairs
Cause		
The most recent structural audit by BBA in 2012 has been considered and addressed, however it is important to get a more current assessment of the plant conditions.		
Effect		
Identification of new repairs could increase sustaining capital cost.		
Action Item		
A detailed structural assessment must be undertaken, which may result in additional necessary repairs and associated costs.		
#	**Assessment**	**Description**
13	C4	Blasting in Boot Pit
Cause		
The location of the Boot Pit, an ore resources part of the mine plan, is in proximity of structures with possible impact to their integrity.		
Effect		
It is the responsibility of Tacora to repair or mitigate damage to those structures.		
Action Item		
Retain specialist to reduce vibration. Reducing the size of the blast. Structural integrity monitoring of bridges.		
#	**Assessment**	**Description**
16	D5	Tailing Dams Slope Stability
Cause		
Erroneous slope stability analysis and poor construction practises.		
Effect		
Potential loss of life in the event of failure for workers in area or downstream of rupture. Significant loss of habitat for fisheries and potential destruction of wetland areas High production downtime. High reputational cost. Fines and legal liability.		
Action Item		
Slope stability analysis regular inspection by highly competent experts and proper construction practices.		
#	**Assessment**	**Description**
7	D5	Tailings Deposition Amendment
Cause		
The tailings deposition schedule relies on amendments to the Fisheries Act, Schedule 2, which if not granted would put the business in violation of regulatory compliance.		
Effect		
Production could be shut down or possible fines.		
Action Item		
Permit process in progress.		

Table 24.3 lists the "Outstanding" opportunities and their associated next steps.

Table 24.3: Outstanding Opportunities

#	Assessment	Description
18	**A5**	**Conversion of inferred resources**
Cause		
• Based on the resources classification there are over 200 Mt of ore that can be converted into reserves if properly drilled and tested.		
Effect		
• Increase mine life or throughput capacity.		
Action Item		
• Definition drilling program being carried out in 2021 and 2022.		
#	**Assessment**	**Description**
29	**C5**	**Tailings Reprocessing**
Cause		
• Over the life of the Scully mine, more than 400 Mt of high-quality tails have been deposited in the TIA.		
Effect		
• Mining and processing of the tailings would increase concentrate production. This could also increase iron recovery of the current concentrator by scavenging our current tails before sending them to the TIA. This will also favorably affect the tailings deposition plan.		
Action Item		
• Drilling and testing in progress to establish proof of concept.		
#	**Assessment**	**Description**
31	**C4**	**Super Mill Implementation & Expansion**
Cause		
• Desire for increasing mill utilization and throughput beyond the current plan.		
Effect		
• Increase concentrate capacity.		
Action Item		
• Concept study to move toward PFS level Engineering in 2021 & 2022.		

24.3.3 Conclusions

It is recommended that, as a responsible operator, Tacora continues to engage in additional risk review sessions to prevent liability to the business.

Additionally, it should be recognized that there are many high potential opportunities available to increase the value of the organization.

25 INTERPRETATION AND CONCLUSIONS

25.1 Geology and Mineral Resources

- The geological interpretation for the Scully Deposit is based primarily on diamond drilling data and 2D sectional interpretations by representatives of Cliffs in 2014. The geology of the deposit is well understood, however some significant gaps in the drilling grid exist in areas planned for near-term production.
- The mineralization is found in three main members (Upper, Middle and Lower) of the Sokoman Formation. Each member is subdivided into sub-units, which present different ore mineralogy, recoveries, and contaminant levels. The mineralization controls of the deposit are also well understood.
- The protocols followed to collect sample data post-2002 are considered sufficient for NI 43-101 purposes. No evidence of established protocols was found for drilling prior to 2002; therefore, confidence in this information is lower. Sampling has been undertaken based on geological logging and is adequate for the mineralization style and size of the deposit.
- QA/QC samples submitted as part of the reanalysis program returned values within expectations. No QA/QC data was available for historical drilling (all drilling pre-2014), and all historical analyses were undertaken by the on-site laboratory. The laboratory was not certified. G Mining Services Inc. ("GMS") considers this insufficient, however Scully has been a long-term past producer of iron ore concentrate and has demonstrated the ability to reconcile grade with the geological model. GMS outlined considerable recommendations in Section 26 to improve confidence in the drilling and sampling database.
- Réjean Sirois, P. Eng., from GMS, supervised re-sampling of drill core during the site visit on August 17-18th, 2017 to validate the grades of the assays in the drilling database. GMS believes the check assay results are sufficient, however a relative reduction of 7% was noted in the reanalyses for Fe%.
- The geological model includes a total of 13 lithology units which were modelled using the provided 2D sectional interpretations and the coded drill hole database. 3D wireframe solids were produced in Leapfrog™ and are representative of the folded lithologies present in the Scully Deposit.
- Each bedding orientation is appropriately defined within the 14 structural domains dividing the geological model.
- Mineral Resources were estimated in the mineralization and structural domains, using Geovia GEMS™ from 6.0 m long composites using four ordinary kriging interpolation passes. Blastholes were included in a small initial pass, and the remaining three passes only used exploration drill

holes. Each search ellipse was incrementally larger than the previous and was based on variogram ranges and the drill holes spacing.

- The performance of the block model to predict resource estimates was evaluated through global and local validation methods, including visual comparisons, descriptive statistics, swath plots, Q:Q plots and blasthole data comparisons. Only limited production data were available to validate the accuracy of the model to true known grade. Block grades were found to reproduce composite grades sufficiently in the block model.
- The Mineral Resources are reported within a Lerchs-Grossman open pit shell and are effective as of January 1st, 2021, using a minimum Lab Weight Recovery (Wilfley Table results) of 10% and a long-term iron concentrate price of USD 94/t as follows:
  - Open Pit Measured and Indicated Mineral Resources total 721.9 Mt at an average grade of 34.65% Fe.
  - Open Pit Inferred Mineral Resources total 263.4 Mt at an average grade of 34.1% Fe.
- Mineral Resources were classified into Measured, Indicated, and Inferred categories according to the CIM Definition Standards on Mineral Resources and Mineral Reserves as adopted in NI 43-101.
- A compilation of the mineralized blocks generated a graph of cumulative tonnage for various iron grades. It confirmed that the iron mineralization is relatively uniform and entirely contained in the mineralized members. The graph confirms that there is very limited ore tonnage below 28% Fe (Table 14.22).

25.2 Mining and Mineral Reserves

- Open pit optimization was conducted using Whittle software to determine the optimal economic shape of the open pit to guide the final pit design process. Pit optimizations use a long-term Benchmark Iron Ore Price 62% Fe CRF China of USD 70/t of concentrate; after taking into account premiums, marketing fees and ocean freight, the Benchmark Price translates to USD 64.93/t of concentrate FOB Pointe Noire for a 65.6% iron ore concentrate.
- The mine design and Mineral Reserve Estimate have been completed to a level appropriate for feasibility studies. Definitions for Mineral Reserve categories used in this Report are consistent with the CIM definitions as adopted by NI 43-101. Historically, the Mineral Reserve Estimate was based on a minimum Lab Weight Recovery of 27% for all sub-units except sub-unit 52 at 30%. In addition, sub-unit 34 has a ratio of weight recovery to iron of at least 1. Another compilation of the mineralized blocks generated a graph of cumulative tonnage for various Lab Weight Recovery (Table 14.21). It can be observed that minimal tonnage is added with a Wet Recovery below 18%. However, almost

100 Mt of reserves could be added if the minimum Wet Recovery is reduced from 27% to 20%. It was subsequently calculated (Table 15.6) that ore with a minimum 20% Wet Recovery still generated a positive cash flow and should be included in mineral reserves.

- Proven and Probable Mineral Reserves are estimated to be 478.9 Mt at an average grade of 34.89% iron and 2.62% manganese for 159.4 Mt of iron concentrate at 65.6% Fe.
- An owner mining approach with conventional open pit mining techniques is planned on 12 m benches with a final 24 m bench height excavated in most pit walls. Bench face angles, and berm widths are variable by domain. The bench face angles are designed to follow the bedding plane for Domain 3 in the hanging wall. The inter-ramp angles range from 32 to 46 degrees.
- The current equipment hours were considered in the replacement schedule of the mine equipment; and the value of the current fleet was considered as sunk costs for this Feasibility Study Update.
- Loading in the pit uses a fleet of hydraulic shovels consisting of one Komatsu PC4000-6 (diesel), one Komatsu PC5500-6E (electric), one Komatsu PC5500-6 (diesel) and a Caterpillar 994 front-end loader. The loading fleet is matched with 240-t Komatsu 830E mining trucks. The fleet reaches a total of 15 units in 2023.
- Production drilling is accomplished with two electrical rotary blasthole rigs. The blasthole diameter is 349 mm (13 ¾ in.) with a 9.25 m x 9.25 m pattern for a targeted powder factor of 0.21 kg/t in waste and 0.30 kg/t in ore. No pre-split drilling is planned due to the incompetent nature of the rock mass.
- The Tacora open pit has been divided into five sectors referred to as East Pit, Center Pit East, Center Pit West, Center Pit South and Boot Pit. The final pit design consists of pushbacks to the existing pit limits and a deepening of the current excavation. Most of the tonnage remaining to be mined is from the Center Pit West (34%), Center Pit South (34%) and Center Pit East (13%). Dewatering of the pits will be undertaken in advance of mining with a combination of well fields on the west side of the pit and in-pit sumps.
- Waste rock will be stored in several waste dumps around the pit limits including the North Dump, NW Dump, SW Dump and South Dump. In addition, an in-pit dump is planned in the West Extension to reduce haulage costs and footprint impact.
- The LOM plan details 27 years of production (2021 to 2047) which assumes one year of continuing ramp-up before achieving the target production rate of 6 Mt of concentrate per year.
- The peak mining rate will be approximately 36 Mt with a peak of 37.4 Mt moved. The average mining rate is 32.2 Mtpy over the LOM.

- The open pit generates 390.9 Mt of overburden and waste rock for a strip ratio of 0.82:1 over the LOM with 18.8 Mt of ore transiting through stockpiles for blending purposes and to balance mining and milling constraints. A total of 868.8 Mt is mined from the pits.

25.3 Processing

- The concentrator has operated for about 50 years; consequently, the process for treating the ore is proven and considered robust.
- The process plant upgrades have for objective to produce 6 Mt of concentrate per year from a maximum of 18 Mt of ore feed to the plant.
- Mill upgrades are planned to increase plant reliability and availability. These improvements include replacement of the gears on two mills, replacement of the mill lube systems, replacement of the mill starters and relays and improvement of the mill recirculation line and pump type.
- The operation philosophy for the autogenous (“AG”) mills will be modified to utilize the motor power to its full capacity in order to reach the targeted throughput.
- Scavenger spirals are to be added to increase the plant wet recovery and the high-tension scavenger circuit is to be restarted to improve dry recovery.
- Efficient preventive maintenance and the use of contractors to assist in major planned shutdowns are considered key factors to improve plant availability.

25.4 Transportation

- The concentrate storage capacity will be increased to allow the loading of 168 car trains compared to the original 124 cars operation. This will include additional buffer storage of dry concentrate.
- Concentrate will be transported by train to Pointe-Noire where it will be loaded onto ships.
- Rail haulage is contracted out to Quebec North Shore and Labrador Railway (“QNS&L”) up to Sept-Iles. From there, trains are transported to Pointe-Noire by Chemin de Fer Arnaud. Train unloading and ship loading is contracted out to *Société ferroviaire et portuaire de Pointe-Noire* (“SFPPN”). This operation will use the same unloading facilities as the original operations. SFPPN will be responsible for the refurbishment of the existing equipment to meet required throughput.
- Contracts with providers are in place for transportation services.

25.5 Environmental and Permitting

The reactivation project was released from further environmental assessment in November 2017. Following the environmental assessment release, numerous approvals, permits, and authorizations required from municipal, provincial, and federal regulators, were obtained or amended prior to reactivation.

The reactivation of the mine and associated deposition of mine tailings to the end of the mine life in 2047 will result in an expansion of the area affected by tailings and this will require an amendment to Schedule 2 of the Metal and Diamond Mining Effluent Regulations ("MDMER") under the Fisheries Act. This amendment will generate a requirement for an environmental assessment ("EA"). Continued tailings deposition to ELOM 2047 will impact several lakes and streams in the Flora Basins watershed and Tacora will be required to offset the fish habitat losses associated with these waterbodies. The expanded tailings deposition will affect approximately 360.6 units (100 m^2) of stream habitat and 50 ha of lake habitat. Tacora is currently exploring the technical and financial feasibility options to offset the habitat losses. Tacora will be required to complete a fish offset plan, acceptable to Fisheries and Oceans Canada ("DFO"), and issue Letters of Credit to cover the plan's implementation and monitoring costs as required under the Fisheries Act. Tacora will need to complete the fish habitat offsetting prior to any habitat losses associated with continued tailings deposition.

Revised Development and Rehabilitation and Closure Plans ("DP & RCP") were submitted to IET in 2017 / 2018 and approved in 2018. A proponent is required to submit an updated DP and RCP every five years, or if there is a significant change to the project, therefore, updates will be required to these documents in 2023. Tacora is also required to provide a financial assurance covering the estimated closure costs outlined in the RCP.

Since re-activation in 2019, Tacora continues to comply with numerous conditions and commitments of approval, and various standards contained in federal and provincial legislation, regulations and guidelines. These conditions and commitments will need to be considered during any remaining design, construction, operations and closure phases of the mine.

25.5.1 Potential Environmental Risks

Potential Environmental Risks to the Project are noted below:

Stack Emission Testing

Under the Certificate of Approval ("CoA"), Tacora is currently required to complete stack emission testing and dispersion modelling every two years. Upon demonstrating that the Mine meets the guidelines, the testing and modelling can be completed every four years. Tacora's Environmental Assessment Registration document committed to completing stack testing within 12 months of resuming operations which was June 2019. Stack testing has yet to be completed. A request was submitted to the Newfoundland & Labrador Department of Environment and Climate Change ("NL DECC") in December 2020 to postpone the first round of stack testing until Spring 2021. This is to allow improvement to the APC systems in advance of stack testing. The last round of stack testing was completed at the Mine between September 2011 and January 2012. Results from this test program were used in a dispersion model, with results reported in January 2014. The modelling report identified that the Mine was in exceedance for ground-level dust concentrations.

To meet the Approval requirements, source emission testing will be completed in 2021 and every subsequent two years if in noncompliance; however, if in compliance this frequency is reduced to every four years. These results will be used for air dispersion modelling of the site for submission to the NL government. This submission will support the ongoing requirements for demonstration of compliance with provincial air emissions regulations.

Schedule 2 Amendment

To continue mining activities to 2047 an expansion of the tailings impoundment area is required. To expand the tailings storage outside of the existing permitted tailings basin, an amendment is required to the existing Schedule 2 regulation. The Schedule 2 amendment process involves regulatory approval from provincial and federal agencies and engagement with Indigenous groups. Due to the scale of the project and large number of sequential approval steps involved, there is a risk that the Project will be delayed, or alternative solutions explored.

To better position for approval, Tacora has assembled an experienced multifaceted team that is engaging in dialog with regulators. The team will prepare an environmental assessment registration and complete robust field studies in 2021 that will provide key information on how to best structure the project moving forward. The team will also review past 'assessments of alternatives' to determine if other tailings disposal options are available and if new assessments are required. The process for achieving approval may take in excess of 3 years. The tailings deposition plan could be adjusted to accommodate possibly for an additional one year of storage should the approval be delayed.

25.5.2 Potential Environmental Opportunities

Potential Environmental Opportunities to the Project are noted below:

Reducing the Frequency of Testing for a Parameter in the Effluent Discharge Criteria ("EDC")

To ensure compliance with Provincial and Federal Regulations, Tacora has an Effluent Monitoring Program. The Effluent Monitoring Program includes effluent sampling, analysis and reporting of results to both the NL DECC as well as Environment and Climate Change Canada ("ECCC").

Both the CoA and the MDMER present the opportunity to reduce the frequency of testing for certain parameters. Tacora will implement reduced testing when possible for deleterious substances that meet the reduction criteria. An effort will be made to reduce testing for arsenic, copper, lead, nickel, zinc, radium 226 and Acute Lethality Testing ("ALT's") at all final discharge points.

Reduce Water Withdrawal from Little Wabush

Tacora currently has two Water Use Licences ("WUL") that permit water withdrawal from Little Wabush Lake and dewatering from the mine pits. Both licences require annual reporting of water withdrawal and usage amounts which then presents a cost to the business as Tacora is charged under the Water Use Charge Regulations NLR 60/16.

Tacora plans to implement a water recycling project which will allow water from the mine pits to be used as process water. The project will reduce both the amount of water withdrawn from Little Wabush Lake and the amount of water discharged into various receiving water bodies. Tacora will be able to recycle water while reducing WUL fees.

Habitat Banking

Tacora will be required to offset fish habitat losses as result of tailings expansion to support continued mining. DFO administers the Fisheries Act to ensure fish habitat offsetting is achieved. Recent revisions to the Fisheries Act have expanded fish habitat banking as a viable option for offsetting. Fish habitat banking is considered a preferred method for offsetting because positive environmental outcomes are realized in advance of any habitat loss. There will be an opportunity for Tacora, when offsetting lost fish habitat due to tailings expansion, to develop a fish habitat bank to offset future habitat losses. The banked habitat would provide Tacora with certainty with respect to regulatory compliance and possibly provide economic benefits by creating the additional habitat at reduced cost.

25.6 Tailings Storage Facility and Dams

- The existing storage facility will be used with an estimated capacity sufficient for 3 years before Schedule 2 permit limits are likely exceeded on the South Flora Lake level and associated three tributary flooding extents. As this capacity is depleted, planning and construction of new dikes and additional pumping capacity will be required.
- Current capacity is available without needing to directly discharge tailings into North Flora Lake
- To meet life of mine targets for tailings impoundment, the storage capacity of South Flora Lake and part of its surrounding watershed is required.

25.7 Electrical, Instrumentation and Controls

- Mill Main Motor 5 kV MCC and steam plant 600 V MCC shall be upgraded to improved equipment reliability.
- Many instruments need to be replaced or added to increase robustness and precision of the control system.
- Old PLC5 at steam plant must be replaced by a new ControlLogix PLC.

25.8 Project Economics

When compared to the global suppliers of iron ore, the Scully Mine will always present a cash operating cost above average for the following fundamental reasons:

- Nature of the ore and its grade require concentration with inherent Fe losses and additional costs
- Relatively higher stripping ratio
- Costs of rail transportation and port operations owned and supplied by third parties
- Higher ocean freight due to a larger distance to the main Asian customers.

On the positive side, the Scully Mine benefits from the following:

- Large resources and reserves
- High concentrate quality that results in significant premium from customers
- A stable political and social environment.

The cash operating revenues and costs based on long-term estimates for the Scully operations are as follows:

Table 25.1: Cash Operating Revenues & Costs on Long-term Estimates

	CAD/t of Concentrate	USD/t of Concentrate
Benchmark 62% Fe China	87.50	70
Premium-Marketing	15.00	12
Ocean Freight	(21.34)	(17.07)
Concentrate Losses – Pointe Noire	(0.81)	(0.65)
Rail Transportation & Port	(19.60)	(15.68)
Royalties	(6.33)	(5.06)
Scully Mine Operating Cost	(31.66)	(25.33)
Operating Cash Flow	**22.76**	**18.21**

Total Cash Operating Costs are USD 63.79 /t of concentrate (CAD 79.74/t) or 77% of revenues; only 37% of the Total Cash Operating Costs are under Tacora's direct control. Obviously, Tacora must realize excellence in its operations and demonstrate discipline in its sustaining capital to remain in business. Tacora's capability to maintain its assets and meet and exceed its targeted performance will be critical to its financial future.

26 RECOMMENDATIONS

The following recommendations are put forward for the successful ramp-up to forecasted production rates of over 6 Mt of concentrate per annum.

26.1 Geology and Mineral Resources

For all future drilling and sampling campaigns (including blasthole drilling), implementation of a comprehensive QA/QC protocol including the insertion of blanks, certified reference material (CRM standards) and field duplicates regularly into the sample stream is recommended. In addition, external umpire and check assaying should be included in the procedure.

The XRF analysis methodology should be reviewed by an external expert. The fused-bead methodology used in the past uses an extremely small sample size considering the composite lengths; for each 5 – 10 m composite of NQ drill core, only 0.8 g of pulverized sample is used for analysis. G Mining Services Inc. ("GMS") recommends adjusting the methodology to use a larger sample size for pressed pellet, which generally uses between 8 and 15 g of pulverized material (depending on pellet diameter). In addition, manual laboratory data collection systems used in the past should be replaced by an automated system which reduces the chance of human-induced errors.

Specific gravity readings should be taken regularly on drill core, and downhole density logs are acquired for new drilling. This will assist significantly in characterizing the ore units which present variable porosity and mineralogy and in creating a new bulk density model for mine planning purposes.

All exploration and blasthole sampling and logging data should be stored in a comprehensive, centralized, fit-for-purpose drilling database which is exclusively managed by a geological database administrator. This system should reduce the amount of manual data entry and result in better database integrity. These systems can commonly be linked to logging packages that will further assist with accurate data collection.

GMS recommends that Tacora Resources Inc. ("Tacora") conduct a drilling program to convert the Inferred Mineral Resources that are contained in the current mine plan into Indicated category. Initial estimates suggests that 20,000 m of drilling is required to achieve this goal.

Table 26.1: Cost of Infill Drilling Campaign

Item	Cost
Drilling Program (20,000 m, 160 Drill Holes)	$4,000,000
Sampling and Assaying (XRF And Limited Shaker Table Test Work)	$500,000
Personnel, Vehicle Rental and Other	$300,000
Total	**$4,800,000**

Regarding the resource estimation, GMS recommends that the Scully Deposit be unfolded for the purposes of variography, so more accurate ranges of the variogram can be determined. In addition, GMS recommends using a variable search ellipse orientation during resource estimation (dynamic anisotropy) to avoid the use of structural domains and using a sub-blocked model to simplify the block modelling process.

26.2 Mining and Mineral Reserves

The resources are compared to mineral reserves in Table 26.2. Obviously, there is a large potential to increase reserves beyond the current level. In addition, pit modeling for resources and reserves estimates were limited in space to provide buffers to rail loop, primary crushers, and lakes. Sensitivity analyses should be conducted to determine if the value of additional resources/reserves could justify the removal and / or relocation of some of the obstacles. In any case, the location of the resources outside of the reserves envelopes should be identified to guide additional exploration and drilling programs and aim to complete total reserves for the Scully Deposit.

Table 26.2: Comparison Resources and Reserves

	Resources M&I	Resources Inferred	Reserves
Tonnage (Mt)	721.9	263.4	478.9
Fe Grade (%)	34.7	34.1	34.89
Mn Grade (%)	2.4	2.1	2.62
WTREC (%)	35.8	34.0	33.29

The additional and final exploration program should be combined with additional hydrology, hydrogeology and geotechnical studies to allow for a final pit design. These final reserves and related pit designs would allow to properly plan the locations of waste dumps (both outside and inside depleted pits) and tailings disposal facilities.

More specifically, additional hydrogeological investigations are recommended for the East, South and Boot Pits to investigate groundwater infiltration. It should include the following:

- Start a groundwater level monitoring program.
- Drill and conduct hydraulic tests at Boot Pit, South Pit, East-Pit-East and East-Pit-West.
- Validate the perched condition hypothesis for Knoll and Vern Lakes.
- Carry out additional geophysical surveys in the Duley and Little Wabush Lakes areas.
- Update the groundwater model with the data obtained during the field operations.

Additional waste rock storage options should be investigated. In the event of expanded larger open pit limits optimized for higher iron ore prices additional waste dump storage capacity will be required and may limit or defer the possibilities of in-pit waste storage.

Recommendations were made for further work to gain confidence in pit slope design. Additional work would include:

- Drilling geotechnical holes and the installation of wire piezometers to verify pore pressure buildup.
- Carrying out uniaxial compressive strength tests on samples from geotechnical holes for all geological units.
- Carrying out 4 direct shear tests along bedding planes.
- Review variation in unit quality related to degree of weathering.
- Gather structural data and review potential kinematic failures on the hanging walls and end walls.
- Install an inclinometer in the east wall of the East Pit East.

26.3 Processing

Implement a control system to operate the autogenous (“AG”) mills while maximizing power utilization in order to prioritized mill throughput. Make sure that the operating staff understands and accepts the new mill operating culture aiming at maximizing the throughput rather than relaxed operation.

Optimize the Mill preventive maintenance should be optimized particularly for mill relining to reduce plant downtime. Plant shutdown should be planned for the optimal use of the internal and external resources

Monitor the performance of the MRC in order to verify if the recovery is negatively affected by the increased throughput and evaluate the possibility of adding new manganese lines.

Assess solutions for increasing the percent solids in the AG mills, namely by modifying the current screen oversize recirculating system. Increasing the percent solids in the mills will allow increasing the grinding efficiency.

26.4 Tailings Storage Facility and Dikes

Monitor the South Flora Lake level for compliance within existing Schedule 2 permit requirements.

Obtain regulatory approvals for the amendment of existing Metal and Diamond Mining Effluent Regulations (“MDMER”) Schedule 2 permit as well as surface leases to extend the current Tailings Impoundment Area (“TIA”) limits and accommodate the storage of the life of mine tailings production.

Implement and optimize method of multiple spigot discharge operation to improve vertical storage capacity of available tailings impoundment area and avoid the tailings deposit from interrupting or impeding the flows of Diversion Channel for drainage of the Flora watershed.

Drone aerial topographic survey to measure performance of tailings beach development (deposit slope profile and density) and to video inspect potential obstructions or constrictions to the flows in the natural drainage path of the Diversion Channel. Repeat on a yearly basis following the spring freshet. Review tailings deposition plan and update volume capacity projections with actual tailings data when appropriate.

Develop, plan and carry out the geotechnical field investigation programs, tailings characterization laboratory program and engineering studies to support the construction of the southern extension of the tailings impoundment area. Planning should take into consideration the limited duration of construction season.

26.5 Plant Structures and Foundations

A detailed structural inspection must be undertaken for all plant structures and foundations to ensure their integrity and that their capacity is adequate. The previous structural audit done by BBA in 2012, classified several items as critical (Risk Rating 4). These were addressed prior to start-up, however, many of the less urgent items were not repaired / completed. Any damage / corrosion which may have occurred since the 2012 audit should also be assessed.

26.6 Electrical, Instrumentation and Controls

Ongoing capital upgrades to MCC's need to be sustained over the next several years as outlined in the capital plan. Lighting upgrades in various areas of the plant are required along with replacement of selected scales, flowmeters and control valves.

The implementation of density and flow monitoring of tailings lines is required to better control the plant water balance and to better utilize tailings pumping infrastructure.

Finally, Advanced Process Control of the mills needs to be implemented to support the maximum use of mill power capacity.

26.7 Plant Maintenance

As a significant part of the improved availability is based on reduced downtime of equipment. Over the life of the mine Tacora must continuously identify and perform the required maintenance as well as monitor the efficiency of the maintenance tasks.

26.8 Environmental and Permitting

The mine has continued tailings deposition into Flora Lake however a planned expansion to the Tailings Impoundment Area ("TIA") will require an amendment to Schedule 2 of the Metal and Diamond Mining Effluent Regulations ("MDMER"), which will generate the requirement for a series of associated regulatory approvals, and baseline studies. Approvals for the expanded TIA will be required when the tailings reach their current authorized storage capacity, therefore, a focus on the permitting process for this expansion should continue as a high priority component.

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28 ABBREVIATIONS

Abbreviations	Full Description
ACM	Asbestos Containing Materials
APC	Air Pollution Control
CAD	Canadian Dollar
CAGR	Compound Annual Growth Rate
CAPEX	Capital Expenditure
CIM	The Canadian Institute of Mining, Metallurgy and Petroleum
CFR	Cost and freight (CFR) - legal term used in international trade
CoA	Certificate of Approval
CoG	Cut-off Grade
dmt	Dry Metric Tonnes
DP	Development Plan
DFO	Department of Fisheries and Oceans
DWT	Dead Weight Tonnage
DNR	Department of Natural Resources
EA	Environmental Assessment
ECCC	Environment and Climate Change Canada
EDMS	Environmental Database Management System
EEM	Environmental Effects Monitoring
EEMER	Environmental Effects Monitoring Electronic Reporting
EPA	Environmental Protection Act
EPP	Environmental Protection Plan
ERP	Emergency Response Plan
ESA	Environmental Site Assessment
ETP	Effluent Treatment Plant
F	Degrees Fahrenheit
Fe	Iron
FEL	Front End Loader
FoB	Free on Board
FoS	Factors of Safety
FS	Feasibility Study
G	Giga – (000,000,000's)
g	Gram
g/cc	Density unit (Gram per cubic centimeter)

Abbreviations	Full Description
GDP	Gross Domestic Profit
gpt or g/t	Grams per tonne
g/L	Gram per liter
G&A	General & Administration
GMS	G Mining Services Inc.
GPS	Global Positioning System
h	Hour
h/d	Hours per day
h/y	Hours per year
h/wk	Hours per week
hp	Horsepower
Hz	Hertz
IAA	Impact Assessment Act
IAAC	Impact Assessment Agency of Canada
IBA	Impact Benefits Agreement
IET	Department of Industry, Energy and Technology
IRR	Internal Rate of Return
ITUM	Innu Takuaikan Uashat Mak Mani-Utenam
ISO	International Organization for Standardization
k	Kilo – (000's)
kg	Kilograms
kg/t	Kilograms per tonne
kV	Kilovolts
km	Kilometer
km/h	Kilometer per hour
kPa	Kilopascal
KPIs	Key Performance Indicators
kt	Kiloton
kW	Kilowatts
kWh	Kilowatts per hour
L	Liter
LOM	Life of Mine
LOC	Letters of Credit
M	Mega or Millions (000,000's)
m	Meter
masl	Meters above sea level
M&I	Measured and Indicated

Abbreviations	Full Description
MII	Measured Indicated and Inferred
MCC	Motor Control Center
MDMER	Metal and Diamond Mining Effluent Regulations
MERS	Mining Effluent Reporting System
m/min	Meter per minute
Mn	Manganese
m/s	Meter per second
m^2	Square meter
m^3	Cubic meter
mg	Milligram
mg/L	Milligram per liter
mm	Millimeter
ml	Milliliter
MLJ	Innu Nation of Matimekush-Lac John
min	Minute
Mo	Month
MOA	Memorandum of Agreement
Mt	Million tonnes
Mtpd	Million tonnes per day
Mtpy	Million tonnes per year
MVA	Megavolt-ampere
MW	Megawatt
NCC	NunatuKavut Community Council
NI 43-101	National Instruments 43-101- Canadian Standards of Disclosure for Mineral Projects
NOH	Net Operating Hours
NPI	Net Profit Interest
NPV	Net Present Value
OK	Ordinary Kriging Methodology
OPEX	Operating Expenditures
PEA	Preliminary Economic Assessment
PFS	Pre-feasibility Study
Pb	Lead
PM	Particulate Matter
PV	Present Value
QC	Province of Quebec
QP	Qualified Person

Abbreviations	Full Description
RAP	Remedial Action Plan
RC	Reverse Circulation
RCP	Rehabilitation and Closure Plan
RoM	Run-of-mine
rpm	Revolutions per minute
SAT	Saturation Magnetization Analyzer
Sec	Second (time)
STP	Sewage Treatment Plant
t	Tonnes (1,000 kg) (metric tonne)
t/y or tpy	Tonnes per year
t/d or tpd	Tonnes per day
t/h or tph	Tonnes per hour
TIA	Tailings Impoundment Area
t/m^3	Tonnes per cubic meter
tph	Metric tonnes per hour
TRS	Tailings Reclaim Sump
TSF	Tailings Storage Facility
TSP	Total Suspended Particulate
TSS	Total Suspended Solids
TTP	Thickened Tailings Plant
UCS	Unconfined Compressive Strength
USD	United States Dollar
V	Volt
VAT	Value Added Tax
WRP	Waste Rock Piles
wk	Week
WMP	Waste Management Plan
W:O	Stripping Ratio (Waste to Ore ratio)
WTREC	Weight Recovery
y	Year