NI 43-101 Technical Report
Hardrock Project
Ontario, Canada

Prepared for:

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IMPORTANT NOTE

This Report was prepared as a National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Premier Gold Mines Limited ("Premier") by G Mining Services Inc. ("GMS") and Wood Canada Limited. The quality of information, conclusions, and estimates contained herein is based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this Report.

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1. SUMMARY

1.1 Introduction

The 2016 Hardrock Project Feasibility Study ("FS") concluded that the Hardrock Project (the "Project") is technically and economically feasible. Since the FS, Greenstone Gold Mines GP Inc. acting as the managing partner of Greenstone Gold Mines LP (collectively, "GGM"), has advanced Project de-risking activities, including receiving federal and provincial approval of the EIS/EA, establishing the Long Term Relationship Agreements, advancing permitting activities, confirming the continuity of gold grade through additional drilling, optimizing the pit design and mining plan and concluding a value engineering initiative.

As part of the 2019 approved work plan, GGM, G Mining Services Inc. ("GMS") and several engineering consultants collaborated to prepare a Technical Report Feasibility Study Update. The objective of the Project Update included updating the resource and reserve models, revising the life of mine plan, advancing detailed engineering and completing firm price bid processes for all major process plant, equipment, power plant equipment, effluent and sewage treatment plants and the mine mobile equipment fleet. This work formed the basis of the capital cost, operating cost and project economic update. This Report Feasibility Study Update Report ("Report") was prepared for Premier Gold Mines Limited ("Premier") and summarizes the results of this work.

On December 15, 2020 the Orion Mine Finance Group ("Orion"), has entered into an agreement (the "Purchase Agreement") with Centerra and Premier pursuant to which Orion will acquire Centerra's 50% interest in the GGM Partnership. On December 16, 2020 Equinox Gold Corp. and Premier entered into a definitive agreement (the "Agreement") whereby Equinox Gold will acquire all of the outstanding shares of Premier. Equinox Gold will retain Premier's interest in the world-class Hardrock Project in Ontario.

The scope of this Report includes the geology and Mineral Resources of the Hardrock Property and the other properties (Brookbank, Kailey and Key Lake, collectively known as the "Hardrock Satellite Deposits"). The Mineral Reserves, mining, infrastructure, processing and financial analysis sections of this Report consider the Hardrock Deposit only.

The Report and Project Update responsibilities of the engineering consultants follow:

- G Mining Services Inc. ("GMS") - overall Report and integration, property description and location, accessibility, history, geological setting and mineralization, deposit types, exploration, drilling, sample preparation and security, data verification, Mineral Resource estimates, Mineral Reserves (pertaining to the Hardrock deposit only), mining methods, economic analysis, operating costs pertaining to mining, review of capital costs;

- Stantec Consulting Limited ("Stantec") - climate and physiology, environmental, permitting unless otherwise noted, and social aspects;
- Soutex Inc. ("Soutex") - metallurgical testing, recovery methods, mineral processing operating cost;
- Ausenco Engineering Canada Inc. ("Ausenco") – processing plant and supporting infrastructure engineering;
- Wood Canada Limited ("Wood") - Tailings Management Facility (TMF) design, Goldfield Creek diversion design, geotechnical engineering for the waste rock storage areas, TMF Closure Plan and permitting of TMF related facilities;
- SLR Consulting (Canada) Ltd. ("SLR") – northside water management, mine rock stockpile pond design and mine site water balance.

1.2 Property Description and Land Tenure

GGM's Greenstone Gold Property, formerly known as the Trans-Canada Property (the "Property") is located approximately 275 km northeast of Thunder Bay, Ontario. The Property includes three blocks of contiguous claims known as the Hardrock, Brookbank and Viper areas, over a distance of more than 100 km, located along, or in close proximity to, the Trans-Canada Highway between the towns of Beardmore and Longlac, Ontario.

The Hardrock claim group includes the Hardrock, Key Lake and Kailey Deposits. The Brookbank claim group hosts the Brookbank, Cherbourg and Fox Ear deposits and the Irwin prospect.

As of the date of this Report, the Property consists of a contiguous block of patented claims, mining leases, licences of occupation and cell claims covering a total area 39,072.1 hectares ("ha"), of which 15,862.7 ha relates to Hardrock Project claims. All claims, leases and licences of occupation are beneficially held by GGM on behalf of the Partnership and are subject to terms under a number of agreements.

1.3 Mineral Resource Estimate

1.3.1 Hardrock Project

Since the previous Mineral Resource was released in 2016, substantial drilling has been conducted and was successful in de-risking the Mineral Resource Estimate ("MRE") in the early years of production.

RC grade control drilling on a 20 m (X) by 10 m (Y) spacing was undertaken in 2018 and 2019 targeting the first three benches of production, and also partially tested an additional four benches in certain areas. In 2019, diamond drilling was undertaken in areas identified as requiring infill drilling, and resulted in the validation of the new geological interpretation and confirmation of the grade continuity.

The principal factors contributing to the increase in the current MRE are as follows:

- The 2019 MRE is constrained by a deeper pit optimization, which incorporates significantly more resources compared to the 2016 MRE;
- The reduction of internal dilution within the seventeen principal domains has resulted in a 24% increase in average grade of assays within these domains, thus a higher overall gold grade in the mineral resource;
- Grade capping was revisited in 2019 (due to the refined wireframes), and new capping thresholds were chosen. They are generally less restrictive than the capping chosen in 2016;
- RC grade-control drilling and validation diamond drilling conducted in 2018 and 2019 confirmed grade continuity, and generally intersected higher-grades than expected in the 2016 block model.

Definitions for Mineral Resource categories used in this Report are consistent with the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 ("CIM definitions") and adopted by NI 43-101. GMS is not aware of any environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the Mineral Resource estimate.

This Report is based on an open pit mining scenario. The in-pit Mineral Resources at the Hardrock Deposit are constrained within the design pit using a cut-off grade of 0.30 g Au/t. In addition to in-pit Mineral Resources, an underground Mineral Resource was estimated outside the open pit using a 2.0 g Au/t cut-off grade. The Project open pit and underground Mineral Resources are summarized in Table 1.1.

The Mineral Resource estimate covers a corridor of the Hardrock deposit with a strike length of 5.7 km and a width of approximately 1.7 km, down to a vertical depth of 1.8 km below surface. Mineralized zones were interpreted in 3D using Leapfrog GEO™ software based on a litho-structural model and the drill hole database. The drill hole database used in the estimate contained 312,408 sampled intervals from 696,125 m of diamond drilling in 1,682 holes, and 11,871 assays from 25,961 m of RC drilling in 481 holes. Channel samples were not used in the estimation.

Mineral Resources were estimated by applying a minimum true thickness of 3.0 m and using the grade of the adjacent material when assayed or a value of zero when not assayed. High-grade capping on raw assay data was established on a per zone basis. Compositing was conducted on drill hole sections falling within the mineralized zones (composite = 2 m). Mineral Resources were estimated using 3D block modelling and 3-pass ID^3 interpolation with high-grade restraining.

Mineral Resources were classified as Measured in areas within 15 m of the RC grade control drilling and Indicated in areas where the maximum distance to drill hole composites was less than 35 m for blocks interpolated in Passes 1 and 2 (using a minimum of two drill holes). Mineral Resources were classified as Inferred in remaining blocks interpolated during Passes 1 to 3. Lastly, all blocks in the underground resource estimated in Pass 1 to 3 in the external grade shell domain (500, 501 and 506) were downgraded to Inferred category. A grooming step was undertaken on the classification to ensure that the Resource category is coherent for mine planning purposes.

Table 1.1: Mineral Resource Estimate (Exclusive of Mineral Reserves) for the Hardrock Project

Resource Type	Cut-off (g Au/t)	In-Pit	Underground	Total
		> 0.30 g Au/t	> 2.00 g Au/t	
Indicated	Tonnes (t)	5,972,000	9,792,000	15,764,000
	Grade (g Au/t)	1.21	3.93	2.90
	Au (oz)	231,400	1,237,400	1,468,800
Inferred	Tonnes (t)	356,000	24,593,000	24,949,000
	Grade (g Au/t)	1.14	3.87	3.83
	Au (oz)	13,100	3,059,100	3,072,200

Mineral Resource Estimate Notes:

1. *The Independent and Qualified Person for the Mineral Resource Estimate, as defined by NI 43-101, is Rejean Sirois, B.Sc., P.Eng. of G Mining Services Inc., and the effective date of the estimate is 04/09/2019;*
2. *These Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability;*
3. *Mineral Resources are exclusive of Mineral Reserves;*
4. *In-Pit results are presented undiluted within a merged surface of the pit optimization shell 24 and the 2019 pit design, using a USD 1,250 gold price and a revenue factor 0.78;*
5. *Whittle parameters (all amounts in Canadian dollars): Reference mining cost: $1.98/t, Incremental bench cost ($/10 m bench): $0.033, Milling cost: $7.54/t, Royalty: 4.4%, G&A: $1.59/t, Sustaining capital: $0.70/t, Gold price: $1,625/oz, Milling recovery: 91.1%;*
6. *Ounce (troy) = Metric Tonnes x Grade / 31.10348. Calculations used metric units (metres, tonnes and g/t);*
7. *The number of metric tonnes was rounded to the nearest thousand and ounces was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in Regulation NI 43-101;*
8. *GMS is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the Mineral Resource estimate.*
9. *2014 CIM definitions were followed for Mineral Resources*

1.3.2 Other Greenstone Gold Deposits

In addition to the Hardrock deposit, Mineral Resources were updated for the Brookbank, Kailey and Key Lake Deposits, collectively known as the "Hardrock Satellite Deposits". Open pit optimization using Whittle software, based on the Lerchs-Grossmann algorithm, was completed to estimate in-pit Mineral Resources for all three deposits. For Brookbank, underground Mineral Resources were also estimated. All these Mineral Resources are effective as of September 3rd, 2020. There are no Mineral Reserves currently estimated for these deposits. Refer to Table 1.2.

Table 1.2: Summary of Brookbank, Kailey and Key Lake Mineral Resources

Deposit	Mining Method	Category	Tonne (Mt)	Gold Grade (g/t)	Contained Gold (koz)
Brookbank	Open Pit	Indicated	1.147	2.24	83
		Inferred	0.045	2.07	3
	Underground	Indicated	2.281	7.06	517
		Inferred	0.706	3.38	77
Key Lake	Open Pit	Indicated	3.781	1.16	141
		Inferred	1.836	1.39	82
Kailey	Open Pit	Indicated	11.276	0.96	348
		Inferred	4.858	0.87	136

Notes:

1. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*
2. *No Mineral Reserves are quoted for Brookbank, Kailey or Key Lake.*
3. *The independent and qualified person for the 2020 Brookbank, Kailey and Key Lake MRE's is Mr. James Purchase P.Geo of GMS.*
4. *The effective date of the estimates is September 3rd,2020.*
5. *Open-pit Mineral Resources are constrained within a pit shell using a gold price of US$1,500, a CAD:USD exchange rate of 1.3 and a metallurgical recovery of 92% for Brookbank, and 90% for Kailey and Key Lake. An incremental ore haulage cost of $17.90/t is assumed for Brookbank, $1.70/t for Kailey and $4.51/t for Key Lake.*
6. *Open Pit Mineral Resources are reported at a cut-off grade of 0.60 g Au/t for Brookbank, and 0.40 g Au/t for Kailey and Key Lake. Underground Mineral Resources are reported at a cut-off grade of 2.4 g Au/t for Brookbank.*
7. *GMS is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the Mineral Resource estimate.*
8. *2014 CIM definitions were followed for Mineral Resources.*

1.4 Mineral Reserves

The Mineral Reserve for the Hardrock Project was estimated based on the open pit mining scenario proposed in this Report and is summarized in Table 1.3. The Mineral Reserve estimate was prepared by GMS.

Table 1.3: Mineral Reserve Estimate (Open-Pit)

Category	Diluted Ore Tonnage (kt)	Gold Grade (g Au/t)	Contained Gold (koz Au)
Proven	5,623	1.28	232
Probable	129,700	1.27	5,307
Total P&P	**135,323**	**1.27**	**5,539**

Notes:

1. *CIM definitions were followed for Mineral Reserves;*
2. *Effective date of the estimate is August 8th, 2019;*
3. *Mineral Reserves are estimated at a cut-off grade of 0.35 g Au/t;*
4. *Mineral Reserves are estimated using a long-term gold price of USD 1,250/oz and an exchange rate of CAD : USD 1.30;*
5. *A minimum mining width of 5 m was used;*
6. *Bulk density of ore is variable but averages 2.78 t/m^3;*
7. *The average strip ratio is 5.10:1;*
8. *Dilution factor is 17.2%;*
9. *Numbers may not add due to rounding.*

The mine design and Mineral Reserve estimate have been completed. The Mineral Reserve estimate is consistent with the CIM definitions and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated Mineral Resources ("M&I"), and do not include any Inferred Mineral Resources. Indicated Mineral Resources were converted in Probable Mineral Reserve and Measured Mineral Resource in Proven Mineral Reserve. The Inferred Mineral Resources contained within the mine design are classified as waste.

Open pit optimization was conducted using Whittle software to determine the optimal economic shape of the open pit to guide the pit design process. The Mineral Reserve estimate includes a 17.2% mining dilution at an average grade of 0.13 g Au/t and a 1.5% ore loss factor.

A feasibility level pit slope design study was carried out by Golder. The conclusions of this study have been used as an input to the pit optimization and design process.

1.5 Mining

Mining will be carried out using conventional open pit techniques with 10 m benches. An Owner mined open pit operation is planned with hydraulic shovels and mining trucks, including outsourcing of certain support activities such as explosives manufacturing and blasting.

Production drilling of the 10 m benches will be by blast hole drill rigs with both rotary and down-the-hole ("DTH") drilling capability. Blast holes are loaded with bulk emulsion. The majority of the loading in the pit will be carried out by two 29 m^3 hydraulic face shovels, one 29 m^3 hydraulic excavator, and two 30 m^3 front-end wheel loaders. The shovels and loaders will be matched with a fleet of 216 t payload mine trucks. The presence of underground stopes was considered when designing the pits mainly for the void in the F-Zone, which is 150 m high and 30 m wide. Most of the other underground openings are backfilled with sand fill or rock fill.

Mining of the Hardrock main pit will occur in five main phases preceded by a starter pit. Waste rock will be disposed of in five distinct waste dumps with four located around the pit and one further to the south. The open pit generates 689.6 Mt of overburden and waste rock (inclusive of historic tailings and underground backfill) over the life of mine ("LOM") for an average LOM strip ratio of 5.1:1.

The LOM plan (Figure 1.1) details 12.9 years of mine production (from April of Year 1 to February of Year 14), preceded by a preproduction period of 20 months which includes a 4-month plant commissioning period (excludes crushing). The processing plant will take 13 months to reach its full processing rate of 27,000 tpd. Once the open pit is depleted, the process plant is fed for an additional 9 months from low grade stockpiles. Commercial operations are scheduled for 13.7 years (from April of Year 1 to November of Year 14).

Figure 1.1: Annual Mine Production

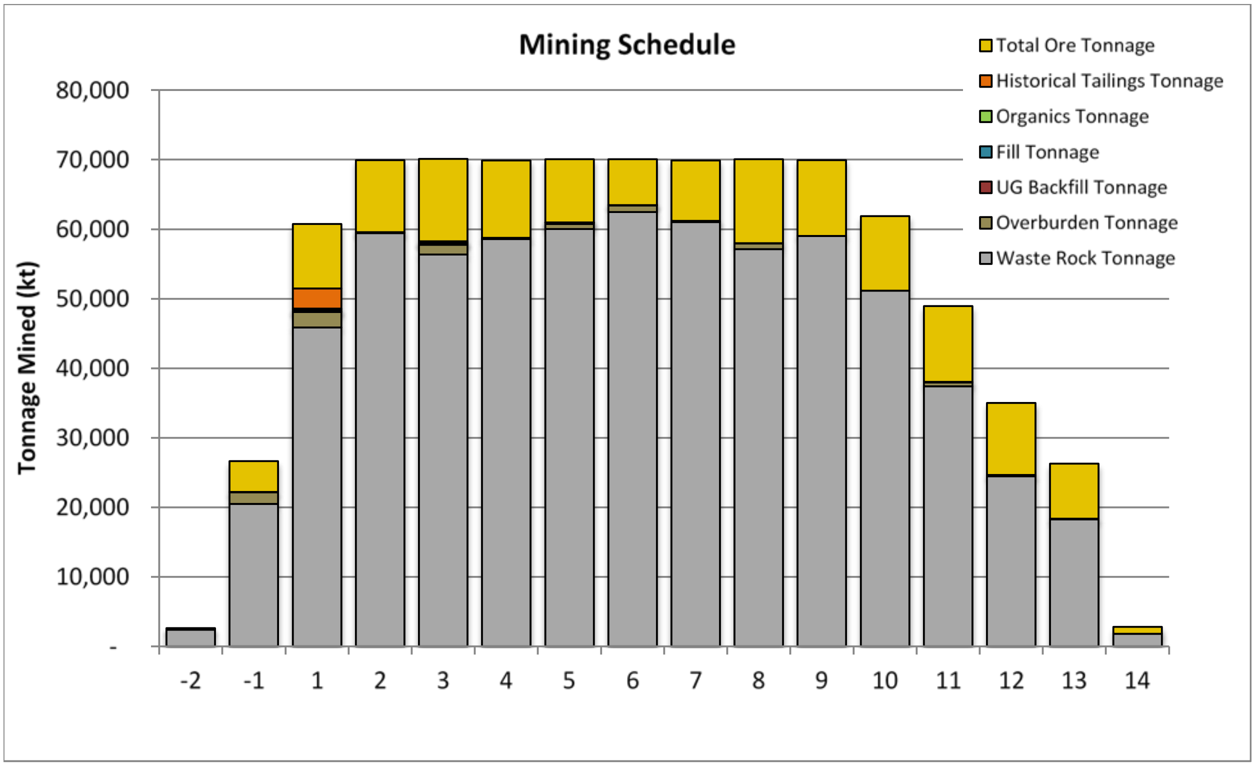
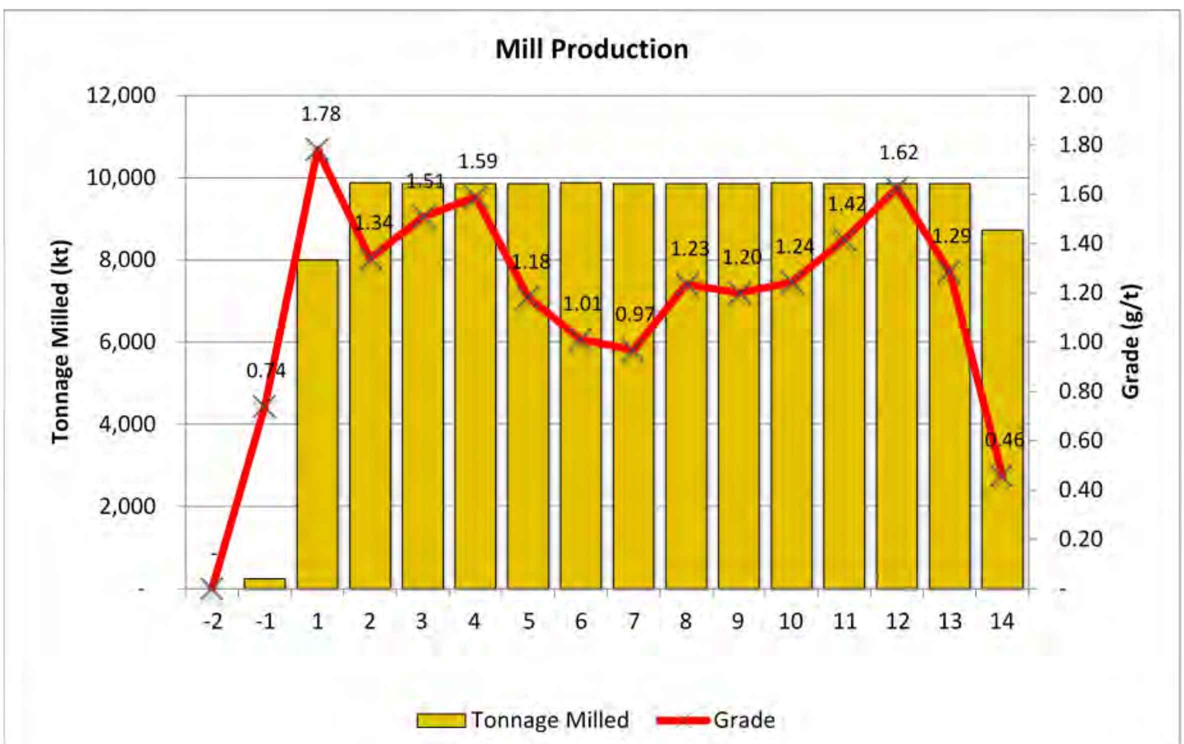


Figure 1.2: Annual Mine Production



1.6 Mineral Processing and Metallurgical Testing

The process design criteria have been established based on testwork results, GGM and vendor recommendations or requirements and industry practices.

Prior to the start of the FS, between 2011 and 2013, mineralogy, grindability and gold recovery testwork was performed by SGS Lakefield Research Limited ("SGS Lakefield") and McClelland Laboratories Inc. ("McClelland"). The SGS Lakefield testwork showed that the ore is composed mainly of quartz and plagioclase with minor amounts of pyrite and arsenopyrite, gold occurs mainly as native gold, the ore is in the category of medium hardness to moderately hard, a portion of the gold can be recovered by gravity concentration and gold can be recovered to a bulk flotation concentrate. The subsequent McClelland testwork showed that gold recovery increased with finer grind size and was unaffected by cyanide concentration.

In the course of the Preliminary Economic Assessment ("PEA") and FS, additional testwork was carried out by SGS Lakefield, JKTech Pty Ltd and FLSmidth. Primarily, high pressure grinding roll ("HPGR") tests confirmed the ore amenability for high pressure grinding, and facilitated equipment selection and operating cost estimation. Grindability, head grade determination, mineralogy, magnetic separation, gravity recovery, flotation, cyanidation, cyanide destruction, solid-liquid separation and other tests were completed. Additional thickening and rheology testwork were carried out to determine the sizing and operating parameters of a pre-leach thickener.

The HPGR testing program included laboratory scale tests to determine the amenability of the ore to HPGR milling and yield preliminary sizing data; abrasion tests to predict the service life of the rolls and a large-scale pilot plant test to size the equipment. Bond grindability testing was performed to evaluate the Ball Work Index ("BWI") reduction of the HPGR product compared to the feed. A detailed comminution trade-off study recommended two-stage crushing followed by HPGR and ball milling over crushing followed by semi-autogenous ("SAG") milling and ball milling, to reduce throughput risk and increase energy efficiency.

In the detailed engineering phase additional leach test work was carried out on near-surface samples from the 2018 drilling campaign to characterize gold recovery, oxygen consumption, solid-liquid separation and rheology.

A multivariate linear regression analysis was used to estimate gold recovery based on ore grade and mineralogical composition. The results of the cyanidation tests conducted on composites were used as the basis for the analysis. The residual gold grade from the cyanidation testwork was found to be highly correlated to the gold, arsenic and sulphur head sample grades, and somewhat less on grind size.

The gold recovery process for the Project consists of a crushing circuit (gyratory and cone), a grinding circuit (HPGR and ball mill), pre-leach thickening and cyanide leaching, a carbon in pulp ("CIP") circuit, carbon elution and regeneration, electrowinning and gold refining, cyanide destruction and tailings disposal. The plant is designed to operate at a throughput of 27,000 t/d. The process operation schedule is 24 hours per day, 365 days per year, with an overall availability of 92%.

Gold production averages 414 koz for the first five years of production (from start of Year 1 to end of Year 5) with an average head grade of 1.45 g Au/t and an average metallurgical recovery of 91.2%.

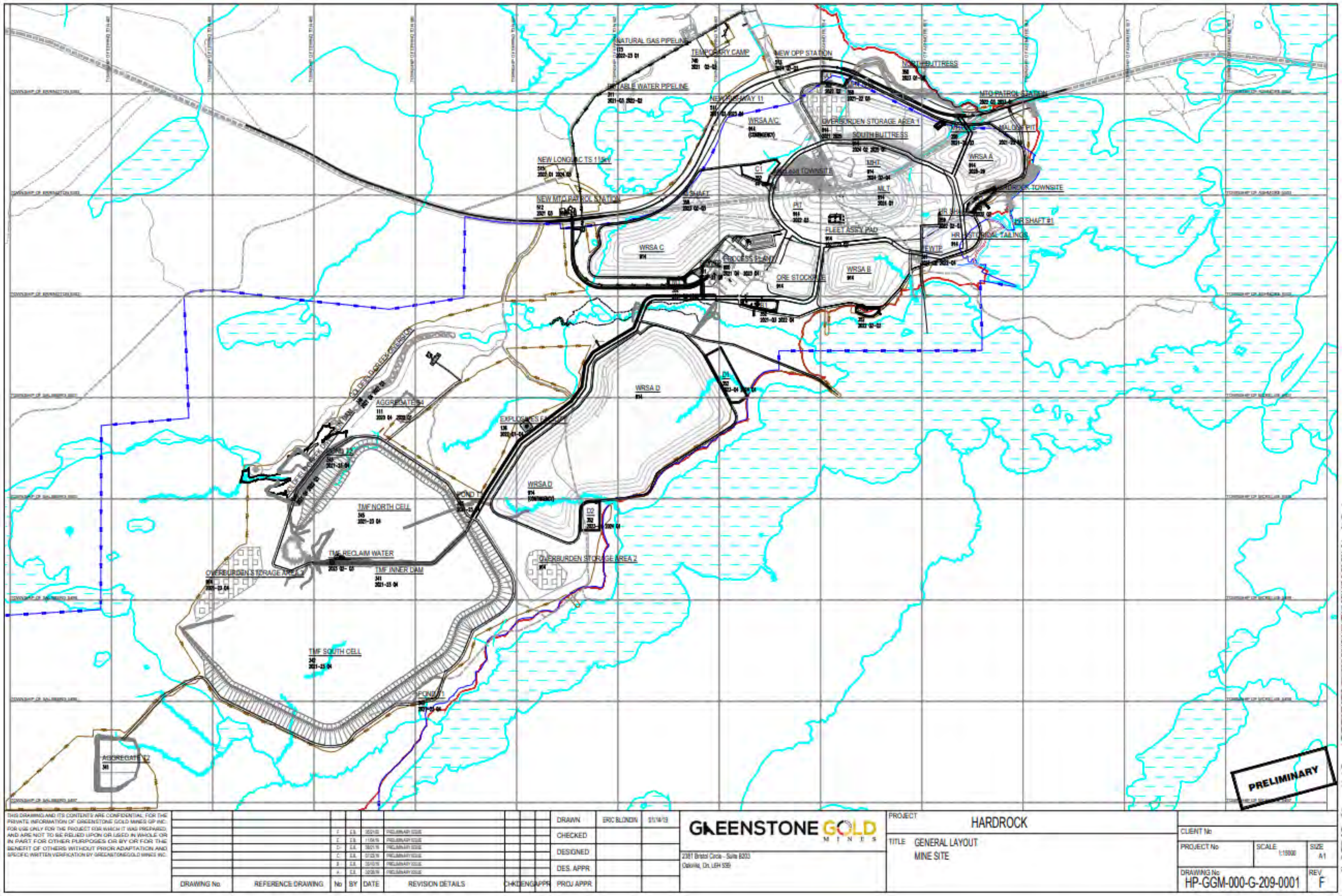
1.7 Mine Infrastructure and Services

The Project occurs in a district with active mines and processing facilities located at Hemlo and Timmins, Ontario and therefore has access to good transportation and regional mining related infrastructure. The Project is located in close proximity to the Trans-Canada Highway 11, TransCanada Pipelines Limited Canadian Mainline ("TCPL Mainline") natural gas pipeline, a Hydro One electrical substation and Geraldton hosts a municipal airport, which has a 1,500 m runway capable of accommodating large aircraft. Geraldton has its own potable water treatment system and water distribution network, which are proposed to be used for the Project.

General infrastructure for the Project that will be constructed to support mining and processing will include:

- Site access and haul roads;
- Workshop and maintenance facility;
- Warehousing for spare parts and reagents;
- Administration building including a dry facility, gatehouse and parking area;
- Explosive reagent storage;
- Fuel storage and distribution;
- Recycling and sorting facility;
- Potable water and sewage systems;
- Fire water systems;
- Site security and fencing.

Figure 1.3: Hardrock Site General Arrangement



Note: North is upward, and the grid is 1 km by 1 km

Figure 1.4: Process Plant and Mine Infrastructure

Existing infrastructure within the footprint of the property limits that will need to be relocated includes:

- Trans-Canada Highway 11;
- Existing Hydro One 115 kV station;
- OPP Station;
- MacLeod High Tailings (portion covering the open pit mine);
- MTO patrol station.

Portions of a golf course and the MacLeod-Cockshutt (MacLeod-Mosher) mine headframe will be purchased from the municipality during the operations phase. Private properties in the MacLeod townsite and Hardrock townsite (65 in total) and the gas station have now been purchased.

The existing Hydro One grid is insufficient for powering the processing facilities and associated infrastructure. A 65 MW natural gas-fired power plant will be constructed, with a designed capacity of 46.5 MW, which will include a natural gas pipeline originating from the existing TCPL Canadian Mainline pipeline directly to the site power plant.

GGM has committed as part of the EA to remove a portion of the historical tailings. Approximately 23% of the historical MacLeod tailings will be removed as part of the starter pit and pit expansion during the first year of operations, while 70% of the historical Hardrock tailings will to be relocated to the TMF (Year 6 to Year 9 of operations). To ensure the stability of the remaining Historical tailings in place, a buttress will be constructed along the north side of the Historical tailings.

1.8 Water Management

Two types of effluents will be generated during Project activities: mine effluent and sanitary effluent. The water quality standards applicable to mine effluent are the Provincial Water Quality Objectives (“PWQOs”) (MOE, 1994), Ontario Regulation (“O.Reg.”) 560/94-MISA Metal Mine Sector Effluent Criteria, and Federal Metal Mining Effluent Regulations (“MMER”) Effluent Criteria. The Assimilative Capacity Study (Stantec, 2016) conducted for the Project identified discharge locations and proposed quality criteria for both mine and sanitary effluents discharging to the Southwest Arm of Kenogamisis Lake which are protective of the receiving environment. The effluent criteria proposed meet and exceed MMER and O.Reg. 560/94 criteria at end of pipe and the PWQOs for all parameters are met within a small mixing zone in the receiving waterbody.

All collected mine water, surface runoff water and underground workings water will be directed through various runoff and seepage collection ponds to the centralized mine water Collection Pond M1, which is designed to provide buffer flows for mill make-up water and the effluent water treatment plant supply. The treated water is then released to the Southwest Arm of Kenogamisis. A seepage collection system will be installed to manage seepage from the Macleod historical tailings as an early project development activity. Runoff and seepage collection from the exterior of the TMF dams will be collected in a series of ponds and generally pumped back to the TMF for re-use in processing. In case there is surplus water, which cannot be pumped into the TMF, water will be treated prior to discharging to the environment.

1.9 Tailings Management Facility

The TMF dams have been designed to meet the requirements of the Lakes and River Improvement Act Ministry of Natural Resources ("MNR, 2011") and the Canadian Dam Association guidelines ("CDA, 2014") with a relatively low permeability core protected by filters and transition zones upstream of the main embankment, and constructed of geochemically benign mine rock. The TMF dam foundation was characterized by conducting extensive geotechnical investigations. A site-specific seismic hazard study was carried out for the seismic design of the TMF dams. The stability of the dams meets the target factors of safety required as per CDA.

The TMF site is located approximately five kilometres southwest of the process plant site and was selected to minimize the disturbance to fish bearing water bodies, maximize the use of natural containment and optimize Project economics. Prior to construction of the TMF, Goldfield Creek will be diverted around the north side of the TMF into a permanent channel designed to provide fisheries compensation.

The site has a positive water balance, and as such, the TMF will be developed to minimize the surplus water requiring treatment. It is planned to complete tailings deposition early in one cell to allow for progressive rehabilitation and shedding of runoff from the system.

Closure of the TMF involves lowering of the spillways and vegetation of the exposed beaches. Runoff will be directed through spillways constructed in natural ground when deemed suitable for discharge to the environment.

1.10 Environmental Studies

Environmental baseline studies were initiated for the Project in 2013 and were used to identify environmental constraints during the development of preliminary layouts and designs for the Project. This included consideration of siting and layout of Project infrastructure as well as consideration of design alternatives from an environmental management and approvals perspective. This environmental baseline was the basis for determining incremental changes and predicting environmental effects associated with the Project.

A final environmental impact statement / environmental assessment ("EIS/EA") has been completed and approved by provincial and federal regulatory agencies. Project interactions were analyzed for 13 valued components ("VCs") to determine potential environmental effects associated with the Project for construction, operation and closure phases. In addition to the VCs, the effects assessment also considered effects of the environment on the Project, accidents and malfunction scenarios and cumulative effects. Conceptual environmental management and monitoring plans ("EMMPs") were provided in the final EIS/EA, including measures related to both compliance and EIS/EA monitoring for all phases of the Project. The EMMPs are being advanced beyond the conceptual stage throughout 2019 and 2020 and will be 'living' documents as the Project progresses.

A conceptual Closure Plan was developed as part of the EIS/EA to provide an early opportunity to discuss the closure approach and initial costing. At the end of mining operations, the main features requiring closure will include the main open pit, water management and drainage systems, waste rock storage areas, TMF, site access roads and buildings and associated infrastructure. After the closure works have been completed, a post-closure monitoring program will be carried out to verify that the closure objectives and criteria have been met and confirm that the Project can proceed to final close out status. The Closure Plan has now advanced to a final version and was approved by the MENDM in January 2020.

The results of the final EIS/EA, including implementing the identified mitigation measures, supports the conclusion that the Project will not cause significant adverse environmental effects. There are no issues identified to date that would materially affect the ability of GGM to extract minerals from the Project. Since completing the final EIS/EA, GGM has completed slight modifications of Project components as detailed engineering advances, which form the basis for the final mine plan used for this Report. Active consultation with stakeholders (community members, agencies and interested parties) and Indigenous communities has been undertaken throughout Project planning and will continue as the Project progresses through permitting and detailed engineering.

GGM has established Long Term Relationship Agreements ("LTRAs") with the five local Indigenous communities. The agreements establish increased clarity regarding GGM's ability to develop the Project and the Indigenous communities' opportunity to benefit from future mining opportunities in the region, including the potential to extend the life of the Project.

1.11 Execution Plan

The Project will be executed using an “Owner-managed” project delivery model. All aspects of engineering, procurement and construction for the Project will be managed directly by the Owner. Detailed engineering and a portion of the procurement will be outsourced. The pre-production CAPEX period is planned for 39 months (start of CAPEX period to commercial production), which includes 8 months of construction readiness and pre-construction and 27 months for major construction. The process plant cold commissioning, hot commissioning and ramp up to commercial production is planned to be completed over 7 months. The peak construction workforce on site is estimated at 660 people.

The operating organization consists of three departments: mine, including mine operations, geology, engineering and maintenance; process and power plant, including operations and maintenance; and general and administrative including human resources, environment, health and safety, site services and accounting. The planned peak total operating workforce is 521 employees (reached in Year 4).

1.12 Capital Cost Estimate

The CAPEX estimate was based on material take-offs from detailed engineering in most areas (50% to 90%). The process plant equipment, power plant equipment, temporary and permanent effluent treatment plants, sewage treatment plant and mine mobile equipment fleet was based on firm price Request for Proposal (“RFP”) processes. The remaining equipment and material costs were based on budgetary bid processes, quotes, consultant’s historical data and in-house databases or benchmarked from previous projects. Labour unit rates were developed from first principles based on budgetary quotations and direct installation hours were based on a combination of firm price proposals, budgetary quotes and feasibility study estimates, benchmarked against previous projects and reviewed by experienced construction personnel. Firm price RFP processes were completed for the TMF, Goldfield Creek diversion and Highway 11 relocation. A Quantitative Risk Assessment (“QRA”) session was held to establish the contingency for the CAPEX update.

The initial CAPEX for Project construction, equipment purchases and pre-production activities is estimated to be CAD 1,301M (before preproduction revenue and LTRA costs), as shown in Table 1.4.

The CAPEX includes a contingency of CAD 108M, which is 9.0% of the total before contingency. The project initial CAPEX excludes pre-production revenue and IBA payments scheduled prior to commercial production.

Table 1.4: Capital Expenditures Summary

Work Breakdown Structure	Total CAD M
100 - Infrastructure	79.4
200 - Power & Electrical	70.4
300 - Water & Tailings Management	94.2
400 - Mobile Equipment	155.1
500 - Infrastructure Repositioning	61.3
600 - Process Plant General	313.8
700 - Construction Indirect Cost	242.1
800 - General Services - Owner's Cost	44.8
900 - 980 - Preproduction, Startup, Commissioning	131.2
990 - Contingency	108.4
Grand Total	**1,300.7**

Sustaining capital is required during operations for additional equipment purchases, mine equipment capital repairs, mine civil works, TMF dam raises and additional infrastructure relocation. The sustaining capital is estimated at CAD 420M.

The total salvage value is estimated at CAD 45M, and includes mining equipment purchased during operations that will not have been utilized to its useful life, a residual value for some of the process plant major equipment and a residual value for the power plant as the units will have a remaining useful life of 10 to 15 years at the end of operations.

Reclamation and closure costs include infrastructure decommissioning, site preparation and revegetation, maintenance and post closure monitoring. The reclamation costs are spent over three years at the end of operations. The total reclamation and closure cost is estimated to be CAD 54M.

1.13 Operating Cost Estimate

Operating costs ("OPEX") are summarized in Table 1.5. The OPEX includes mining, processing, general and administration ("G&A"), transportation and refining, other costs and royalties. The average OPEX is CAD 708/oz Au or CAD 26.51/t milled over the LOM. The all-in sustaining cost ("AISC") which includes closure, reclamation and sustaining capital costs averages CAD 803/oz Au over the LOM.

Table 1.5: Operating Cost Summary

Category	Total Costs (CAD M)	Unit Cost (CAD/t milled)	Cost per oz (CAD/oz)
Mining	1,890	14.13	378
Processing	968	7.23	193
G&A	401	3.00	80
Transp. & Refining	15	0.11	3
Royalties	253	2.04	55
Total Operating Cost	**3,547**	**26.51**	**708**
Closure & Reclamation	54	0.40	11
Sustaining Capital	420	3.14	84
All-in Sustaining Cost (AISC)	**4,020**	**30.05**	**803**

1.14 Economic Analysis

The base case economic model has been developed using a long-term gold price assumption of USD 1,400/oz and an exchange rate of CAD/USD 1.30.

Gold production over the LOM is 5,051 koz based on an average processing recovery of 91.2%. Gold production begins during the pre-production period and is treated as revenue partially offsetting pre-production costs.

The economic model excludes any Project or equipment financing assumptions. The Project funding is assumed to be through equity for the purposes of the Report. The economic results are calculated as of the start of the pre-production CAPEX phase at Year -3 which includes the remaining detailed engineering and all procurement.

The Partnership is not subject to income taxes and each respective joint venture partner will bear the responsibility for paying tax on profits generated by the Partnership. The post-tax results in this Report are based on the assumption that GGM is a taxable Canadian entity and tax is calculated based on the tax rules in Ontario. The calculations include tax losses incurred by GGM since the inception of the partnership, but do not reflect the benefit of any historical tax positions held by either joint venture partner (if any).

The before-tax Project cash flow over the Project life is estimated at CAD 3,911M. The Project before-tax net present value ("NPV") at a discount rate of 5% is estimated to be CAD 2,054M with a before-tax internal rate of return ("IRR") of 24.7%.

The total after-tax cash flow over the Project life is estimated to be CAD 2,716M. The Project after-tax NPV at a discount rate of 5% is estimated to be CAD 1,364M. The after-tax Project cash flow results in a 3.2-year payback period from the commencement of commercial operations with an after-tax IRR of 20.1%. Table 1.6 is a summary of the Project economics.

Figure 1.5: Annual Gold Sales

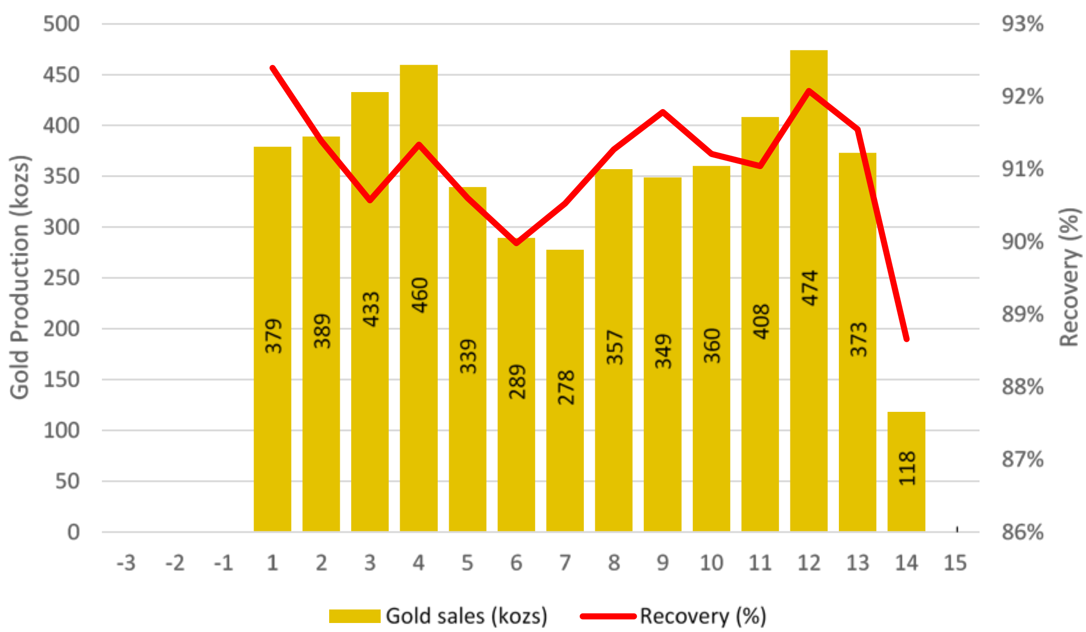


Table 1.6: Project Economics Result Summary

Project Economics		Base Case Results
Production Summary		
Tonnage Mined	Mt	824.9
Ore Milled	Mt	135.3
Head Grade	g Au/t	1.27
Gold Processed	k ozs	5,539
Recovery	%	91.2%
Gold Production	k ozs	5,051
Cash Flow Summary		
Gross Revenue	M CAD	9,112
Mining Costs (incl. rehandle)	M CAD	(1,890)
Processing Costs	M CAD	(968)
G&A Costs (incl. transport & refining)	M CAD	(416)
Royalty Costs	M CAD	(273)
Total Operating Costs	M CAD	(3,547)
Operating Cash Flow Before Taxes	M CAD	5,566
Initial CAPEX	M CAD	(1,226)
Sustaining CAPEX	M CAD	(420)
Total CAPEX	M CAD	(1,646)
Salvage Value	M CAD	45
Closure Costs	M CAD	(54)
Interest and Financing Expenses	M CAD	-
Taxes (mining, prov. & fed.)	M CAD	(1195)
Before-Tax Results		
Before-Tax Undiscounted Cash Flow	M CAD	**3,911**
NPV 5% Before-Tax	M CAD	**2,054**
Project Before-Tax Payback Period	years	**2.8**
Project Before-Tax IRR	%	**24.7%**
After-Tax Results		
After-Tax Undiscounted Cash Flow	M CAD	**2,716**
NPV 5% After-Tax	M CAD	**1,364**
Project After-Tax Payback Period	years	**3.2**
Project After-Tax IRR	%	**20.1%**

Table 1.7 is a summary of the Project NPVs at various discount rates.

Table 1.7: Project Net Present Values at Various Discount Rates

Discount Rate	Before-Tax Project NPV (M CAD)	After-Tax Project NPV (M CAD)
5%	2,054	1,364
6%	1,804	1,183
7%	1,584	1,022
8%	1,389	879.9

A sensitivity analysis was performed for ±10% and ±15% variations for gold price, exchange rate, operating costs and initial capital expenditure.

The Project is most sensitive to gold price followed by exchange rate, initial capital costs and operating costs. The Project is somewhat less sensitive to the CAD/USD exchange rate than the gold price in USD/oz as some of the CAPEX are in US dollars. The sensitivity on gold grade is identical to that of the gold price and is therefore not presented in the following figures.

The results of the sensitivity analysis on after-tax undiscounted NPV and IRR are presented in Table 1.8.

Table 1.8: Project After-Tax Sensitivities

	NPV 5%			IRR		
Technical Report Feasibility Study Variable	-15% (M CAD)	Update (M CAD)	+15% (M CAD)	-15% (% IRR)	Update (% IRR)	+15% (% IRR)
Operating Costs	1,553	1,364	1,175	21.8%	20.1%	18.4%
Capital Costs	1,486	1,364	1,240	23.6%	20.1%	17.4%
Exch. Rate (CAD/USD)	837	1,364	1,885	15.1%	20.1%	24.4%
Gold Price	816	1,364	1,905	14.7%	20.1%	24.9%

1.15 Interpretation and Conclusions

Following the 2016 FS, the completion of the Technical Report Feasibility Study Update has reconfirmed the technical feasibility and economic viability of the Project based on an open pit mining operation with average gold production at 366 koz per year. The processing of ore extends beyond the life of the open pit with the processing of low grade stockpiles for a total of 13.7 years. The principal conclusions by area are detailed below.

- Geology and Mineral Resources:
  - Since the 2016 MRE, there has been significant RC and diamond drilling at the Hardrock Project. Drilling focused on de-risking the early years of production (RC grade control targeting the first 3 benches of production), in-filling gaps in the drill pattern and validating the new mineralization interpretation.
  - Cut-off grades of 0.30 g Au/t for the in-pit resource and 2.00 g Au/t for the underground resource are appropriate for reporting Mineral Resources for the Hardrock Project.
  - At a cut-off grade of 0.30 g Au/t, the in-pit Measured and Indicated Mineral Resources are estimated to be 137.7 Mt grading 1.33 g Au/t for 5.9 Moz of gold, inclusive of mineral reserves. In-pit Inferred Mineral Resources are estimated to be 0.9 Mt grading 1.19 g Au/t for 36 koz of gold, inclusive of mineral reserves.
  - At a cut-off grade of 2.00 g Au/t, the underground Indicated Mineral Resources are estimated to be 9.8 Mt grading 3.93 g Au/t for 1.2 Moz of gold. Underground Inferred Mineral Resources are estimated to be 24.6 Mt grading 3.87 g Au/t for 3.1 Moz of gold.
  - The Brookbank, Kailey and Key Lake Deposits show potential to be converted to Mineral Reserves and provide further plant feed in later years of production. The Brookbank underground deposit has the potential to be integrated into the Hardrock Mine Plan to increase the overall head grade of the combined operations.
- Mining and Mineral Reserves:
  - The mine design and Mineral Reserve estimate have been completed
  - At a cut-off grade of 0.35 g Au/t, the Proven Mineral Reserves total 5.6 Mt at an average grade of 1.28 g Au/t for 232 k in-situ ounces of gold. The Probable Mineral Reserves total 129.7 Mt at an average grade of 1.27 g Au/t for 5,307 k in-situ ounces of gold. The total Proven and Probable reserve is 135.3 Mt at an average grade of 1.27 g Au/t for 5,539 k in-situ ounces of gold.
  - The mining activities will occur over a period of 12.9-year (from start of commercial production to end of in-pit mining activities), excluding the pre-production period.
  - The open pit generates 689.6 Mt of overburden and waste rock (inclusive of historic tailings and underground backfill) for a strip ratio of 5.1:1.

- Metallurgical Testing and Mineral Processing:
  - The process design criteria have been established based on test results, Owner and Vendor recommendations and on industry practices.
  - Processing options for the Project were selected based on the results of this testwork and are well known technologies currently used in the mining industry.
  - Trade-off studies have determined that the ore processing plant should be sized at 27,000 t/d and using standard proven technology for crushing, grinding, thickening, leaching, CIP and gold recovery.
  - The gold recovery process for the Project consists of a crushing circuit, a HPGR/ball mill grinding circuit, pre-leach thickening and cyanide leaching; CIP circuit, carbon elution and regeneration; electrowinning and gold refining; cyanide destruction and tailings disposal. The process plant is designed to operate at a throughput of 27,000 t/d.
  - The overall gold recovery is 91.2% and is based upon metallurgical testing completed comprising of composite samples representing the full (global) deposit, early production years, lithological zones, low grade and near surface areas. The results demonstrate that the ore is amenable to gold recovery via cyanidation. Gold recovery is correlated to grind size, gold, sulphur and arsenic head grade. Block models have been created and each is assigned a gold recovery based upon the block attributes and the target grind size.
- Infrastructure:
  - Existing infrastructure within the footprint of the property limits will need to be relocated or purchased and dismantled. The most significant relocation is that of the TransCanada Highway 11. All private properties within the project area have been purchased.
  - Power availability from the existing grid is deemed insufficient and unreliable. Construction of a 65 MW natural gas-fired power plant is planned, with a designed capacity of 46.5 MW.
  - As with the other main infrastructure, the administration building, truck shop, reagent storage and explosives plant and tailings management facility have been sized to support the mine and process operation.
  - GFC which currently traverses through the TMF footprint will be permanently diverted towards northeast to Kenogamisis Lake. The permanent GFC diversion channel design meets the fish offset guidelines. The GFC diversion dyke required for the diversion is designed in accordance with CDA and LRIA guidelines.
  - TMF has been designed in accordance with LRIA and CDA guidelines. The stability of the dams meets the target factors of safety required as per CDA. Tailings deposition plans have been

developed in such a way that the wide tailings beaches abut the perimeter rock fill dams and pond being pushed to the west abutting natural ground.

- Seepage and runoff form TMF will be pumped back to the TMF. In case there is surplus water, which cannot be pumped into the TMF, water will be treated prior to discharging to the environment.

- Environmental Considerations:
    - The EIS/EA received Federal approval on December 13, 2018 and Provincial approval on March 12, 2019. The EIS/EA, including implementing the identified mitigation measures, supports the conclusion that the Project will not cause significant adverse environmental effects, including effects from accidents and malfunctions, effects of the environment on the Project and cumulative effects.
    - Conceptual environmental management and monitoring plans ("EMMPs") were provided in the final EIS/EA and include measures related to both compliance and EIS/EA monitoring for all phases of the Project. The collective monitoring activities associated with the Project will also be used to inform adaptive management for the Project, as required. The management and monitoring requirements have been incorporated into Project plans and budgets.
    - Active consultation with stakeholders (community members, agencies and interested parties) and Indigenous communities has been undertaken throughout Project planning and will continue as the Project progresses.
    - GGM has established LTRAs with the five local Indigenous communities. The agreements establish increased clarity regarding GGM's ability to develop the Project and the Indigenous communities' opportunity to benefit from future mining opportunities in the region, including the potential to extend the life of the Project.
- Capital and Operating Costs:
    - The CAPEX estimate was based on material take-offs from detailed engineering in most areas (50% to 90%), and firm price bids for the majority of process plant, power plant equipment, and the mine mobile equipment fleet. A QRA session was held to establish the contingency for the CAPEX update.
    - The initial CAPEX for Project construction, including processing, mine equipment purchases and pre-production activities, infrastructure and other direct and indirect costs, is estimated to be CAD 1,301M (before pre-production revenue and LTRA costs). The total initial capital includes a contingency of CAD 108M, which is 9.0% of the total CAPEX.

- Sustaining capital required during operations for additional equipment purchases, mine equipment capital repairs, mine civil works, TMF dam raises and additional infrastructure relocation is estimated at CAD 420M.
- A salvage value of CAD 45M is estimated for some mining and processing equipment and the power plant that will not have been utilized to their useful life.
- The total reclamation and closure cost is estimated to be CAD 54M.
- The average operating cost is CAD 708/oz Au or CAD 26.51/t milled over the life of the mine. The all-in sustaining cost ("AISC") which includes closure, reclamation and sustaining capital costs average CAD 803/oz Au over the mine life.

1.16 Risks and Opportunities

GGM's risk identification and assessment process is iterative and has been applied throughout the FS and Detailed Engineering phases. Through a series of risk assessment sessions, risks are identified in relation to Project objectives and the internal and external context at the time of each assessment and are summarized into the Hardrock Project Corporate Risk Register. The risk evaluation process uses a 5 x 5 Impact and Likelihood matrix to rate risks at the enterprise level. A broad range of Project risk areas (health and safety, technical, environmental, community, financial, etc.) are assessed in order to provide a business or enterprise level perspective. Risk assessments by department or discipline are undertaken at the appropriate stages. Various standard engineering risk assessment processes, such as HAZOPs and Failure Mode and Effects Analysis ("FMEA") are undertaken during the detailed engineering phase. Health and safety risk assessment processes will be implemented for the construction phase.

Risk treatment plans are developed for each risk to reduce the risk's probability of occurring and / or impact to an acceptable or practical level. Certain risk mitigation activities were completed as planned during the current project phase, while other actions are planned for construction, operations or closure phases as appropriate. These mitigation plans are incorporated in the project execution plans and where required in the CAPEX and OPEX budgets. The key risks areas that are being managed through current controls and future phase mitigation plans include project execution people, gold production (process plant ramp-up), permitting approval timelines, tailings management facility, project costs, safety, pit wall failure, relocation of infrastructure and water management,

There are several opportunities to improve overall Project economics and sustainability.

- Revenue Related Potential Opportunities:
  - The use of the Hardrock process plant and TMF for the future processing of gold from other GGM properties including the Hardrock underground resource and the regional exploration to improve the LOM average grade and/or extend the LOM. The Indigenous Agreements include agreed financial benefits to allow GGM to develop Hardrock Underground.
  - The Project is permitted for 30,000 tpd providing the opportunity to increase throughput, post ramp-up through optimization. Additional throughput may be achieved through the milling circuit as a result of microcracks generated from the high-pressure grinding rolls; not accounted for in the throughput rate estimate. The use of the Hardrock process plant and TMF to process some portion of the existing surface historic tailings in order to recover gold, generate revenue, and also potentially mitigate environmental liabilities related to sulphides, arsenic and other contaminants.
  - Connecting the natural gas power plant to the grid and selling spare power generation to the grid during times of shutdowns or excess capacity.
- OPEX Related Potential Opportunities:
  - Consider the possibility of a potential blend of LNG and diesel as a fuel source is possible for the mine haul trucks. Currently, the mine fleet uses 100% diesel.
  - Utilize new, commercially available technologies/autonomous haulage to increase operational effectiveness and reduce costs (Year 3+).
  - Optimize remote assisted drilling to its full potential. While the base case includes remote assisted drilling, additional benefits can be achieved via labour mine productivity improvements.
  - Investigate the availability of high-quality second-hand mining equipment.
- CAPEX Related Potential Opportunities:
  - Pursue possible improvements in the TMF deposition plan and dam raise schedule, and the elimination of certain temporary structures in the initial construction in order to improve initial CAPEX.
  - Complete construction phase and/or ramp-up to commercial production earlier than planned.
  - Consider the possibility of equipment leasing to reduce upfront capital while protecting overall project economics.
  - Actively pursue government financial assistance for certain existing infrastructure relocations.

1.17 Recommendations

After the completion of the 2016 Hardrock Project Feasibility Study, GGM successfully completed work plans in 2017 and 2018 to further de-risk the project. Federal approval of the Environmental Assessment was received in December 2018 and Provincial approval was received on March 2019. Permitting work has advanced, and after thorough consultation with the affected Indigenous communities and agencies, all pre-construction permit applications have been submitted. All permits required for construction were submitted. The forecast approval timelines support the planned construction schedule. GGM has now signed agreements with the local Indigenous communities, and implementation of these agreements is underway. The Independent Tailings Review Board ("ITRB") was implemented in 2017 to provide advice and guidance during the TMF detailed engineering phase and extensive TMF geotechnical programs have been completed. Conceptual construction execution plans were developed for the TMF, Goldfield Creek and Highway 11 and the construction schedule for these facilities has been de-risked. The 2018 and 2019 drilling programs were successful and provided the basis for improved resource and reserve estimates.

The scope of the Technical Report Feasibility Study Update included updating the resource and reserve models, revising the mine plan, advancing detailed engineering in higher risk areas and completing firm price bid processes for all major process and power plant equipment, effluent and sewage treatment plants and the mine mobile equipment fleet. This work formed the basis of the capital cost, operating cost and project economic update. Activities related to establishing financing for the Project are underway.

The Technical Report Feasibility Study Update has reconfirmed the technical feasibility of the Project and significantly improved the economic results.

The list of preliminary recommendations that follows was prepared by GGM and GMS and reflects recommendations for subsequent phases of work, including completion of the detailed engineering phase, the construction phase and the operations phase. The cost of addressing each of these recommendations are generally within the scope of Project CAPEX, sustaining capital, closure costs and OPEX outlined in this Report.

1.17.1 Exploration and Geology

- Revalidate collar coordinates from the 2018 RC drilling campaign at Hardrock before the commencement of mining activities.
- Incorporate underground drill holes at Hardrock (not included in Mineral Resource) and adjust mineralization interpretation to produce a more locally accurate block model for internal purposes.

- Update the level of definition of the PAG (potentially acid generating) and non-PAG model.
- Undertake metallurgical test-work for the Kailey Deposit to confirm metallurgical recoveries assumed in the MRE.
- Retake core duplicates of existing Metalore-era drill core at Brookbank to confirm historical results where QA/QC protocols were lacking. Compile and digitise all QA/QC data for the Ontex-era drilling pre-2009 (present in drill logs and assay certificates).
- Resample drill core at Key Lake to increase the overall sample coverage and to overcome the effects of undersampling in the past.
- Undertake further resource definition drilling at Kailey, targeting the No.9 Zone near surface to convert existing Inferred to Indicated category, and to discover new ounces in the existing pit shell.
- Undertake a scoping study at Brookbank to understand if it could be a potential source of higher-grade plant feed for the overall Hardrock operation.

1.17.2 Detailed Engineering Phase

- Review specific sections of waste rock storage designs C and D based on the latest geotechnical stability analysis produced by Woods in September 2019.
- Review specific sections of the overburden storage design based on the latest geotechnical stability analysis produced by Woods in August 2019.
- Conduct additional pit slope geotechnical work such as detailed review of variation in structural fabric orientation to identify possible localized sub-domains with stronger controls on achievable bench face angles; and conduct sensitivity analyses on slope saturation and lower effective shear strength. Additional laboratory testing such triaxial testing and intact shear strength of foliation is recommended.

2. INTRODUCTION

On March 9, 2015 Centerra Gold Inc. (“Centerra”) and Premier Gold Mines Limited (“Premier”) formed a 50/50 partnership to facilitate the joint ownership, exploration and future development of the Trans-Canada Property (subsequently renamed the Greenstone Gold Property) (the “Property”), which includes the Hardrock Project (the “Project”). The partnership was originally called TCP Limited Partnership and was subsequently changed to Greenstone Gold Mines LP (the “Partnership”). The Partnership is managed by Greenstone Gold Mines GP Inc. which acts on behalf of the Partnership (“GGM”).

On December 15, 2020 the Orion Mine Finance Group (“Orion”), has entered into an agreement (the “Purchase Agreement”) with Centerra and Premier pursuant to which Orion will acquire Centerra's 50% interest in the Greenstone Gold Mines Partnership (“GGM”). On December 16, 2020 Equinox Gold Corp. and Premier entered into a definitive agreement (the “Agreement”) whereby Equinox Gold will acquire all of the outstanding shares of Premier. Equinox Gold will retain Premier’s interest in the world-class Hardrock Project in Ontario.

The Hardrock Project Feasibility Study (“FS”) was completed in 2016 and concluded that the Project is technically and economically feasible. Since the FS, GGM advanced Project de-risking activities, including approval of the EIS/EA, establishing the planned Long-Term Relationship Agreements, advancing on permitting activities, including the submission of all initial construction permit applications and completing a value engineering initiative.

As part of the 2019 approved workplan, GGM, GMS and several engineering consultants collaborated to prepare a Technical Report Feasibility Study Update. The scope of the Report included updating the resource and reserve models, revising the life of mine plan, advancing detailed engineering in higher risk areas and completing request for proposal (“RFP”) processes for all major process plant, equipment, power plant equipment, effluent and sewage treatment plants, the mine mobile equipment fleet as well as for the TMF, GFC and Highway 11.This work formed the basis of the capital cost, operating cost and project economic update.

The scope of this Report includes the geology and Mineral Resources of the Hardrock Property and the other properties (Brookbank, Kailey and Key Lake, collectively known as the “Hardrock Satellite Deposits”). The Mineral Reserves, mining, infrastructure, processing and financial analysis sections of this Report consider the Hardrock deposit only.

The Report and Project Update responsibilities of the engineering consultants follow:

- G Mining Services Inc. (“GMS”) - overall Report and integration, property description and location, accessibility, history, geological setting and mineralization, deposit types, exploration, drilling, sample preparation and security, data verification, Mineral Resource estimates, Mineral Reserves (pertaining to the Hardrock deposit only), mining methods, economic analysis, operating costs pertaining to mining, review of capital costs;
- Stantec Consulting Limited (“Stantec”) - climate and physiology, environmental, permitting unless otherwise noted, and social aspects;
- Soutex Inc. (“Soutex”) - metallurgical testing, recovery methods, mineral processing operating cost;
- Ausenco Engineering Canada Inc. (”Ausenco”) – processing plant and supporting infrastructure engineering;
- Wood Canada Limited (“Wood”) - Tailings Management Facility, Goldfield Creek diversion, and geotechnical engineering for the waste rock storage areas, TMF Closure Plan and permitting of TMF related facilities;
- SLR Consulting (Canada) Ltd. (“SLR”) – northside water management, mine rock stockpile pond design and mine site water balance.

A summary of the qualified persons (“QP”) responsible for each section of the Report is detailed in Table 2.1.

Table 2.1: Summary of Qualified Persons

QP	Company	Report Sections
Louis-Pierre Gignac, P.Eng.	GMS	1, 2, 3, 4, 5, 15, 16, 19, 21.1.5, 21.2.1, 21.3, 21.3.2, 22, 25.1, 25.1.2, 25.1.6, 25.2.3, 26.1.2, 27
Réjean Sirois, P.Eng.	GMS	1,6.1, 7, 8, 9.1, 10.1, 11.1, 12.1, 14.1, 23, 25.1.1, 26.1.1
James Purchase, P.Geo.	GMS	1, 7, 6.2-6.4, 8, 9.2, 10.2, 11.2, 12.2, 14.2, 25.1.1, 26.1.1
Mickey M Davachi P.Eng.	Wood plc.	Relevant parts of Subsections 1.9, 2.3, 12.3, 18.2, 18.3, 25.1.4, 25.2.2.4
Michael Franceschini, P. Eng.	Ausenco	18.1, 18.5, 18.6.1, 18.6.2, 18.6.3, 18.6.4, 18.6.5, 18.6.9, 18.6.10, 18.7.4.
Tommaso Roberto Raponi, P.Eng.	Ausenco	1.7,1.15,1.17.2, 17, 25.1.3
Michelle Fraser P.Geo.,	Stantec Consulting Ltd.	1.10, 20 (excluding 20.7), and 25.1.5
David Ritchie, P.Eng.	SLR Consulting Ltd.	1.8, 18.4.2, 18.4.3, 18.7.6, 18.4.4, 25.1.4, 25.2.2.10
Pierre Roy, P.Eng.	Soutex Inc.	1.6, 1.15, 13, 17.1.2, 17.1.3, 25.1.3

2.1 Sources of Information and Data

Unless otherwise stated, all the information and data contained in the Report or used in its preparation has been provided by GGM, and all currencies are expressed in Canadian dollars (CAD).

The QPs who prepared the Report relied on information provided by the following sources who are not QPs for this Report:

- SRK Consulting Inc. ("SRK") provided input regarding key parameters for the economic evaluation;
- Eagle Mapping Ltd provided 0.5 m accuracy topography in digital format of the Project area which was used to determine the ground surface shapes used in open pit and infrastructure / processing/ tailings management facility earthworks quantity estimates;
- SGS Minerals Services, ThyssenKrupp and SimSAGe provided metallurgical reporting and studies as referenced in Section 13 - Mineral Processing and Metallurgical Testing, managed principally by GGM;
- Golder Associates Ltd. ("Golder") – provided rock mechanics and open pit geotechnical studies as referenced in Section 16 - Mining Methods;
- TBT Engineering Limited ("TBT") - re-alignment of Trans-Canada Highway 11, Ontario Ministry of Transportation ("MTO") patrol station, MTO related permits;
- SGS Mineral Services provided laboratory geochemical and mineralogical testing, managed principally by GGM;
- Golder Associates Ltd. ("Golder") relied on the oriented core data collected by MD Engineering ("MDE") for the evaluation of the open pit geotechnical parameters and pit slope studies. Golder validated the MDE methodology and validated < 5% of the total oriented core. Golder has no reason to believe that the remainder were not also collected in a professional manner;
- Enbridge provided engineering studies, market information and a construction cost estimate for a natural gas pipeline from the TCPL Canadian Mainline pipeline to the Project site.
- Greenstone Gold Mines GP Inc ("GGM") – capital cost estimate compilation and basis of estimate, power generation, effluent treatment plants, infrastructure relocation estimates, construction indirects, schedule, project execution plan.

2.2 Site Visit

The following QPs visited the Project site as detailed below.

- Louis-Pierre Gignac, P.Eng., GMS, visited the site on June 3, 2014;
- Réjean Sirois, P.Eng., GMS, visited the site at numerous occasions since August 2016. The last visit was on August 5, 2020;
- James Purchase, P.Geo., GMS, visited the site from July 27, 2020 to July 30, 2020;
- Pierre Roy, P.Eng., Soutex, visited the site on June 3, 2014;
- David Ritchie, P.Eng., SLR, visited the site from July 2 to 3, 2014.

2.3 TMF Site Visits

The site visit revealed the necessity for field investigations to characterize the sub-surface conditions along the TMF dam footprint. Geotechnical investigations were undertaken during 2014, 2015, 2016, 2018 and 2019 which included test pitting, borehole drilling, cone penetration test and field vane shear tests. Disturbed and undisturbed samples were extracted and tested for various index, strength parameters in an approved laboratory. The stratigraphic and strength information of the subsurface units were reviewed, interpreted and utilized in the design of the TMF dams.

2.4 Units of Measure, Abbreviations and Nomenclature

The units of measure presented in this Report, unless noted otherwise, are in the metric system.

A list of the main abbreviations and terms used throughout this Report is presented in Table 2.2.

Table 2.2: List of Abbreviations

Abbreviations	Full Description
3SD	Three Standard Deviations
A	Ampere
AA	Atomic Absorption
ABA	Acid-Base Accounting
AECO	Alberta Energy Company
AERT	Aboriginal Environmental Review Team
Ag	Silver
AISC	All-In Sustaining Cost
Amec	Amec Foster Wheeler Americas Limited
APV	Aquatic Protection Value
ARD	Acid Rock Drainage
As	Arsenic
Au	Gold
AZA	Anumbiigoo Zaagi'igan Anishinaabek
BGB	Beardmore-Geraldton Greenstone Belt
BIF	Banded-iron Formation
BNA	Bongwi Nevaashi Anishinaabek
BWI	Ball Mill Work Index
BZA	Biinjitiwaabik Zaaging Anishinaabek
°C	Degree Celsius
C	Carbon
Ca	Calcium
CAD	Canadian Dollar
CAPEX	Capital Expenditures
CCTV	Closed circuit television
CEA	Canadian Environmental Assessment
CEAA 2012	Canadian Environmental Assessment Act 2012
CHP	Combined Heat and Power
CHVI	Cultural Heritage Value or Intertest
CIA	Cultural Impact Assessment
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIM Definitions	CIM Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014
CIP	Carbon in Pulp
CN	Cyanide

Abbreviations	Full Description
COG	Cut-off Grade
CoV	Coefficient of Variation
CRM	Certified Reference Material
CSD	Critical Solid Density
Cu	Copper
DCF	Discounted Cash Flow
DD	Diamond Drilling
DDH	Diamond Drill Hole
DGPS	Differential Global Positioning System
DTH	Down-the-hole
DWT	Drop Weight Test
EA	Environmental Assessment
EAA	Environmental Assessment Act
EDF	Environmental Deign Flood
E-GRG	Extended Gravity Recoverable Gold
EIS	Environmental Impact Statement
EM	Electromagnetic
EMP	Environmental Management Plans
ETP	Effluent Treatment Plant
˚F	Degree Fahrenheit
FA	Fire Assay
Fe	Iron
FEL	Front-End-Wheel Loaders
FS	Feasibility Study
Ft	Foot or Feet
G	Giga - (000,000,000’s)
g	Gram
g/t	Grams per tonne
g Au/t	Grams of gold per tonne
g/L	Grams per litre
G&A	General & Administration
GGM	Greenstone Gold Mines GP Inc. (the managing partner) and Greenstone Gold Mines LP (the partnership), collectively referred to as Greenstone Gold Mines
GHG	Greenhouse gas
GMS	G Mining Services Inc.
gpm	Gallons per minute (US)

Abbreviations	Full Description
GPS	Global Positioning System
GRG	Gravity Recoverable Gold
h/d	Hours per day
h/wk	Hours per week
h/y	Hours per year
ha	Hectares
h	Hour
HCl	Hydrochloric Acid Solution
HDPE	High-Density Polyethylene
HG	High Grade
HONI	Hydro One Networks Inc.
hp	Horsepower
HPC	Hazard Potential Classification
HPGR	High Pressure Grinding Rolls
HSE	Health, Safety and Environmental
HVAC	Heating, Ventilation and Air Conditioning
Hz	Hertz
ICMI	International Cyanide Management Institute
ICPAES	Inductively Coupled Plasma Atomic Emission Spectroscopy
ICPMS	Inductively Coupled Plasma Mass Spectroscopy
ID3	Inverse Distance Cube Interpolation
IDF①	Inflow Design Flood
IDF②	Intensity Duration Frequency
IEC	International Electrotechnical Commission
IESO	Independent Electricity System Operator
in	Inch (imperial unit)
IP	Induced Polarization
IR	Information Requests
IRR	Internal Rate of Return
ISO	International Organization for Standardization
IT	Information Technology
ITRB	Independent Tailings Review Board
JV	Joint Venture
k	Kilo - (000's)
kg	Kilograms

Abbreviations	Full Description
kg/t	Kilograms per tonne
koz	Thousands of troy ounces
kV	Kilovolts
km	Kilometre
km/h	Kilometre per hour
kPa	Kilopascal
KPIs	Key Performance Indicators
kV	kilovolt
kW	Kilowatt
kWh	Kilowatt hour
kWh/t	Kilowatt hour per tonne
L	Litre
LAA	Local Assessment Areas
LEL	Lowest Effect Level
LG	Low Grade
LIMS	Low Intensity Magnet Separation
LNG	Liquid Natural Gas
LOM	Life of Mine
LTRAs	Long Term Relationship Agreements
M	Mega or Millions (000,000's)
MARC	Maintenance and Repair Contract
masl	Metres above sea level
m	Metre
m/min	Metre per minute
m/s	Metre per second
m^2	Square metre
m^3	Cubic metre
m^3/h	Cubic metre per hour
MCC	Motor Control Centers
mg	Milligram
MG	Medium Grade
mg/L	Milligram per litre
MHT	MacLeod High Tailings
min	Minute
ml	Millilitre
mm	Millimetre

Abbreviations	Full Description
MMAH	Ministry of Municipal Affairs and Housing Act
MMER	Metal Mining Effluent Regulations
MNDN	Ministry of Northern Development and Mines
MNO	Métis Nation of Ontario
MNRF	Ministry of Natural Resources and Forestry
mo	Month
MOECC	Ministry of the Environment and Climate Change
Moz	Millions of troy ounces
MOWL	Maximum Operating Water Level
MPa	Megapascal
MRE	Mineral Resource Estimate
Mt	Million tonnes
MTO	Ministry of Transportation - Ontario
MVA	Megavolt-ampere
MW	Megawatt
N	Newton
NaCN	Sodium Cyanide
NI 43-101	National Instrument 43-101 - Canadian Standards of Disclosure for Mineral Projects
Non-PAG	Non-Potentially Acid Generating
NPI	Net Profit Interest
NPV	Net Present Value
NQ	Drill Core Diameter (47.6 mm)
NSR	Net Smelter Return
NTS	National Topographic Systems
NVR	Network Video Recorder
Ø	Diameter
OG	Original
OK	Ordinary Kriging Methodology
OPEX	Operating Expenditures
OPP	Ontario Provincial Police
O.Reg	Ontario Regulation
oz	Troy Ounce (31.10348 grams)
P80	Dimension, in size distribution, for which 80 percent of the material is smaller
PAG	Potentially Acid Generating
PDA	Project Development Area

Abbreviations	Full Description
PEA	Preliminary Economic Assessment
PFS	Pre-feasibility Study
Premier	Premier Gold Mines Limited
Pb	Lead
PMF	Probable Maximum Flood
PLC	Programmable Logic Controller
POE	Power Over Ethernet
POX	Pressure Oxidation
ppb	Parts per Billion
ppm	Parts per Million
psi	Pounds per square inch
PV	Present Value
PWQOs	Provincial Quality Objectives
QA/QC	Quality Assurance/Quality Control
QP	Qualified Person
R&D	Research and Development
RA	Repeat Assays
RAA	Regional Assessment Area
RC	Reverse Circulation
RoM	Run-of-mine
RQD	Rock Quality Designation
RPA	Roscoe Postle Associates Inc.
rpm	Revolutions per minute
RSMIN	Red Sky Métis Independent Nation
RWI	Rod Mill Work Index
S	Sulfur
SAG	Semi-autogenous Grinding
SAR	Species at Risk
SCC	Standard Council of Canada
Sec	Second (time)
SEL	Severe Effect Level
SMC	SAG Mill Comminution
SMU	Selective Mining Unit
SOCC	Species of Conservation Concern
SPT	Standard Penetration Tests
STP	Sewage Treatment Plant

Abbreviations	Full Description
t	Tonnes (1,000 kg) (metric ton)
t/y	Tonnes per year
t/d	Tonnes per day
t/h	Tonnes per hour
t/m3	Tonnes per cubic metre
TBTE	TBT Engineering Limited
TCPL	TransCanada-PipeLines Limited
TK	Traditional Knowledge
TMF	Tailings Management Facility
ToR	Terms of Reference
TRLU	Traditional Land and Resource Use
TS	Transmission Station
µm	Micron (10-6 metre)
UCoG	Underground Cut-off Grade
USD	United States Dollar
V	Volt
VC	Valued Components
VFD	Variable Frequency Drive
VLF-EM	Very Low Frequency Electromagnetic
VSA	Vacuum Swing Adsorption
WHIMS	Wet High Intensity Magnetic Separation
wk	Week
WRSAs	Waste Rock Storage Area
WSP	WSP Canada Inc.
XRF	X-ray Fluorescence
y	Year
% w/w	Percent weight by weight

3. RELIANCE ON OTHER EXPERTS

This Report has been prepared by G Mining Services Inc. ("GMS") for Premier Gold Mines Limited ("Premier"). The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to GMS at the time of preparation of this Report;
- Assumptions, conditions, and qualifications as set forth in this Report;
- Data, reports, and other information supplied by Greenstone Gold Mines ("GGM") and other third-party sources.

For the purpose of this Report, GMS has relied on ownership information provided by GGM. GMS has not researched property titles or mineral rights for the Project and expresses no opinion as to the ownership status of the property.

GMS has relied on GGM for guidance on applicable taxes, royalties, and other government levies or interests, applicable to revenue or income from the Project.

Except for the purposes legislated under Canadian provincial securities law, any use of the Report by any third part is at that party's sole risk.

4. PROPERTY DESCRIPTION AND LOCATION

Greenstone Gold Mines ("GGM") Greenstone Gold Property (formerly the Trans-Canada Property) includes three blocks of contiguous claims known as the Hardrock, Brookbank and Viper areas. The Hardrock Project is located within the southeast portion of the Hardrock claim block.

4.1 Location and Access

The property is situated in the Thunder Bay Mining Division of Ontario, with all claims located on NTS sheets 42 E/10 and 42 E/11. The property is located approximately 275 kilometres ("km") northeast of the city of Thunder Bay, Ontario and approximately 4 km south of the town of Geraldton, Ontario. The city of Thunder Bay has a population of 110,000 and provides support services, equipment and skilled labour for both the mineral exploration and the mining industry. Rail, national highway, port and international airport services are also available out of Thunder Bay. The town of Geraldton has a population of approximately 2,400 and can provide basic support services such as food and lodging.

The Hardrock deposit area covered by the Mineral Resource estimate in this Report is located in the townships of Errington and Ashmore on NTS sheet 42E/10, approximately 4 km south of the town of Geraldton. The approximate geographic centre coordinates of the Hardrock deposit resource area are 49°40'47"N and 86°56'32"N (UTM coordinates: 504175.9E and 5503024N, NAD 83, Zone 16).

4.2 Property Description

As of November 2019, GGM's property holdings consist of three blocks of contiguous mining claims known as the Hardrock, Brookbank and Viper Project areas as show in Figure 4.2. The Hardrock Project is also known as the Kenogamisis property. The land tenure consists of cell claims, patented claims, mining leases, and licenses of occupation ("MLO") covering a total area of 39072.1 hectares ("ha") as summarized in Table 4.1. The properties are located in the townships of Lindsley, Errington, Ashmore, Parent, Salsberg and McKelvie in the Thunder Bay Mining Division. A leasehold patent of mining rights or of surface rights, or of both mining rights and surface rights is a conveyance or grant of possession of land for a set length of time. There is usually a requirement to pay rent.

There are several past producing gold mines on the Property, including the Hard Rock, MacLeod-Cockshutt, Mosher (all later combined as the Consolidated Mosher), Little Long Lac, Bankfield, Jellicoe and Magnet mines. There are also a number of less significant historical occurrences of gold mineralization within the property boundary. The mineralized zone that is host to the most recently delineated mineral resources is within or adjacent to the former Hardrock and MacLeod-Cockshutt Mines.

Figure 4.1: Location Map for the Hardrock Project

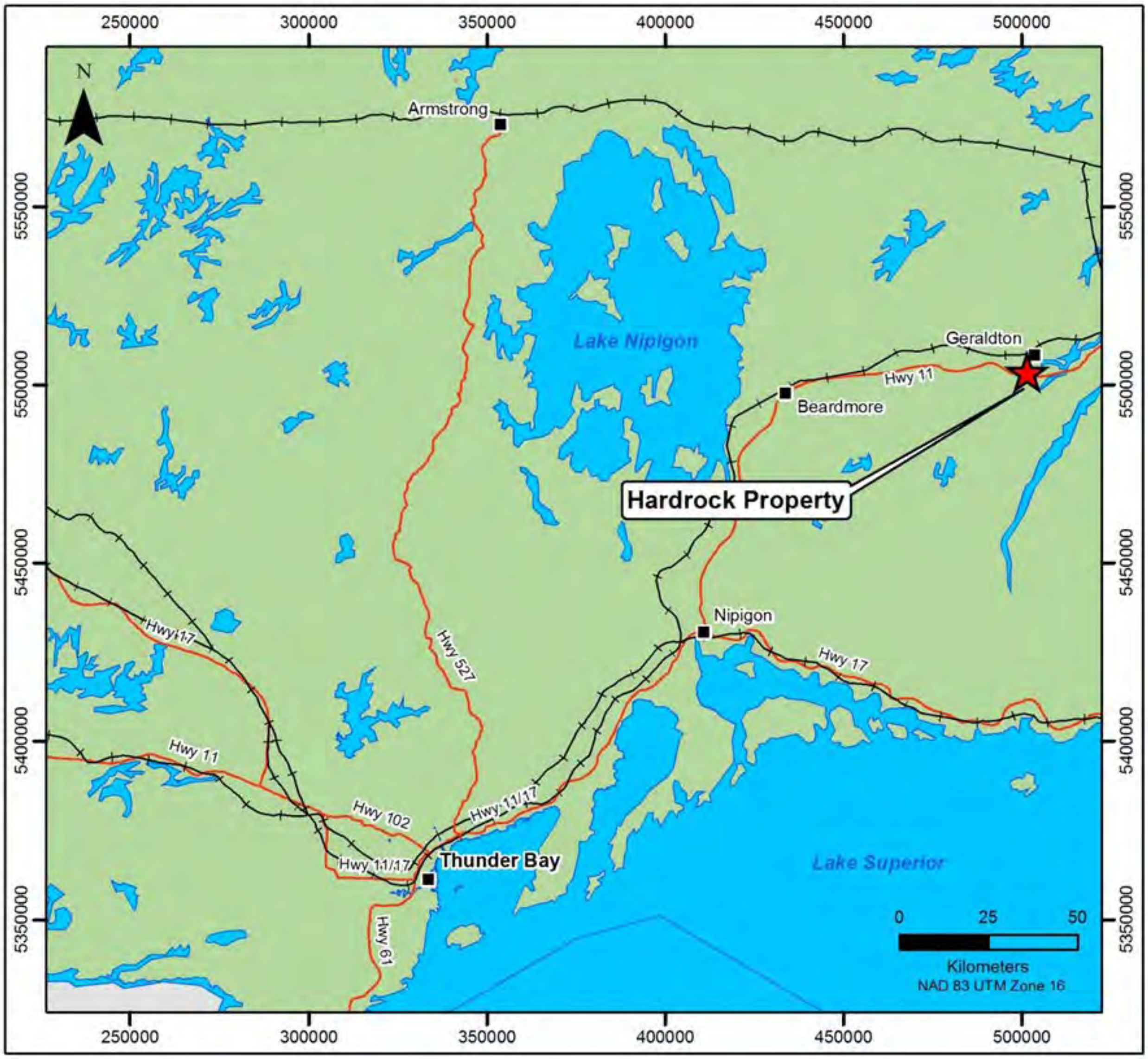
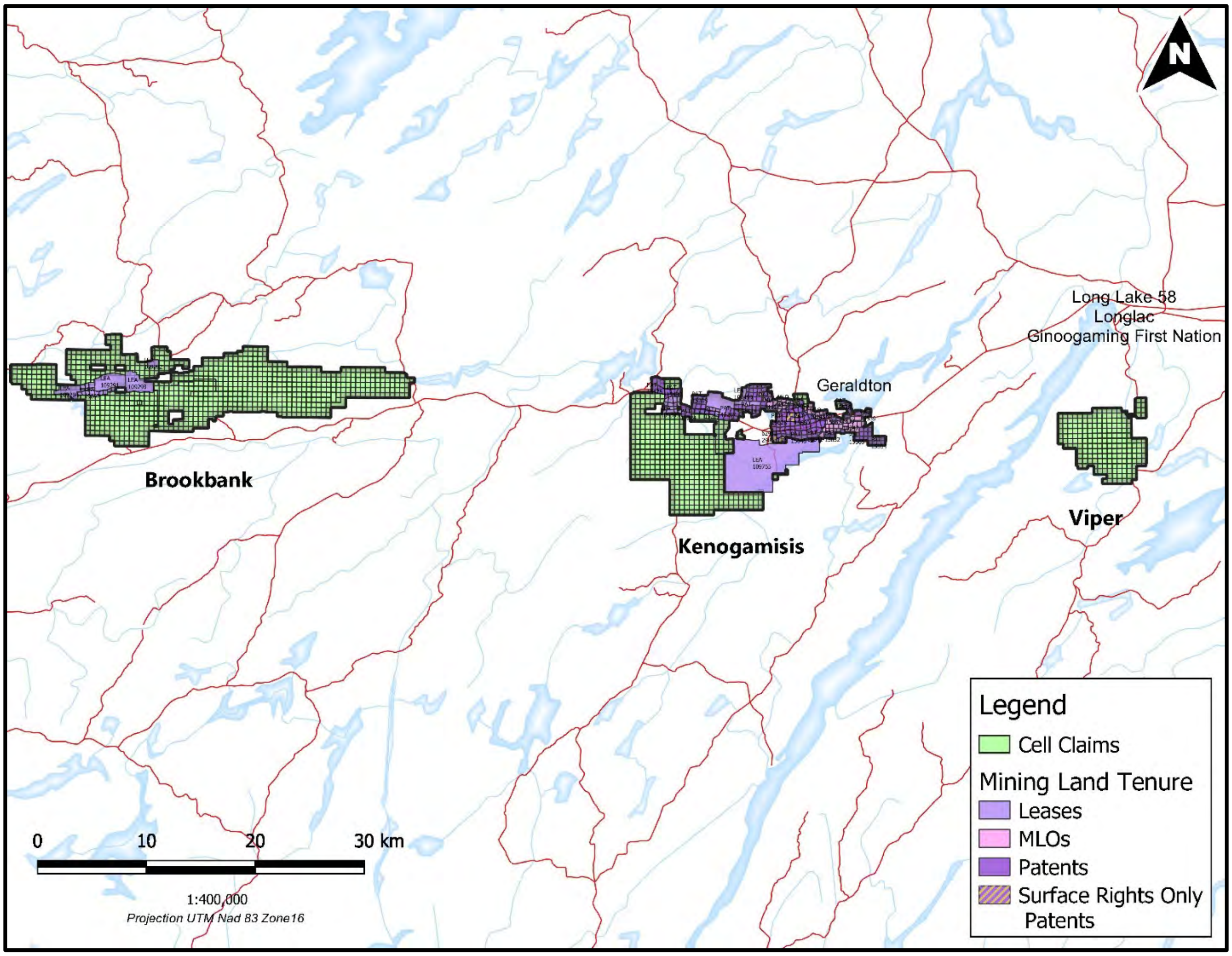


Table 4.1: Summary of Types of Land Tenure in GGM Land Package – as of November 6, 2019

Property	#Cell Claims	# Patents	# Leases	# MLOs	Area (ha)
Hardrock / Kenogamisis	447	191	23	78	15,862.7
Brookbank	920	0	19	0	18,958.5
Viper	216	0	0	0	4,250.9
Total	**1,583**	**191**	**42**	**78**	**39,072.1**

Figure 4.2: Overview of GGMs Land Tenure



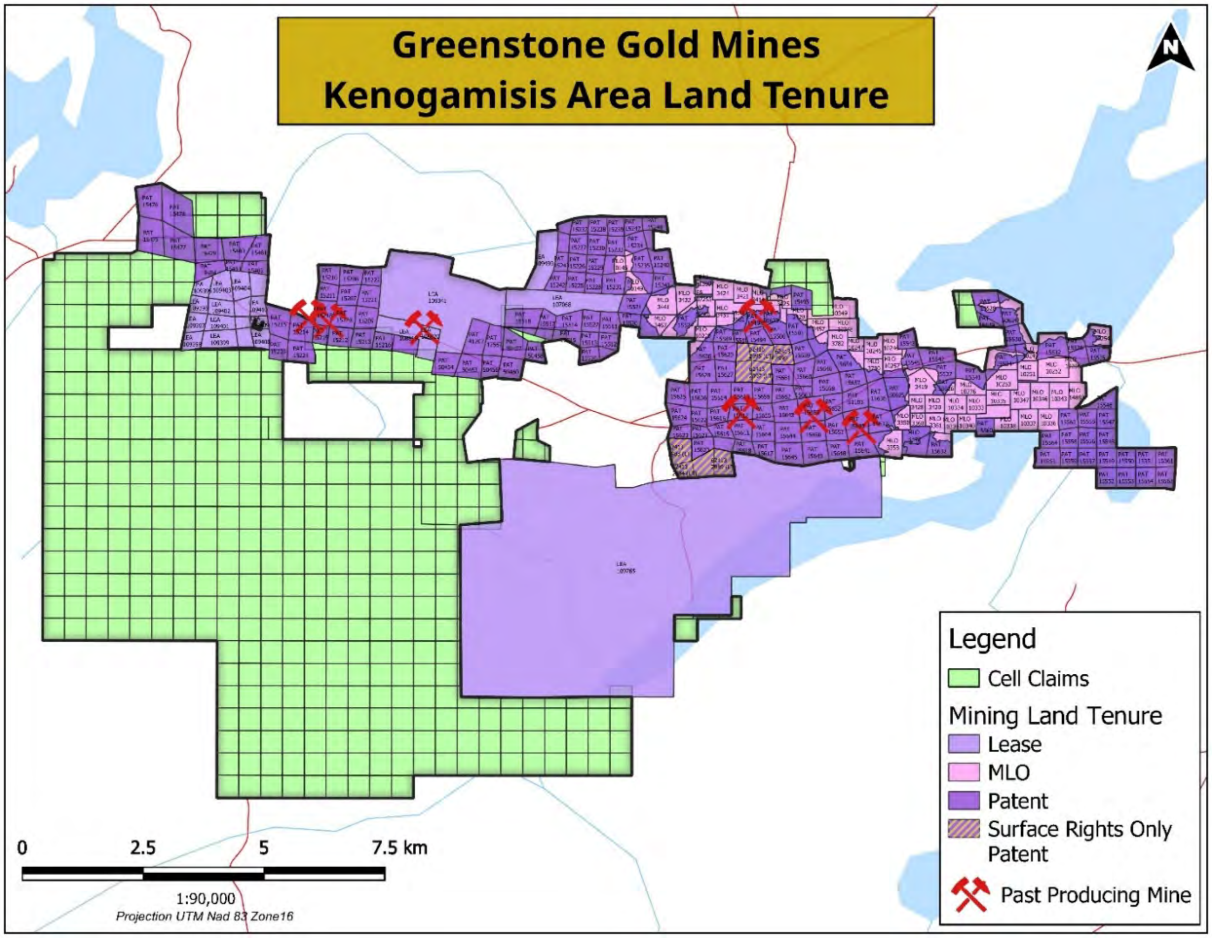
4.3 Hardrock Project Area

4.3.1 Hardrock Properties

The Hardrock properties consist of a 25 km long, east west striking package of cell claims, patents, leases and licenses of occupation totaling 15,862.7 ha as shown in Figure 4.3. This land package includes the set of claims previously referred to as the Key Lake property.

In October 2018, a mining lease was granted over CLM 535, which covers the southern part of the Hardrock Project area. The lease, LEA-109765, is subject to renewal in 2039.

Figure 4.3: Hardrock Project Properties

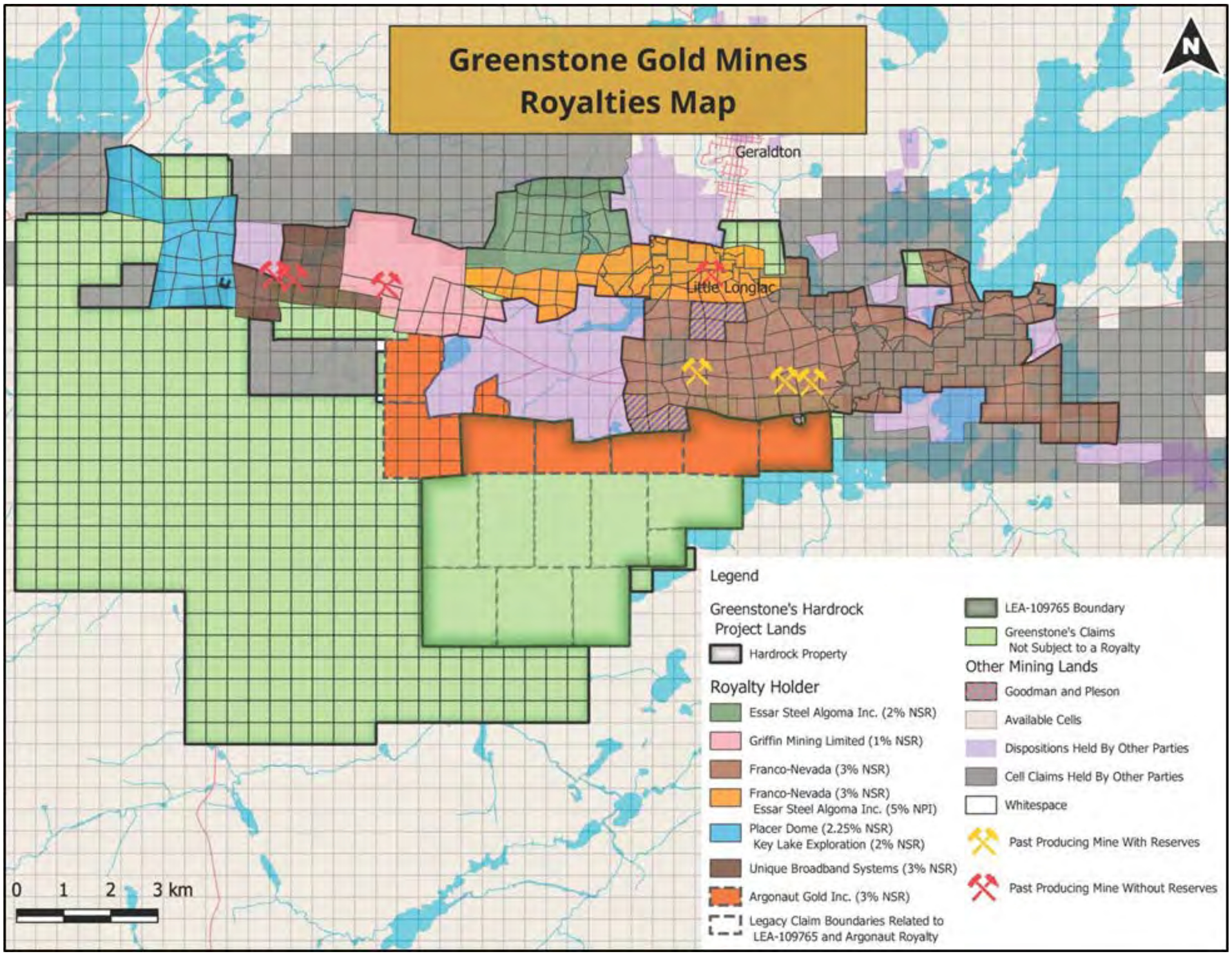


4.3.2 Hardrock Agreement Overview

The package is an amalgamation of multiple historical mining properties with several underlying agreements and royalties. Gignac et al. (2016) provides a detailed history of the various agreement and acquisitions. A summary of royalties currently in effect are listed below and shown in Figure 4.4.

- Essar Steel Algoma Inc. (2% NSR);
- Griffin Mining Limited (1% NSR);
- Franco-Nevada (3% NSR);
- Franco-Nevada (3% NSR) / Essar Steel Algoma Inc. (5% NPI);
- Placer Dome (2.25% NSR) / Key Lake Exploration (2% NSR);
- Unique Broadband Systems (3% NSR);
- Argonaut Gold Inc. (3% NSR).

Figure 4.4: Hardrock Project Royalties Map



4.3.3 Greenstone Gold Property Partnerships

On March 9, 2015, Centerra Gold Inc. ("Centerra") and Premier Gold Mines Ltd. ("Premier") formed a 50-50 partnership for the exploration, development and operation of the GGM properties. Greenstone Gold Mines GP Ltd. was formed to hold and manage the Partnerships assets. Centerra made an initial cash contribution to the partnership in the amount of CAD 85 M for its 50% limited partner interest. In accordance with the Partnership Agreement, Centerra committed to solely fund up to CAD 185 M in capital to develop the Hardrock Project, following which all funding for the Partnership would be made on a pro-rata basis.

On December 15, 2020 the Orion Mine Finance Group ("Orion"), has entered into an agreement (the "Purchase Agreement") with Centerra and Premier pursuant to which Orion will acquire Centerra's 50% interest in the GGM Partnership. On December 16, 2020 Equinox Gold Corp. and Premier entered into a definitive agreement (the "Agreement") whereby Equinox Gold will acquire all of the outstanding shares of Premier. Equinox Gold will retain Premier's interest in the world-class Hardrock Project in Ontario.

4.3.4 Agreement with Tombill Mines

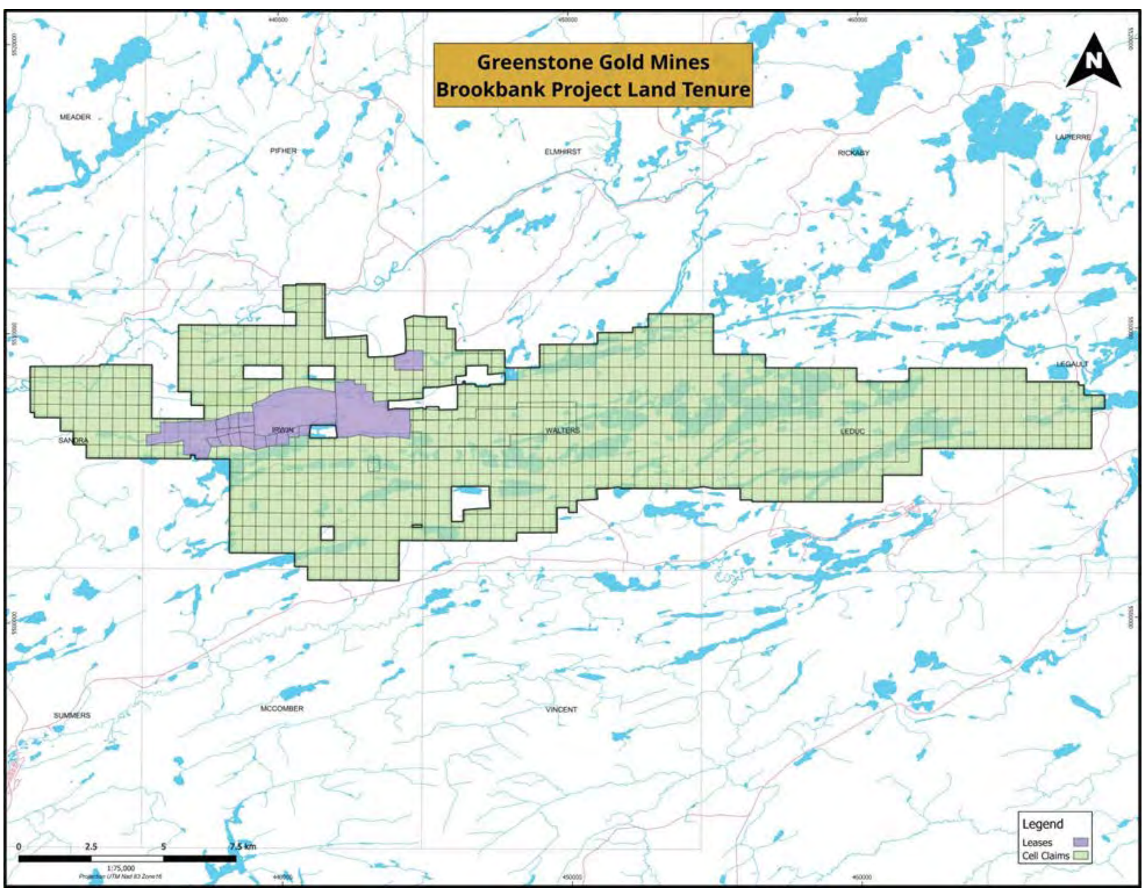
In December 2016, GGM acquired the surface rights for the following patented claims from Tombill Mines Ltd. – TB 10604 to TB 10608, TB 11879, TB 11885, TB 11886, and TB 11888 located in Errington and Ashmore townships.

4.4 Brookbank Project Area

The Brookbank Project area is located within 1; 50,000 scale NTS map sheet 42E/12 and lies 10 km to the northeast of the town of Beardmore as shown in Figure 4.6. By road, the project area is approximately 14 km east of Beardmore along the Trans-Canada Highway and 12 km north of the highway by gravel road. Beardmore is about 205 km by the Trans-Canada Highway northeast of the airport in Thunder Bay, Ontario. The project area hosts the Brookbank, Cherbourg and Fox Ear deposits, and the Irwin prospect.

The Brookbank Project consists of 19 mining leases and 920 staked claims totaling 18,958.5 ha.

Figure 4.5: Brookbank Project Properties



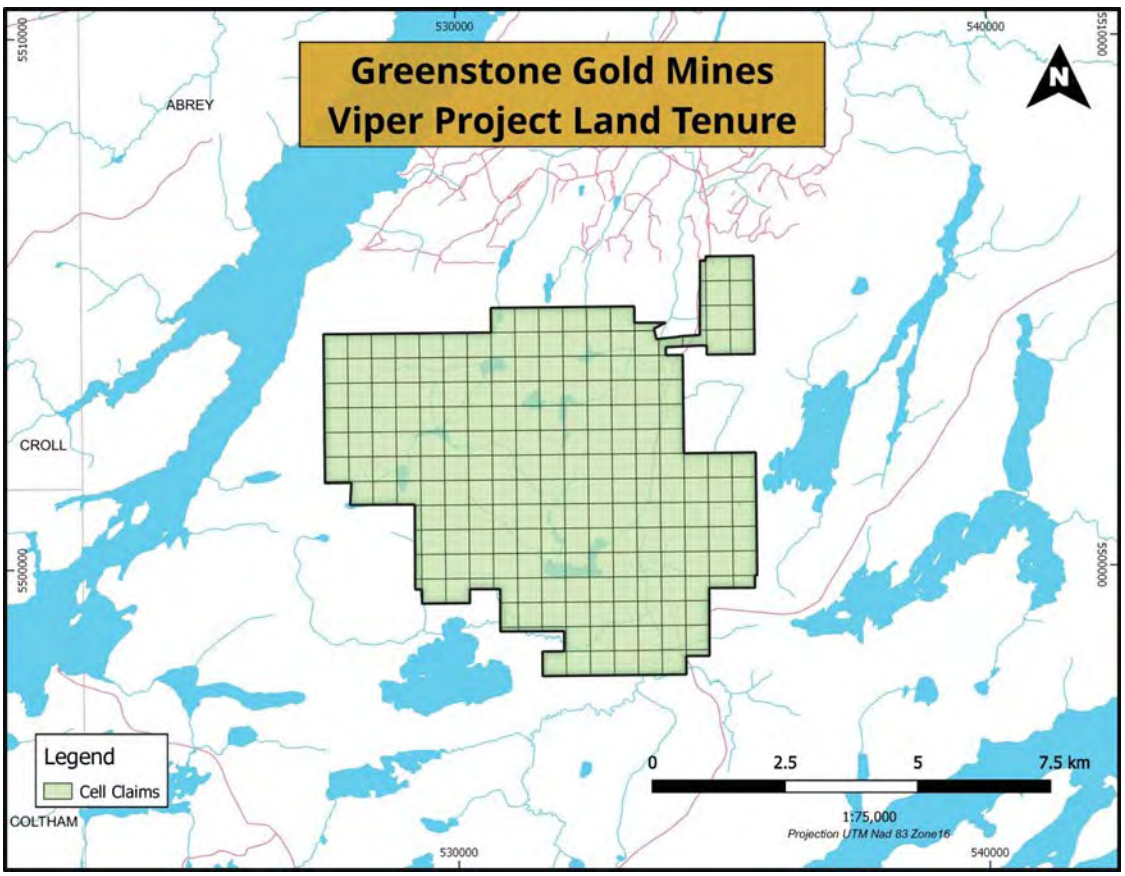
4.4.1 Brookbank Agreements

GGM owns 100% of the lease that covers the Brookbank deposit with the remaining portion of the project tenements being subject to two Joint Venture ("JV") agreements with Metalore Resources Limited ("Metalore"). The first JV is a GGM 74% / Metalore 26% split with the second a GGM 79% / Metalore 21% split.

4.5 Viper Project Area

The Viper claims were staked by Premier between 2013 and 2015 (Figure 4.5). The Viper claim group is 100% owned by GGM. The Viper Project is made up of 216 contiguous cell claims totaling 4,250.9 ha.

Figure 4.6: Viper Project Properties



4.6 Permits

Permits are required to undertake surface stripping and trenching, and drilling. Table 4.2 lists all the permits in place for the GGM properties as of September 9, 2020.

Table 4.2: Permits on GGM Properties

Permit	Permit No.	Issued by	Effective Date	Expiry Date
Closure Plan		MENDM	05-Apr-2012	
ECA (Air/Noise)	9088-94LRDR	MOE	29-Apr-2013	
ECA (Dewatering)	6096-SXZPUV	MOE	23-Apr-2012	
Land Use Permit	1176-1003064	MNR		
Encroachment Permit	EC-2014-61T-61	MTO	16-Mar-2014	16-Mar-2025
Exploration Plan	PL-19-000157	MENDM	09-Feb-2020	09-Feb-2022
Exploration Plan	PL-19-000158	MENDM	09-Feb-2020	09-Feb-2022
Exploration Plan	PL-19-000159	MENDM	09-Feb-2020	09-Feb-2022
Exploration Permit	PR-20-00089	MENDM	14-Aug-2020	13-Aug-2023
Exploration Permit	PR-20-00090	MENDM	14-Aug-2020	13-Aug-2023
Exploration Permit	PR-20-00091	MENDM	14-Aug-2020	13-Aug-2023
Hardrock Project				
Federal EA Approval	80068	IAA (formerly CEAA)	10-Dec-2018	N/A
Provincial EA Approval	EA-02-10	MECP	12-Mar-2019	N/A
Paragraph 35(2)(b) Fisheries Act Authorization	14-HCAA-00498	DFO	21-Nov-2019	31-Dec-2024
Release of Tree Reservation (55 of 60 patent applications)	Various	MNRF	Nov-2019	N/A
Closure Plan	N/A	ENDM	09-Jan-2020	N/A
Aggregate Permits for S1 pit	626463	MNRF	06-Mar-2020	N/A
Aggregate Permits for S4 pit	626461	MNRF	06-Mar-2020	N/A
Aggregate Permits for T2 pit	626462	MNRF	16-Mar-2020	N/A
PTTW for Freshwater Intake, North Side	7023-BNRLGZ	MECP	16-Jun-2020	16-Jun-2030
PTTW for Temporary ETP Phase	2855-BMBLGL	MECP	18-Jun-2020	18-Jun-2030
Amendment to Schedule 2 of the Fisheries Act	ECCC	N/A	20-Jun-2020	N/A
Land Use Permit under the Public Lands Act for freshwater intake pipe	NP2019-0448-LUP001	MNRF	01-Aug-2020	31-Jul-2030
Land Use Permit under the Public Lands Act for temporary ETP discharge pipe	NP2019-0459-LUP001	MNRF	01-Aug-2020	31-Jul-2030
ECA (ISW) for Temporary ETP Phase	0292-BMMK2E	MECP	31-Jul-2020	N/A
Work Permit under the Lakes and Rivers Improvement Act for onsite culverts	NP2020-0458-AP001	MNRF	21-Aug-2020	N/A

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

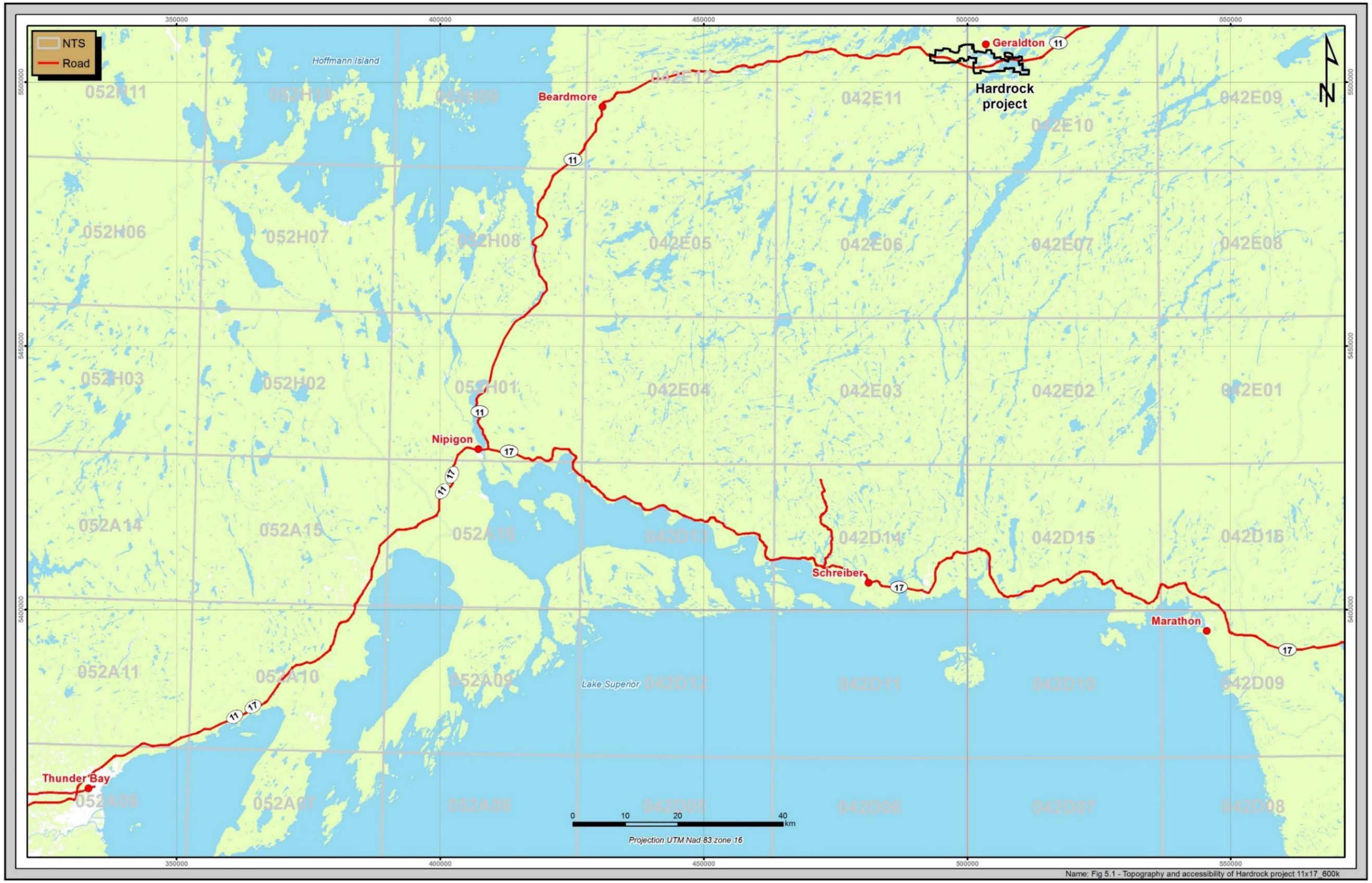
5.1 Accessibility

5.1.1 Hardrock

The Hardrock Project is located in the Municipality of Greenstone in the Province of Ontario, near the town of Geraldton. The area is accessible year-round via paved roads from Geraldton or Highway 11, which crosses the property from east to west (Figure 5.1). The closest major city is Thunder Bay, Ontario, located 275 km to the southwest, and it can be reached by Trans-Canada Highway 11. Public roads are maintained by various levels of government. Geraldton also hosts a municipal airport, which is equipped to accommodate small aircrafts.

The south portion of the Hardrock Project is accessed via Highway 11. The remainder of the Hardrock Project can be easily accessed by four-wheel drive vehicles via numerous logging/bush roads that branch off from the paved highways. Drill roads provide excellent access to the areas being explored by Greenstone Gold Mines ("GGM"). Those areas of the Hardrock Project not serviced by roads can be accessed by ATV, on foot or by boat during the summer and by snowmobile in the winter.

Figure 5.1: Hardrock Project Access Routes



Source: Innovexplo, 2015

5.1.2 Brookbank / Key Lake / Kailey / Viper

The Brookbank, Key Lake, Kailey and Viper projects are also located in the Municipality of Greenstone in the Province of Ontario, between the towns of Beardmore and Geraldton. The main part of the Brookbank property can be accessed via the Wendigokan Road, an all-weather gravel road which connects to Highway 11. The eastern part of the Brookbank property is accessible by Highway 801, a paved secondary road which also connects to Highway 11.

5.2 Climate

The Project is located in northern Ontario, which has a continental climate typical for temperate regions in the mid-latitudes that are influenced by both polar and tropical air masses. In this climate, seasonal temperature variations are represented by short summers and cold winters.

The nearest permanent weather monitoring station is located approximately 14 km north of the Project at the Greenstone Regional Airport, which services Geraldton and the surrounding area. Weather statistics for the period between 1971 to 2000 indicate a mean daily temperature of 3.9°C. Temperature ranges between a maximum of 37°C and a minimum of -50.2°C with a mean annual rainfall of 546.4 mm and the mean annual snowfall of 244.5 cm. On average, precipitations were recorded for 167 days during the course of a year. The annual average relative humidity in the morning is about 83.6%. The annual average wind speed for the area is about 11.2 km/h and the most frequent wind direction, on an annual basis, is from the west. In the summer, winds blow most frequently from the west and south, while in the fall to winter, the most frequent direction is from the west.

Weather conditions do not seriously hinder exploration and mining activities on the property, but adjustments on the type of work performed are subject to season variations, such as, geological mapping in the summer months and drilling in the winter months on frozen lakes.

5.3 Local Resources

The Hardrock Project benefits from local human resources and services in the town of Geraldton and surrounding areas. Geraldton has a population of approximately 1,900 people and is part of the Municipality of Greenstone, which also includes Longlac, Nakina, Beardmore and an extensive area of unincorporated territory. Greenstone itself has an approximate population of 4,700 people. Throughout all phases of the Project, GGM has done extensive consultation with local Indigenous communities and the town of Geraldton.

Although there has been no mining activity in the immediate area since 1970, the area has a workforce to support the future mining activities. Geraldton has all of the services typical for a town of that size including hospital, emergency services, school, sports centre, food, lodging, wireless, and wireline telecommunications.

5.4 Infrastructure

GGM has established a field office in the town of Geraldton near the Hardrock Project itself for core logging/cutting and core storage. This consists of an "Atco" style prefab office space, two office trailers (used primarily by the Environmental group), a core cutting room with space for 2 diamond saws, temporary core storage, sampling area and sediment settling/water re-use area and a separate garage, which has been re-purposed for use as a core logging facility. Likewise, a garage at one of the former Hardrock residences serves as a depot for the RC samples. Off-site storage of core, RC rep samples and assay pulps and rejects is located at the Magnet Mine Site. A house in the MacLeod townsite contains all historical maps and sections from the mines in the JV area as well as exploration projects.

GGM has also established a second office in the commercial district of Geraldton for public relations.

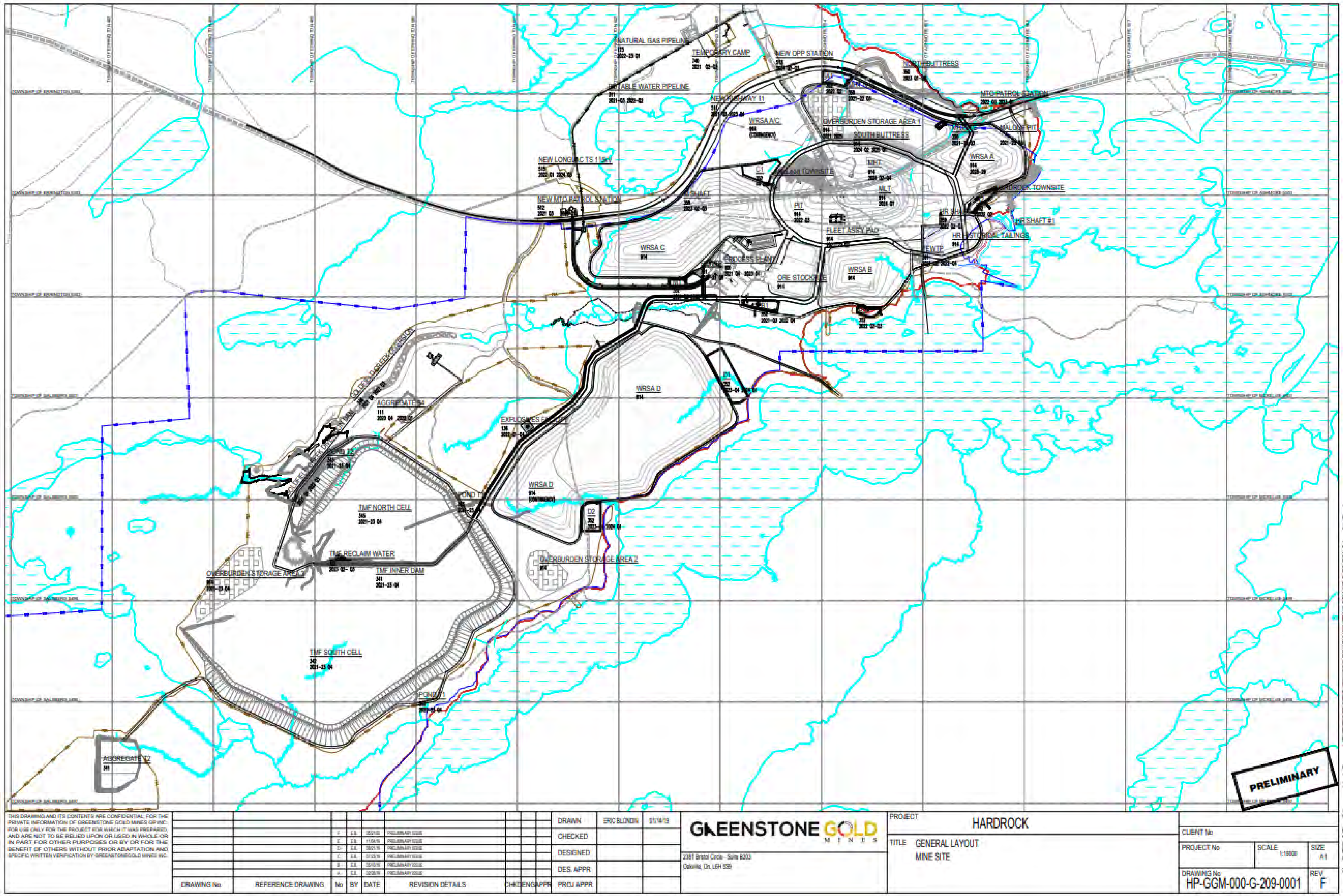
An independent sample preparation facility ("Actlabs") is located in Geraldton. In times of heavy sample flow, this facility may cooperate with the Actlabs facility in Thunder Bay to ensure timely turn-around.

Other significant infrastructure includes the Trans-Canada Highway, a TransCanada Pipelines Limited ("TransCanada") gas pipeline, a Hydro One electrical substation, and Geraldton hosts a municipal airport, which has a 1,500 m runway capable of accommodating large aircraft.

The Hardrock Project occurs in a mining friendly district with active mines and milling facilities located at Hemlo, Thunder Bay, Kapuskasing and Timmins with good transportation and regional mining related infrastructure.

There are adequate surface rights for the planned mining related infrastructure – waste rock storage areas, tailings management facility, processing and administration facilities as depicted in Figure 5.2. The arrangement of mining related infrastructure will be constrained by the surrounding lakes and watercourses.

Figure 5.2: General Site Layout



Note: North is upward and the grid is 1km by 1km

5.4.1 Water

The town of Geraldton has its own potable water treatment system and water distribution network. The GGM field office and houses in the Hardrock sub-division are currently serviced by this system. The plan is to use the Municipality's potable water for the Hardrock Project.

5.4.2 Sewage

The town of Geraldton has its own sewage treatment facility. The collecting network for sewage, however, does not come south of Kenogamisis Lake. Consequently, houses in the MacLeod and Hardrock townsites have their own septic beds.

5.5 Physiography

The Project lies within the Boreal Shield, a Canadian Ecozone where the Canadian Shield and the boreal forest overlap. Precambrian bedrock at or near the surface plays an important role in shaping the biophysical landscape. Lakes, ponds and wetlands are abundant in this landscape and drainage patterns are typically dendritic, with sporadic angular drainage as influenced by bedrock outcrops.

The topography in the Project area is relatively flat to gently rolling with local relief up to 20 m, largely attributed to glacial deposits that blanket the bedrock. There are no distinct topographic features that stand out in relief. Lower lying areas are characterized by swamps and ponds with overall very poor drainage throughout the area. The surrounding land has an altitude of about 335 masl. The largest lake adjacent to the Project is Kenogamisis Lake and it bounds the Project to the south, east and north. This lake elevation is about 330 masl.

5.6 Topography and Vegetation

The topography of the area is relatively flat with some gently rolling hills. Local relief ranges up to 20 m and that is largely due to glacial deposits that cover the bedrock. Lower lying areas are characterized by swamps and ponds with overall drainage in the area being very poor. The largest Lake on the Property is Kenogamisis Lake, which bounds the project area to the south, east and north.

Vegetation in the area is dominated by coniferous trees, with the most common tree species being black spruce, tamarack and cedar. There are local stands of birch, jack pine, and poplar in areas with better drainage, such as eskers and moraines.

6. HISTORY

6.1 Exploration History

Information described in this section was derived from the previous 2013 Technical Report on the Mineral Resource Estimates for the Hardrock, Brookbank and Key Lake Projects by Micon International Ltd. ("Micon") and the 2016 MRE update NI 43 101 Technical Report completed by G Mining Services Inc. ("GMS").

This section provides a summary of the historical work carried out on the Hard Rock, MacLeod-Cockshutt and Mosher Mines. Table 6.1 presents the statistics on gold production, diamond drilling and underground development for all three mines. A detailed chronological summary of the historical work carried out on these mines since 1980 is provided on Table 6.7.

The first gold discovery in the area of the Property was made between 1916 and 1918 when a gold-bearing boulder was discovered south of the Main Narrows of Kenogamisis Lake. In 1931, W.W. "Hard Rock" Smith discovered gold-bearing quartz stringers near the location of the Hard Rock Number One Shaft and Tom Johnson and Robert Wells discovered gold on Magnet Lake, which later hosted the Bankfield Gold Mine. Soon to follow was the discovery of gold by T. A. Johnson and T. Oklend in a small quartz vein along the southern shore of Barton Bay on Kenogamisis Lake which is now the location of the Little Long Lac Property.

In 1934, the period of mine production in the area began with the Little Long Lac Mine, which was the first successfully producing mine in the area. To the west of the 1931 Hard Rock discovery, F. MacLeod and A. Cockshutt staked claims and continually explored the area throughout the 1930's and 1940's. By the late 1940's the F-Zone, a low-grade, large-tonnage orebody in greywacke, was identified on both the MacLeod-Cockshutt and Hard Rock Properties.

Production on the Mosher Long Lac Mine (located west of, and immediately down plunge of the same mineralized zones exploited in the MacLeod-Cockshutt Mine) began in 1962, then in 1967, the MacLeod-Cockshutt, Mosher and Hard Rock Mines amalgamated and remained in production until 1970. The consolidated Hard Rock, MacLeod-Cockshutt and Mosher Mines produced 2,146,326 ounces of gold at an average grade of approximately 0.14 ounces of gold per ton (~14 Mt @ 4.9 g Au/t) in the period from 1934 to 1970.

In the 1980s, Lac Minerals Ltd. reviewed the remaining underground reserves and conducted litho-geochemistry, ground geophysical work and 15,240 m of diamond drilling in 77 holes to target areas with open-pit potential (e.g. Hard Rock D and F; North and South Porphyry; and Porphyry Hill zones).

In 1992, Asarco Exploration Company of Canada Limited ("Asarco") entered into a five-year earn-in agreement with Lac Minerals and in 1993 carried out a program of reverse circulation ("RC") overburden drilling and diamond drilling, the latter largely focusing on the near-surface portion of the F-Zone and targets along the plunging nose of the albite porphyry.

Figure 6.1: Map of the Hardrock Project - 2016 Resource Estimate Area (red outline) Representing Limits of Historical Work

Table 6.1: Gold Production, Diamond Drilling and Underground Development Statistics – Little Long Lac, Hardrock, MacLeod-Cockshutt, Mosher Long Lac and MacLeod Mines

	Past-Producing Mines					
Description	**Little Long Lac**	**Hard Rock**	**MacLeod-Cockshutt**	**Mosher Long Lac**	**MacLeod-Mosher**	**Total**
Years of Production	1934-1953	1938-1951	1938-1967	1962-1966	1967-1970	
Ore Milled (short tons)	1,780,516	1,458,375	9,403,145	2,710,657	1,656,413	**15,228,590**
Ore Milled (metric tonnes)	1,615,713	1,323,038	8,530,533	2,459,108	1,502,698	**13,815,377**
Au Grade (oz/ton)	0.34	0.185	0.145	0.122	0.109	**0.141**
Au Grade (g/t)	11.66	6.33	4.98	4.18	3.74	**4.83**
Gold Ounces	605,449	269,081	1,366,404	330,265	180,576	**2,146,326**
Silver Ounces	52,750	9,009	90,864	34,604	17,321	**151,798**
Total Length of Surface DDH (m)	2,114	14,021.40	16,933.50	1,083.00	0	**32,037.90**
Total Length of Underground DDH (m)	23,353	67,423.60	224,168.50	59,591.10	1,043.00	**352,226.20**
Total Length of Drifting (m)	Unknown	10,572.00	32,698.90	7,292.30	7,259.20	**57,822.40**
Total Length of Crosscutting (m)	Unknown	3,608.50	8,976.10	3,267.20	3,369.30	**19,221.10**
Total Length of Raising (m)	Unknown	1,878.50	10,589.70	2,467.40	4,300.10	**19,235.70**

Source: MRC 13 – Mineral Resource Circular; Ferguson, Groen and Haynes, 1971; Mason and White, 1986

As a result of this work, a geological resource was estimated for the Porphyry Hill, West and East pits as follows (Gray, 1994):

- Pit Resource: 1,920,000 short tons grading 0.079 oz Au/t (with strip ratio, including overburden, of 4.76:1);
- Ramp Resource: 1,160,000 short tons grading 0.127 oz Au/t.

These "Resources" are historical in nature and should not be relied upon. It is unlikely that they conform to current NI 43-101 criteria or to CIM definitions, and they have not been verified to determine their relevance or reliability. They are included in this section for illustrative purposes only and should not be disclosed out of context.

In August 1994, Lac Minerals was taken over by American Barrick Resources. In 1995, American Barrick Resources changed their name to Barrick Gold Corporation. Lac's former properties in the Geraldton area continued to be held by Lac Properties Inc., a wholly owned unit of Barrick.

Asarco continued their exploration program into 1994, completing reverse circulation holes in overburden, sonic holes in tailings, and an additional 40,000 ft of diamond drilling, mostly on the aforementioned targets (Gray, 1994). Cyprus Canada Inc. (“Cyprus”) assumed Asarco’s role in the Lac Minerals agreement in 1996 and drilled 24 holes, leading to the discovery of the B-Zone (Mason and White, 1997).

The agreement ended in 1997. Barrick, through Lac Properties Inc., began a rehabilitation program which continued until 2001. This saw the construction of the current Visitor’s Centre, re-contouring and seeding of the MacLeod tailings near Highway 11 and capping of old mine shafts.

In 2000, Golder was retained by Lac Properties Inc. (“Lac Properties”) to conduct a stability assessment of the F-Zone crown pillar of the MacLeod-Cockshutt mine (Telesnicki and Steed, 2007). From November 27 to December 12, 2000, Golder conducted a field investigation to determine whether caving had occurred above the stoping. One investigation borehole (369.5 m) was drilled in order to perform this investigation. The study also included a literature review of the properties of the mined material at the Hard Rock mine, rock mass classification of the rock core from the investigation borehole, an empirical analysis using the Scaled Span crown pillar stability assessment, an analysis using the CPillar crown pillar stability assessment, numerical modelling to determine the stability of the crown pillar using PHASE software, and a correlation of numerical modelling results with the field investigation and conclusions.

Investigation drilling at the MacLeod-Cockshutt mine allowed Golder to confirm that the crown pillar overlying the workings was intact at the time of the study. No unravelling or caving of the crown pillar above the working was observed. Classification of the rock mass overlying the workings indicated the quality to be "good" to "very good". Empirical, analytical and numerical modelling of the stability of the crown pillar overlying the mined zone indicated that the crown pillar was stable. Due to the depth of the mine workings and quality of the rock mass, it was not considered probable that significant caving could occur or would have an influence on the overlying ground surface.

In 2002, Golder was retained by Lac Properties to conduct a stability assessment of the crown pillar of the Hard Rock Mine (Soni and Steed, 2002). A total of 16 investigation boreholes (2,116.8 m) were drilled to determine whether caving in the crown of the stope had occurred. The study comprised a literature review of the properties of the mined material at the Hard Rock Mine, rock mass classification of the rock core from the investigation boreholes, an empirical analysis using the Scaled Span crown pillar stability assessment, an analytical analysis using the CPillar crown pillar stability assessment, numerical modelling to determine the stability of the crown pillar using PHASE software, and a correlation of numerical modelling results with the field investigation and conclusions.

Investigation drilling at the MacLeod-Cockshutt mine allowed Golder to confirm that the crown pillar overlying the workings was intact at the time of the study. No unravelling or caving of the crown pillar above the working was observed. Classification of the rock mass overlying the workings indicated the quality to be "good" to "very good". Empirical, analytical and numerical modelling of the stability of the crown pillar overlying the mined zone indicated that the crown pillar was stable. Due to the depth of the mine workings and quality of the rock mass, it was not considered probable that significant caving could occur or would have an influence on the overlying ground surface.

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Investigation drilling at the Hard Rock Mine indicated that the crown pillar overlying the workings was intact at the time of the study. No unraveling or caving of the crown pillar above the working was observed by Golder and no unexpected geometries were encountered. Classification of the rock mass overlying the workings indicated the quality to be "good". Empirical, analytical and numerical modelling of the stability of the crown pillar overlying the mined zone indicated the crown pillar to be stable, even when conservative values were used for stope geometries, strength, and rock mass classification, thus ensuring an additional built-in factor of safety.

In 2007, six geotechnical diamond drill holes totaling 1,208.1 m were drilled by Lac Properties in the crown pillars (Murahwi et al., 2011; 2013).

In 2007, Premier Gold Mines Limited (“Premier”) began signing various agreements to gain interest in the property and soon after began exploration drilling on the property (see Section 4).

Following Premier’s acquisition of Lac Properties’ claims in late 2008, a total of 91,802 m in 346 holes were drilled, with work focused on the North Iron Formation Area, the Hard Rock-Porphyry Hill Area and the Hard Rock-East Pit Area.

There were two areas where overburden stripping and related work were carried out. The GP-Zone, located north of the Trans-Canada Highway approximately one kilometre west of the Geraldton turn-off, was stripped, washed and sampled. No mapping was done. The second area, the TAZ-Zone, located approximately 1.5 km west-southwest of the Little Long Lac mine, was stripped, washed and sampled. No mapping was done.

In March 2010, Reddick et al. (2010) published a new Mineral Resource estimate for the Hardrock Deposit and a supporting NI 43-101 technical report. The technical report defined the Mineral Resources as several closely spaced zones considered best suited to open pit mining. The minimum cut-off grade, block size and depth below surface used to constrain the resources were applied with the assumption of a resource with bulk mineable characteristics. Contained metal and Mineral Resource estimates are summarized in Table 6.2.

Table 6.2: Mineral Resources - Hardrock Area (Reddick et al,. 2010)

Mineral Resources Class	Tonnage (Millions of tonnes)	Cut Au Grade (g Au/t)	Tonnage (Millions of short tons)	Cut Au Grade (oz/ton)	Contained Gold, Cut (oz)
Indicated	11.6	1.82	12.7	0.053	675,000
Inferred	7.3	1.81	8.1	0.053	425,000

In 2010, three different areas on the Hardrock Project were stripped:

- The East MacLeod Zone, which is located 500 m due east of the MacLeod-Cockshutt No.1 Headframe along the Trans-Canada Highway (stripping, washing, mapping and sampling).
- The Headframe Zone, which is located at the base of the MacLeod-Cockshutt No. 1 Headframe at the intersection of the Trans-Canada Highway 11 and Highway 584 (stripping and power washing).
- The Portal Zone, which is located 500 m southwest of the MacLeod-Cockshutt No. 1 Headframe (stripping, power washing, sampling). Gold grades ranged from trace values to 13 g Au/t. A structural study was carried out based on observations from the stripped outcrops and drill core.

A regional prospecting program was completed during the summer of 2010. Prospective targets were selected from regional magnetic anomalies. Prospecting covered the majority of the active claim group. Various regions of the property yielded gold values in trace amounts to 3 g Au/t.

Diamond drilling continued in 2010 on and around the old Hard Rock, MacLeod and Mosher mine sites. Drilling was accelerated in 2010 with 11 drills operating on the Hardrock Project in Q4. A total of 114,611 m was drilled in 279 holes. Some limited definition drilling was completed based on the 2009 data and, thereafter, regional exploration became a more important focus, with exploration on magnetic targets and other target surroundings historical mine sites on the property. The main zones that were drilled in 2010 were the North, F and SP-Zones. A new discovery was made, namely the F2-Zone. The F2-Zone was originally discovered when the bottom level was drifted on the 13th level. No follow-up occurred below that level.

In 2011, Premier drilled 204 DD holes with a total length of 107,413 m. The drill program expanded the SP-Zone and F-Zone, and identified new discoveries including the high-grade Tenacity South Zone.

The zones mentioned above are described in detail in Subsection 7.5.3.

Murahwi et al. (2011) prepared an updated Mineral Resource estimate for the Hardrock Deposit and a supporting NI 43-101 technical report. Contained Mineral Resource estimates from the report are summarized in Table 6.3.

Table 6.3: Mineral Resources - Hardrock Deposit (Murahwi et al., 2011)

Material	Resource Classification	Cut-off Grade (g Au/t)	Estimated Gold Grade (g Au/t)	Tonnes	Contained Gold (oz)
Open Pit	Measured	0.83	2.446	6,865,000	540,000
Open Pit	Indicated	0.83	2.280	5,833,000	427,500
Open Pit	Measured + Indicated	0.83	2.370	12,698,000	967,500
Open Pit	Inferred	0.83	2.483	615,000	49,200
Underground	Measured	2.80	5.993	2,312,000	445,800
Underground	Indicated	2.80	5.827	5,757,000	1,078,500
Underground	Measured + Indicated	2.80	5.875	8,069,000	1,524,300
Underground	Inferred	2.80	5.397	6,187,000	1,073,500
OP + UG	Measured	-	3.340	9,177,000	985,800
OP + UG	Indicated	-	4.042	11,590,000	1,506,000
OP + UG	Measured + Indicated	-	3.732	20,767,000	2,491,800
OP + UG	Inferred	-	5.133	6,802,000	1,122,700

Premier drilled 125 DD holes between January and October 2012 for a total length of 68,549 m. Diamond drilling focused primarily on testing specific target areas of the Fortune Zone and its possible extensions, the HGN and P-Zones. The Fortune and HGN zones comprise multiple, en-echelon, narrow-vein veined zones located in close proximity to the historical Hard Rock Mine workings. The primary vein zones were identified over a plunge length of approximately two kilometres and appear to coalesce at depth but remain open further to the west.

A NI 43-101 technical report by Murahwi et al. (2013) presented an updated Mineral Resource estimate for the Hardrock Deposit. Contained metal and Mineral Resource estimates are summarized in Table 6.4.

Table 6.4: Mineral Resources - Hardrock Deposit (Murahwi et al., 2013)

Deposit	Cut-off Category	Mineral Resource Category	Tonnes (Mt)	Gold Grade (g Au/t)	Gold Ounces (000's)
Hardrock Project	Open Pit (0/P)	Measured (M)	12.737	1.41	576
		Indicated (I)	33.920	1.55	1,685
		Subtotal M & I	**46.657**	**1.51**	**2,261**
		Inferred	6.615	1.74	370
	Underground (U/G)	Measured (M)	0.315	5.84	60
		Indicated (I)	4.730	5.42	829
		Subtotal M & I	**5.045**	**5.48**	**889**
		Inferred	16.009	5.91	3,040

Between October 31, 2012 and August 9, 2013, a total of 153 DD holes (72,776.4 m) were drilled on the Hardrock Deposit. These holes were included in an updated Mineral Resource estimate prepared by InnovExplo Inc. (InnovExplo) in 2013 and presented in a NI 43-101 technical report (Brousseau et al., 2013). Premier released the updated Mineral Resource estimate on October 29, 2013. Contained metal and mineral resource estimates are summarized in Table 6.5.

Table 6.5: Mineral Resources - Hardrock Deposit (Brousseau et al, 2013)

Resource Type	Parameters Cut-off (g Au/t)	Area		Total
		In-Pit > 0.50 g Au/t	Underground > 3.00 g Au/t	
Indicated	Tonnes (t)	50,228,100	5,522,200	55,750,300
	Grade (g Au/t)	1.46	5.01	1.81
	Au (oz)	2,351,947	889,022	3,240,968
Inferred	Tonnes (t)	17,792,500	16,918,700	34,711,200
	Grade (g Au/t)	1.50	5.38	3.39
	Au (oz)	858,982	2,925,065	3,784,047

Between August 10, 2013 and December 31, 2013, Premier added 144 DD holes on the Hardrock Deposit for a total of 66,606.7 m. None of these holes were included in the 2013 Mineral Resource estimate by Brousseau et al. (2013).

On March 2014, a Preliminary Economic Assessment ("PEA") for the Hardrock Project was published. The study results indicated that 89,332,152 t grading 1.18 g Au/t (3,392,559 oz Au) could be mined to surface over a nominal 15-year mine life (St-Laurent et al., 2014). The results of the financial analysis for the Hardrock Project indicated that the resource could be extracted at an estimated average operating cost of CAD 23.72/t and a total estimated (initial and sustaining) capital cost of CAD 767.89M. Using the consistent gold price of USD 1,250/oz and a currency exchange rate of CAD 1.00 = USD 0.95, the PEA stated the Project would generate a positive cash flow with an NPV of CAD 518.70M (discounted at 5%) and an IRR of 23% before taxes and CAD 358.97M (discounted at 5%) and an IRR of 19% after taxes.

Between January 1, 2014 and May 26, 2014, Premier added 38 DD holes on the Hardrock Deposit for a total of 12,653.6 m (Brousseau et al., 2014). Thirteen DD holes from 2013 were also deepened in 2014 representing a total of 2,867.3 m of new footage. Seven historical DD holes were re-sampled to add new assay results in the 2014 Mineral Resource estimates. These holes were not previously sampled and had therefore been rejected from the 2013 database (Brousseau et al., 2013). These holes represented a total of 5,709 m of new footage in the 2014 database. InnovExplo included the new data in its updated Mineral Resource estimate presented in a NI 43-101 technical report by Brousseau et al. (2014). Premier released the updated Mineral Resource on August 25, 2014.

Contained metal and Mineral Resource estimates are summarized in Table 6.6.

Table 6.6: Mineral Resources - Hardrock Deposit (Brousseau et al., 2014)

Resource Type	Parameters Cut-off (g Au/t)	Area		Total
		In-Pit > 0.50 g Au/t	Underground > 3.00 g Au/t	
Indicated	Tonnes (t)	83,867,800	5,169,300	89,037,100
	Grade (g Au/t)	1.47	5.40	1.70
	Au (oz)	3,972,542	897,814	4,870,356
Inferred	Tonnes (t)	10,225,080	12,921,700	23,146,700
	Grade (g Au/t)	1.53	5.40	3.69
	Au (oz)	501,349	2,242,288	2,743,638

Premier carried out two small drilling programs in the area of the past producing Bankfield Mine (Brousseau et al., 2014). The Bankfield Mine is located on the Hardrock Project in the west-central part of

Errington Township, extending into Lindsley Township and enclosing the southwest part of Magnet Lake. This historical mine is situated about 10 km west-southwest of the Town of Geraldton. Between December 15, 2013 and January 24, 2014, two DD drill holes were drilled for a total of 1,043 m. Between April 22, 2014 and May 17, 2014, six DD holes were added in this area totaling 2,513 m. None of these holes were included in the Mineral Resource estimate prepared by Brosseau et al in 2014.

June 1, 2014 through 2016, Premier conducted stripping in the 2014 resource area, east of MacLeod Shaft No. 1 (Brousseau et al., 2014). The work consisted of three stripped areas with detailed geological mapping and channel sampling. The channels were five meters apart in the east-west direction and sampled to the extent of the outcrop every one meter. The purpose of this work was to verify and establish structural elements and grade continuity at surface.

During 2014, a total of 128 mechanical test pits were completed on the Hardrock Project to evaluate the overburden thickness. Results of these test pits were not used in the 2014 Mineral Resource estimate update (Brousseau et al., 2015).

In February 2015, Premier and Centerra Gold Inc. formed a definitive 50/50 partnership for development of the Hardrock Property. In July 2015, the joint partnership was named Greenstone Gold Mines GP Inc ("GGM").

Through 2015, work continued to support a Feasibility Study to mine the Hardrock. A NI 43-101 compliant report was authored by GMS of Montreal, Qc., in December 2016. The Feasibility Study envisaged an open pit mining/milling operation processing 27,000 tpd of ore and producing an average of 279 koz of gold over a 15 year period, for a total of 4,193 koz produced from 4,647 koz contained. The pit would effectively mine out the area of the former Hardrock No. 2 Shaft and the MacLeod No.1 and No. 2 Shafts. The push back on the north wall of the pit would mine out a portion of the historic MacLeod tailings and would also require the relocation of Highway 11, the MTO garage and the OPP station. The pit and rock piles would effectively occupy a large portion of the historic MacLeod and Hardrock townsites. To this end, GGM embarked on a campaign to buy the surface rights from individual landowners in these areas. The Feasibility Study also included designs for rock piles, Tailings Management Facility ("TMF") and related run-off collection ditches and water treatment. The TMF would be located southwest of the mine and mill facilities. This would require the relocation of Goldfield Creek.

Total undiscounted cash flow, after taxes, was estimated at CAD 1,636M, or CAD 709M at a 5% discount rate, after total CAPEX of CAD 1,242M and sustaining capital of CAD 257M, generating an after-tax IRR of 14.4%.

Project design work continued through 2017. In 2018, a program of tightly spaced (20 m x 10 m grid) reverse circulation drilling was conducted over key areas to gain more detailed information on the grade continuity of the deposit. This resulted in 19,995 m drilled in 405 holes. As well, 62 blast holes, totaling 535 m were drilled to test penetration rates in the host rock. The results of this program are included in the current resource update.

Definition drilling continued in 2019, with 76 RC holes totaling 5,946 m and 54 NQ size diamond drill holes for 12,108 m being drilled. The results of this drilling are also included in the current resource estimate.

Table 6.7: Summary of Post-Production Exploration Activity

Year	Company	Activity	Comments*	Reference
1980	Long Lac Minerals Ltd	Studies of existing underground reserves; Lithological reconnaissance		Gray, 1994
1982	Lac Minerals Ltd Mining Corporation of Canada	"Ore reserves" and "ore potential" in the Hard Rock and MacLeod/Mosher mines	Historical "reserves" of 1,300,000 tons at 0.140 oz Au/ton (Proven and Probable ore) * 80% of total ore located below Level 13 of the 3.6 Mosher winze (No. 3 shaft) of the down-plunge of the F-Zone and South Zone mineralization	Jarvi, 1982
1987	Lac Minerals Ltd	Line cutting; Ground magnetometer, VLF EM, and IP surveys; Diamond drilling (37 DDH = 6,218.9 m)	DDH program targeted the open pit potential of the Hard Rock D and F-Zones, North and South Porphyry, and Homestake-Hill Several IP anomalies were partially tested	Gray, 1994 2012 Premier's Prospectus
1988	Lac Minerals Ltd	Diamond drilling (40 DDH = 9,052.6 m)	DDH program targeted the open pit potential of the Hard Rock D and F-Zones, North and South Porphyry, and Homestake-Porphyry Hill	Gray, 1994 2012 Premier's Prospectus
1993	Asarco Exploration of Canada Ltd Lac Minerals Ltd	106 reverse circulation overburden (RCO) drill holes (1,483.2 m); Diamond drilling (28 DDH = 5,125.2 m); Geological resource estimate	RCO drilling program was a reconnaissance test for anomalous gold values in glacial till Diamond drilling program tested IP targets associated with iron formations and the near-surface portion of the F-Zone Pit resource: 1,920,000 tons at 0.079 oz Au/t with strip ratio of 4.76:1 * Ramp resource: 1,600.000 tons at 0.127 oz Au/t *	Gray, 1994 Mason and White, 1993
1994	Asarco Exploration of Canada Ltd Lac Minerals Ltd	17 reverse circulation overburden (RCO) drill holes (395.6 m); 21 sonic drill holes (304.8 m); Diamond drilling (78 DDH = 11,961.9 m)	RCO drilling program was a reconnaissance test for anomalous gold values in glacial till Sonic drilling program tested the MacLeod-Mosher tailings Diamond drilling program consisted of infill drilling within a potential open pit zone (F-Zone, North Porphyry Zone, South Porphyry Zone, and No. 2 Vein) and testing of the near-surface portions of the C-Zone and North Zone.	Gray, 1994
1995	Asarco Exploration of Canada Ltd Lac Minerals Ltd	Pre-feasibility study; Mineral resource estimate	Pit resource: 2,900,000 tons at 0.086 oz Au/ton * Underground resource: 1,400,000 tons at 0.131 oz Au/ton *	Reddick et al., 2010 Mason and White, 1995b

Year	Company	Activity	Comments*	Reference
1995	Lac Minerals Ltd	Diamond drilling (7 DDH = 1,024.4 m)	Diamond drilling program to test some of the crown pillars of old stopes in the past producing mines	Murahwi et al., 2011 and 2012
1996	Lac Properties Inc. Cyprus Canada Inc	Diamond drilling (24 DDH = 1,024.4 m); Metallurgical work on the previous sonic holes; Samples from tailings; Environmental assessment work	Diamond drilling program defined the previous open pit area identified by Lac Minerals and Asarco	Reddick et al., 2010
1997	Lac Properties Inc. Cyprus Canada Inc	Diamond drilling (1 DDH = 185.0 m) Geological resource estimate	Pit resource: 9,800,000 tons at 0.047 oz Au/ton * Tailings resource: 11,200,000 tons at 0.023 oz Au/t*	Reddick et al., 2011
2000	Lac Properties Inc.	Diamond drilling (1 DDH = 369.5 m)	Diamond drilling program tested the F-Zone crown pillars at the past producing MacLeod-Cockshutt Mine	Telesnicki and Steed, 2007
2002	Lac Properties Inc.	Diamond drilling (16 DDH = 2,116.8 m)	Diamond drilling program tested some crown pillars at the past producing Hard Rock Mine	Soni and Steed, 2002
2008	Premier Gold Mines Limited	Acquisition of the Lac Claims		
2009	Premier Gold Mines Limited	Diamond drilling (346 DDH = 91,802 m); Overburden stripping with power washing, mapping and sampling	Diamond drilling program focused on the North Iron Formation Area, Porphyry Hill Area and East Pit Area Two areas were stripped (GP-Zone and TAZ Zone)	
2010	Premier Gold Mines Limited	Diamond drilling (279 DDH = 114,611 m); Overburden stripping with power washing, mapping, and sampling; Regional prospecting program	Three areas were stripped (East MacLeod Zone, Headframe Zone and Portal Zone) Diamond drilling focused on the same area as in 2009 The main zones drilled were North, F, SP, NN, and K Discovery of the F2 and Z zones New Mineral Resource estimate and a supporting NI 43-101 technical report	Premier Gold Reddick et al., 2010
2011	Premier Gold Mines Limited	Diamond drilling (204 DDH = 107,413 m)	Diamond drilling program resulting in the expansion of the SP, F, P and K zones Discovery of the Tenacity South Zone Updated Mineral Resource estimate and a supporting NI 43-101 technical	Premier Gold Murahwi et al., 2011
2012	Premier Gold Mines Limited	Diamond drilling (125 DDH = 68,549 m)	Diamond drilling program focused on the Fortune, HGN and P-Zones Updated Mineral Resource estimate and supporting NI 43-101 technical report	Premier Gold Murahwi et al., 2013

Year	Company	Activity	Comments*	Reference
2012/13	Premier Gold Mines Limited	Diamond drilling (153 DDH = 72,776.4 m) (from Oct. 31, 2012 to Aug. 9, 2013) (144 DDH = 66,606.7 m) (from Aug. 10, 2013 to Dec. 31, 2013	Updated Mineral Resource estimate and supporting NI 43-101 technical report	Premier Gold Brousseau et al., 2013
2014	Premier Gold Mines Limited	Preliminary Economic Assessment	Using the consistent gold price of USD1,250 per ounce and an exchange rate of CAD 1.00 = $US0.95, the Project generates an NPV of CAD 518.70M (discounted at 5%) and an IRR of 23.02% before taxes and CAD 358.97M (discounted at 5%) and an IRR of 19.02% after taxes.	Premier Gold St-Laurent et al., 2014
2014	Premier Gold Mines Limited	38 DDH = 12,653,6 m) (from Jan. 01, 2014 to May 26, 2014	Updated Mineral Resource estimate and supporting NI 43-101 Technical Report	Brousseau et al., 2014
2015	Premier Gold Mines Limited Centerra Gold Inc	Formation of a 50/50 Partnership	New NI 43-101 Technical Report	Premier Gold Brousseau et al., 2015
2016	Greenstone Gold Mines GP Inc.	Feasibility Study	Updated Mineral Resource estimate and supporting NI 43-101 technical report	Gignac et al., 2016
2018	Greenstone Gold Mines GP Inc.	RC Drilling 405 holes = 19,995 m, Blast hole drilling 62 holes = 535 m	Updated Mineral Resource estimate (not published)	Sirois, 2018
2019	Greenstone Gold Mines GP Inc.	Drilling 76 RC holes = 5,946 m, 54 DDH = 12,108 m	Resource update and project design work (this study)	

*Note: *Unless specifically indicated as reported in a NI 43-101 technical report, all “resources” listed in the table are historical in nature and should not be relied upon. It is unlikely they conform to current NI 43-101 criteria or to CIM definitions, and they have not been verified to determine their relevance or reliability. They are included in this section for illustrative purposes only and should not be disclosed out of context.*

6.2 Brookbank

6.2.1 Exploration History

The following summary of exploration activities on the Brookbank Project is derived from the 2013 Micon technical report which was mostly based on the Scott Wilson RPA (2009, now RPA) NI 43-101 technical report and is restricted to those leases and claims covering the Brookbank, Cherbourg and Foxear zones.

The earliest known work on the Brookbank Project is a program of surface trenching and limited diamond drilling carried out in 1934 by Connell Mining and Exploration Co. Ltd. ("Connell Mining"). A total of 17 trenches, plus numerous test pits, exposed a rusty shear zone in mafic flows over a strike length of 396 m. Gold values from samples in this zone were low and erratic, and the results for the diamond drilling are not known. Work was suspended in late 1935.

In 1944, Noranda Exploration Company Limited ("Noranda") completed detailed mapping, trenching and 1,860 m of diamond drilling in 40 holes to test the Brookbank Zone. Brookbank-Sturgeon Mines Limited ("Brookbank-Sturgeon"), a predecessor company to Ontex Resources Limited ("Ontex"), acquired the claims covering the current property in 1950; however, there is no record of the work performed (if any) by Brookbank-Sturgeon.

Between 1974 and 1975, Lynx Canada Explorations Limited ("Lynx") completed geological mapping, ground magnetic surveys and diamond drilling over a portion of the property. In 1974, Lynx carried out surface mapping and a magnetometer survey on the eastward extension of the Noranda showing. In the following year, Lynx completed six diamond drill ("DD") holes totaling 376 m to test a thin siliceous band along the metavolcanic-metasedimentary contact.

In 1981, Metalore optioned the property from Brookbank-Sturgeon and completed line-cutting followed by an electromagnetic ("EM") survey over the entire grid and a very low frequency electromagnetic ("VLF-EM") survey over selected portions of the property. Metalore subsequently drilled 30 DD holes totaling 3,567 m.

Between late 1982 and early 1983, Metalore drilled three widely spaced DD holes totaling 330 m to test the metavolcanic-metasedimentary contact on the Brookbank West Property and one 453 m DD hole on the Foxear Property.

From September 1983 to March 1984, Metalore completed an additional 62 DD holes totaling 6,946 m, including four wedges. In July 1984, Metalore commissioned a combined helicopter-borne magnetometer, gamma ray spectrometer, and VLF survey over its holdings in Sandra, Irwin and Walters townships, including the Brookbank Project.

From 1984 to 1985, Metalore drilled 23 DD holes, including 14 wedges, on the Brookbank Zone totaling 4,421 m, six DD holes on the Cherbourg Zone totaling 6,684 m, and 26 DD holes on the Foxear Zone totaling 2,202 m.

In 1986, Metalore concentrated on the Cherbourg Zone and completed 43 drill DD holes for a total of 4,368 m. On October 1, 1986, Metalore entered into an exploration and development agreement with Hudson Bay Mining and Smelting Co., Ltd. ("Hudson Bay").

In 1987, Hudson Bay drilled 44 DD holes for a total of 11,203 m on the Brookbank Zone and 10 DD holes for a total of 2,777 m on the Foxear Zone. Mineralogical studies and preliminary metallurgical testing were completed on one mineralized sample and approximately 70 drill collars were located and surveyed.

Metalore's agreement with Hudson Bay was terminated in 1988 because of an ownership dispute between Metalore and Ontex. In October 1998, Ontex acquired a release of Metalore's right to earn an interest in the Brookbank leases, subject to a 1% Net Smelter Royalty ("NSR") due to Metalore upon production.

In July 1989, Placer Dome Inc. ("Placer") and Metalore signed an option agreement to which Ontex was not a party. From early August to late November of that year, Placer completed a program consisting of power stripping/trenching, detailed geological mapping, channel sampling and diamond drilling. Placer exposed an area of about 650 m x 15 m and took 215 channel samples totaling 244 linear meters. Detailed mapping was completed at an Imperial scale of one inch to ten feet. During 1989, drilling at the Brookbank zone consisted of 18 DD holes totaling 7,010 m to test the lateral and down-dip extensions to a vertical depth of 670 m. A Sperry Sun gyro-log system was used to confirm downhole deviations for 13 of the DD holes drilled in 1989 and 15 of the pre-existing holes. Additional Placer drilling at Cherbourg consisted of five DD holes totaling 1,437 m with a further two DD holes totaling 984 m drilled at Foxear. Placer dropped its option due to ongoing litigation between Ontex and Metalore.

From 1990 through to 1996, the Brookbank Project was the subject of Superior Court of Ontario litigation between Ontex and Metalore [Ontex Resources Ltd. v. Metalore Resources Ltd. (1990), 75 O.R. (2d) 513 (Gen. Div.), with an appeal allowed in part (1993) 13 O.R. (3d) 229, 103 D.L.R. (4th) 158, 12 B.L.R. (2d) 226 C.A.)]. Costs were subsequently awarded to Ontex [(1996), 45 C.P.C. (3d) 237 (Ont. Assmt. Officer)].

Between 1993 and 1994, Metalore completed four DD holes totaling 533 m on the Brookbank Zone, fifteen DD holes totaling 2,107 m at Cherbourg and seven DD holes (including one wedge) totaling 3,323 m at Foxear. In 1994, reviews of the data by both Micon and J.R. Trussler & Associates, on behalf of Metalore, were positive and additional work was recommended by both companies. However, the ongoing litigation between Ontex and Metalore precluded work being done.

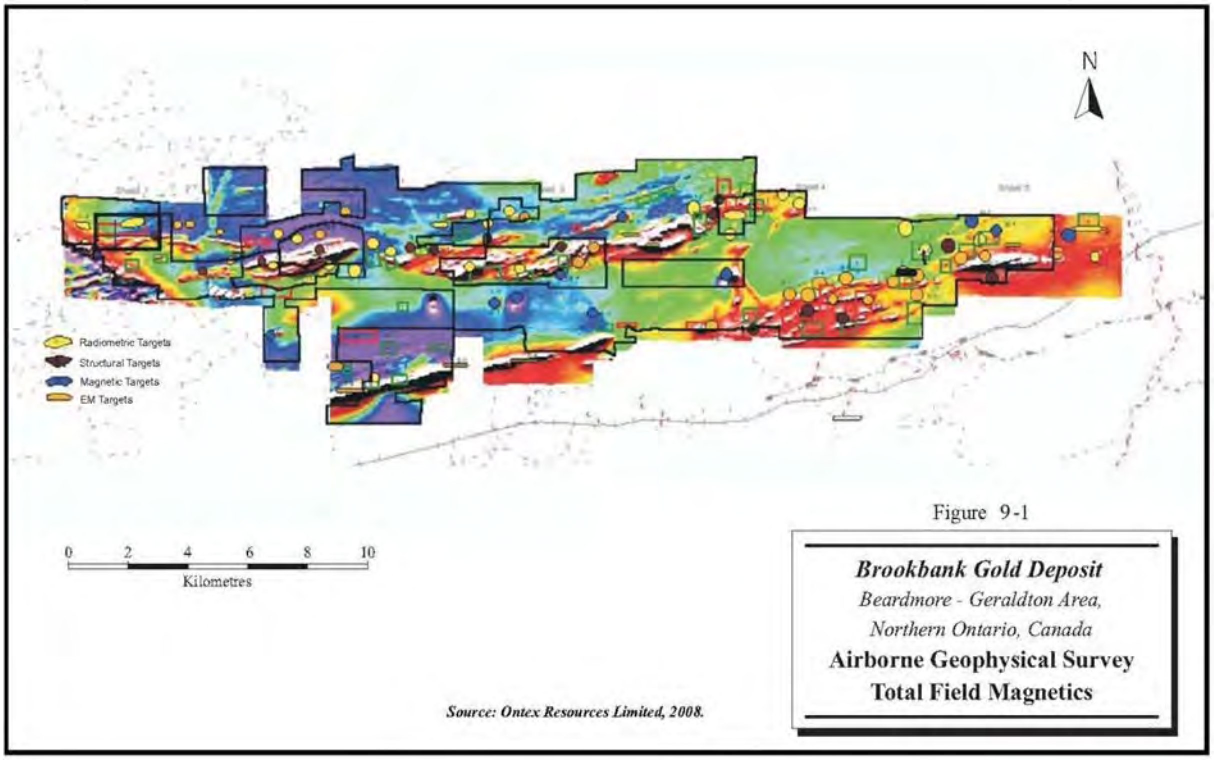
In October 1998, Ontex and Metalore announced a settlement whereby Ontex acquired a release of Metalore's right to earn an interest in the Brookbank leases and Ontex took over as the operator of the Brookbank Deposit and all of the Metalore property in the area.

From 1999 until 2009, all exploration on the property was conducted by Ontex. The most significant of all of Ontex's exploration programs was achieved in September 1999 when Geoterrex-Dighem Ltd. completed a combined helicopter borne magnetic, VLF-EM and radiometric survey along 1,807 line kilometers over the entire property in a north-south direction. The airborne program included the collection and delivery of total field and calculated vertical gradient magnetics, VLF-EM, resistivity and radiometrics K/Th/U ratio. The results are summarized in Figure 6.2.

The airborne survey results are reflective of geology and favorable structure and alteration but are not a direct guide to mineralization. The Brookbank Deposit geophysical signature is very subtle and is too subdued to be a reliable guide to the direct location of further mineralization along the favorable structural break between known gold zones. The geophysical signature, however, can be used to locate alteration on structural breaks that might contain mineralization.

The geophysical targets shown in Figure 6.2 have been used to guide the test-drilling and evaluation programs that have been completed on the Brookbank Deposit to date. Almost all of the completed drilling is in the central part of the claim area. Other targets to the east and west of the Brookbank-Cherbourg-Foxear zone remain to be investigated in greater detail.

Figure 6.2: Major Helicopter Borne Geophysical Targets on the Brookbank Property



Source: Ontex Resources Ltd. 2008

On December 18, 2009, Ontex and Roxmark announced that their respective shareholders had voted in favour of the merger transaction between the two companies. In connection with the merger, Ontex announced that the shareholders approved a one-for-three share consolidation, the election of additional directors and a name change from Ontex to Goldstone.

Table 6.8 summarizes the historic drilling completed at and the Brookbank Deposit prior to Premier's involvement.

Table 6.8: Historic Drilling on the Brookbank Property (1999-2009 Drilling Campaigns)

Company	Drill Hole Type	Years	Number of Holes	Length (m)	Targeted Area
Ontex Resources Ltd.	Surface Diamond Drill Hole (DDH)	1999	32	12,738	Brookbank (17 DDH = 4,995 m)
					Cherbourg (12 DDH = 6,448 m)
					Foxear (3 DDH = 1,295 m)
		2000	52	23,476	Brookbank (34 DDH = 17,120 m)
					Cherbourg (5 DDH = 1,564 m)
					Foxear (13 DDH = 4,792 m)
		2001	21	7,053	Cherbourg (9 DDH = 2,523 m)
					Foxear (12 DDH = 4,530 m)
		2006	14	3,000	Brookbank
		2007	7	1,208	Brookbank
		2008	25	9,461	Brookbank (16 DDH = 5,638 m)
					Cherbourg (9 DDH = 3,823 m)
		2009	50	23,291	Brookbank

In June 2011, Premier and GGM announced that they had entered into a definitive agreement whereby Premier would acquire all the outstanding common shares of Goldstone. Under the terms of the deal, each Goldstone shareholder would receive 0.16 of a Premier common share plus CAD 0.0001 in cash for each Goldstone share held.

On August 16, 2011, Premier completed the previously announced acquisition of Goldstone for approximately CAD 104 M. The acquisition of Goldstone allowed Premier to add the Key Lake, Brookbank, Northern Empire and Leitch-Sand River Projects to its portfolio of projects within the Trans-Canada Property (now called the Greenstone Gold Property) as well as add the remaining portion of the Hardrock Project it did not hold.

On March 9, 2015, Centerra and Premier announced the formation of the Partnership to explore and develop the Greenstone Gold Property. Since its acquisition of the Brookbank Deposit in March 2015, approximately 95% of GGMs exploration expenditures on the Brookbank Deposit have been on diamond drilling, acquisition and claims protection. The details of the drilling are described in Section 10 - Drilling.

6.2.2 Production History

There has not been any historical production from the Brookbank Project area.

6.2.3 Previous Resource Estimates

Scott Wilson RPA completed a previous Mineral Resource estimate on the Brookbank Project in 2009 for Ontex. This estimate is summarized in Table 6.9 and is contained in a NI 43-101 technical report dated May, 2009 and entitled "Technical Report on the Brookbank Gold Deposit, Beardmore-Geraldton Area, northern Ontario, Canada". The Scott Wilson RPA Brookbank Mineral Resource estimate is summarized in Table 6.9.

The 2009 Mineral Resource estimate is compliant with the December 11, 2005 CIM Definition Standards for Mineral Resources and Mineral Reserves as required by NI 43-101 at that time.

Table 6.9: Scott Wilson RPA 2009 Mineral Resource Estimate for the Brookbank Project

Zone	Indicated Mineral Resources			Inferred Mineral Resources		
	Tonnes	Cut Au (g Au/t)	Cut Au (oz)	Tonnes	Cut Au (g Au/t)	Cut Au (oz)
Brookbank	1,217,400	8.8	345,600	813,100	7.4	192,800
Cherbourg	79,900	10.1	25,900	141,200	8.1	37,000
Foxear	34,500	4.3	4,700	54,200	3.7	6,500
Total	**1,331,800**	**8.8**	**376,200**	**1,008,500**	**7.3**	**236,300**

Notes:

1. *A minimum mining width of 1.5 m;*
2. *A minimum grade of 1.0 g Au/t for the Foxear Deposit wireframe;*
3. *A minimum grade of 2.0 g Au/t for the Brookbank and Cherbourg Deposits wireframes;*
4. *Grade capping was at 40 g Au/t for Brookbank, 13 g Au/t for Cherbourg and no capping for the Foxear Deposit; assays were capped prior to compositing;*
5. *A long-term gold price of USD 850/oz and a USD/CAD exchange rate of 1.10 were used.*

A NI 43-101 compliant resource estimate was completed on the Brookbank Deposit by Micon International ("Micon") for GGM in 2012. This is summarized below in Table 6.10.

Table 6.10: Micon 2012 Resource Estimate for the Brookbank Deposit

Area	Category	Tonnes (Mt)	Cut Au (g Au/t)	Gold Oz (000's)
Open Pit	Measured	-	-	-
	Indicated	2.638	2.01	171
	M+I Pit	**2.638**	**0.95**	**171**
	Inferred	0.171	2.38	13
Underground	Measured	-	-	-
	Indicated	1.851	7.21	429
	M+I UG	**1.851**	**7.21**	**429**
	Inferred	0.403	4.07	53

Notes:

1. *CIM Definition Standards of November 27, 2010, were followed for mineral resource estimation.*
2. *Totals may not add correctly due to rounding.*
3. *Mineral resources, which are not mineral reserves, do not have demonstrated economic viability.*
4. *The effective date of the estimate is December 31, 2012.*
5. *Lower gold cut-off used for reporting open-pit mineral resources is 0.5 g Au/t, and 2.8 g/t for underground mineral resources.*

6.3 Key Lake

The following Key Lake section is taken directly from the report titled “Technical Report on the Mineral Resource Estimates for the Hardrock, Brookbank, and Key Lake Projects Trans-Canada Property Beardmore-Geraldton Area Northern Ontario, Canada”, dated January 30, 2013 and prepared by MICON International Ltd.

6.3.1 Exploration History

Drilling by Placer at Key Lake in the 1980s identified extensive zones of gold mineralization but these were initially considered too low grade to be economic (McCormack, L.V 1984). Placer conducted additional drilling in 1990 before abandoning the project. Subsequently, Cyprus confirmed two shallow mineralized shoots with average grades greater than 1 g Au/t (Gasparetto and Stevenson, 1996). Roxmark carried out some drilling in 2010/2011 and identified wide mineralized intervals, such as 1.6 g Au/t (0.047 oz Au/ton) over a drilled length of 30 m in KL-11-109 (including 11.9 g Au/t over 0.3 m). Higher grade intervals, such as 5.6 g Au/t (0.16 oz Au/ton) over 16.1 m in KL-11-112 (including 31.6 g Au/t over 1.85 m) were also encountered. There has been no drilling below a vertical depth of about 250 m.

6.3.2 Production History

The Key Lake Deposit area includes the past-producing Jellicoe Mine. The Jellicoe Mine produced 5,620 oz of gold from 1939 to 1941 and an additional 55 oz in 1949 (Mason and White, 1986). The ore bodies comprised a series of veins, each with a maximum strike length of about 100 m and average width of 0.6 m. The mine workings extend discontinuously for about 1,000 m along strike at depths less than 150 m.

6.3.3 Previous Resource Estimates

A NI 43-101 compliant resource estimate was completed by Micon International for GGM in 2012. This is summarized below in Table 6.11 and is described in more detail in Subsection 14.2.2.

Table 6.11: Micon 2012 Estimate of Key Lake Resources

Area	Indicated Mineral Resources			Inferred Mineral Resources		
	Tonnes (Mt)	Cut Au (g Au/t)	Gold Oz (000's)	Tonnes (Mt)	Cut Au (g Au/t)	Gold Oz (000's)
Open Pit	2.572	1.17	97	1.345	1.29	56
Underground	0.031	6.48	6	0.058	3.57	7
Total	2.603	1.27	103	1.403	1.44	63

Notes:

6. *CIM Definition Standards of November 27, 2010, were followed for mineral resource estimation.*
7. *Totals may not add correctly due to rounding.*
8. *Mineral resources, which are not mineral reserves, do not have demonstrated economic viability.*
9. *The effective date of the estimate is December 31, 2012.*
10. *Lower gold cut-off used for reporting open-pit mineral resources is 0.5 g Au/t, and 2.8 g/t for underground mineral resources*

6.4 Kailey (“Little Long Lac”)

6.4.1 Exploration History

Kailey is located at the former Little Long Lac Mine. In 1917, gold was discovered in the glacial drift along the shore near the Main Narrows on Little Long Lake. In 1932, claims were staked by various individuals. Sudbury Diamond Drilling Co. drilled the area of the gold discovery and outlined a commercial ore shoot. In 1933, Little Longlac Gold Mines Ltd. was formed to develop the mine. A three-compartment shaft was sunk to 137.16 m. In 1934, an electric powerline reached the mine and a 150 t/d mill was built. Between 1935 and 1940; underground development continued in the form of shaft sinking, drifting, winze sinking, cross-cutting, etc. Diamond drilling was extensive. In 1941, the discovery of scheelite in the ore resulted in handpicking of the tungsten rich material. In 1942, the underground development continued. A small mill was built to treat the tungsten. Between 1943 and 1952, the underground development continued, and diamond drilling was extensive. In 1953, the mining operations continued until the end of the year. Salvage of equipment and mill clean-up followed. Between 1954 and 1956, limited production resulted from cleanup. In 1967, a new entity, also called Little Longlac Gold Mines Ltd., drilled 1,524 m to test the iron formation.

6.4.2 Production History

The Kailey Project area includes the past-producing Little Long Lac Mine. The Little Long Lac Mine produced 1,615,713 t at a grade of 11.7 g Au/t for a total of 605,499 oz of gold from 1934 to 1956. This accounts for about 20% of all the gold produced by the 10 mines in the Geraldton gold camp between 1934 and 1970 from approximately 10% of the tons milled.

Past production for the mines on the Geraldton area are presented in Table 6.12

Table 6.12: Historic Mine Production in the Beardmore-Geraldton Area (From Mason and McConnell, 1983)

Past-Producing Mines	Operating Period	Ore Milled		Ore Milled		Oz Au
		Metric Measurements		Imperial Measurements		
		Tonnes	g Au/t	Tons	oz Au/t	
Little Long Lac	1934-54, 1956	1,615,713	11.66	1,780,516	0.34	605,449
Hard Rock	1938-1951	1,323,389	6.33	1,458,375	0.18	269,081
MacLeod-Cockshutt	1938-1968	9,380,425	4.98	10,337,229	0.14	1,475,728
Consolidated Mosher	1962-1966	2,459,761	4.18	2,710,657	0.12	330,265
MacLeod Mosher	1967-1970	847,626	4.01	934,084	0.12	109,324
Bankfield	1937-42, 1944-47	209,627	9.86	231,009	0.29	66,417
Magnet Consolidated	1938-43, 1946-1952	326,599	14.49	359,912	0.42	152,089
Tombill	1938-1942, 1955	172,978	12.43	190,622	0.36	69,120
Jellex	1939-1940	13,359	13.22	14,722	0.39	5,675
Talmora- Long Lac	1942, 1948	6,020	7.32	6,634	0.21	1,417
Total Past Production - All Mines		**15,508,779**	**5.97**	**17,090,676**	**0.17**	**2,975,241**

6.4.3 Previous Resource Estimates

A NI 43-101 compliant resource estimate was completed by Micon for GGM in 2012. This is summarized below in Table 6.13.

Table 6.13: Micon 2012 Kailey Resource Estimate

Area	Category	Tonnes (Mt)	Cut Au (g Au/t)	Gold Oz (000's)
Open Pit	Measured	4.052	1.06	139
	Indicated	4.578	0.86	126
	M+I	**8.630**	**0.95**	**265**
	Inferred	3.688	0.97	115
Underground	Measured	-	-	-
	Indicated	-	-	-
	M+I	**-**	**-**	**-**
	Inferred			

Notes:

1. *CIM Definition Standards of November 27, 2010, were followed for mineral resource estimation.*
2. *Totals may not add correctly due to rounding.*
3. *Mineral resources, which are not mineral reserves, do not have demonstrated economic viability.*
4. *The effective date of the estimate is December 31, 2012.*
5. *Lower gold cut-off used for reporting open-pit mineral resources is 0.5 g Au/t.*

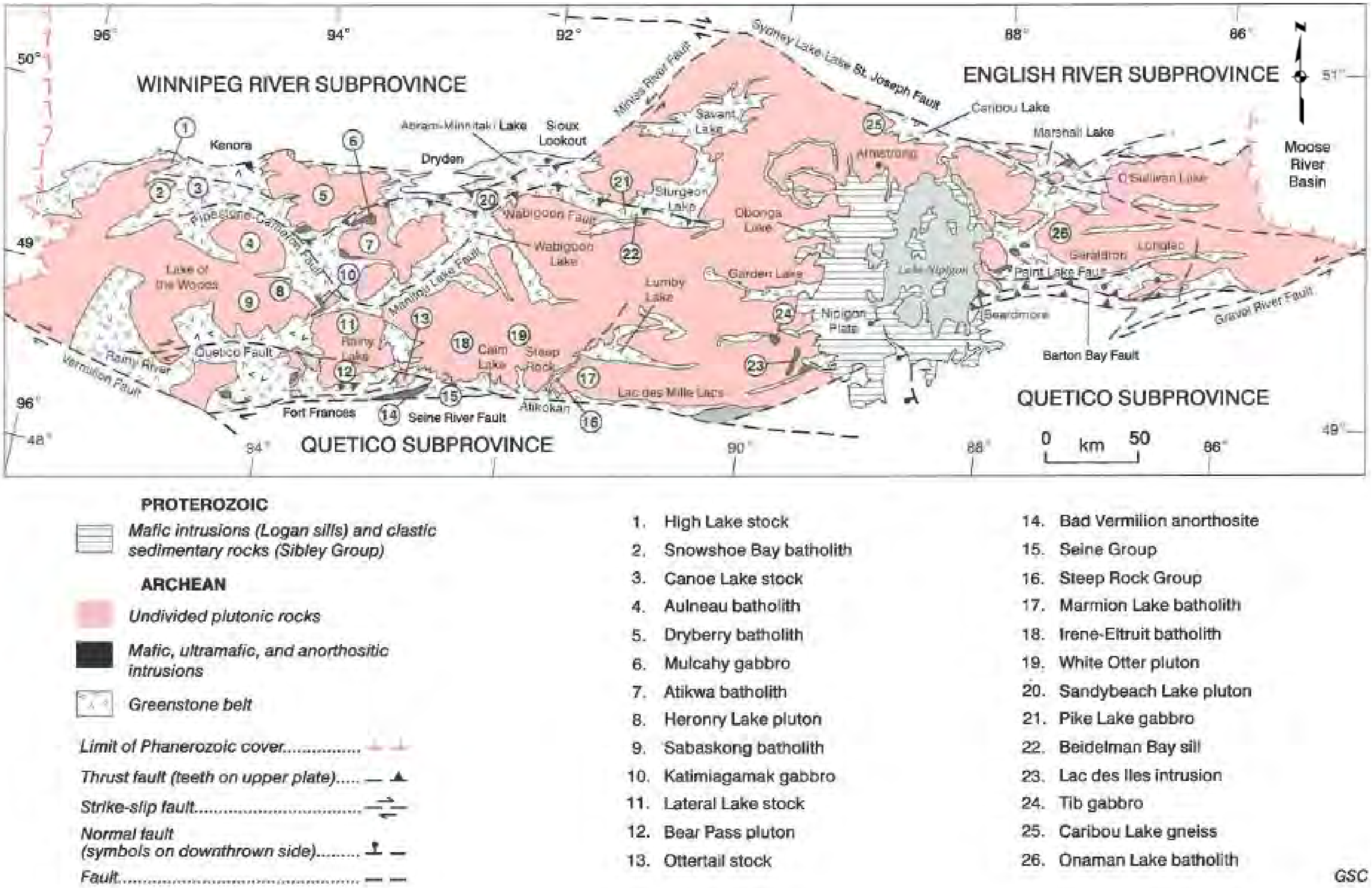
7. GEOLOGICAL HISTORY AND MINERALIZATION

7.1 Hardrock Regional Geological Setting

The Hardrock Project lies within the granite-greenstone Wabigoon Subprovince of the Archean Superior craton, in eastern Canada (Figure 7.1). The Wabigoon Subprovince, averaging 100 km wide, is exposed for some 900 km eastward from Manitoba and Minnesota, beneath the Mesoproterozoic cover of the Nipigon Embayment, to the Phanerozoic cover of the James Bay Lowlands (Card and Poulsen, 1998). The Wabigoon Subprovince is bounded on the south by the metasedimentary Quetico Subprovince, on the northwest by the plutonic Winnipeg River Subprovince, and on the northeast by the metasedimentary English River Subprovince. The Wabigoon-Quetico Subprovince boundary is a structurally complex, largely faulted interface.

The Wabigoon Subprovince can be subdivided into western greenstone-rich domains in the Lake of the Woods-Savant Lake and Rainy Lake Areas, a central dominantly plutonic domain, and an eastern greenstone-rich domain in the Beardmore-Geraldton Area (Blackburn et al., 1991). Deformation and syn- to post-tectonic plutonism occurred between 2711 to 2685 Ma. Based on limited geochronological data, the diverse arc-type volcanic sequences in the eastern Wabigoon Subprovince are thought to be mainly Neoarchean, some as old as 2769 Ma (Anglin et al., 1988).

Figure 7.1: Plan Map of Major Geological Elements – Wabigoon Subprovince (Card and Poulsen, 1998)



A map showing the regional geology of the Beardmore-Geraldton area is shown in Figure 7.2. The following has been taken from the Hardrock Property 2010 NI 43-101 Report completed by Reddick Consulting Inc. (T. Armstrong, M. Srivastava, and J. Reddick, 2010);

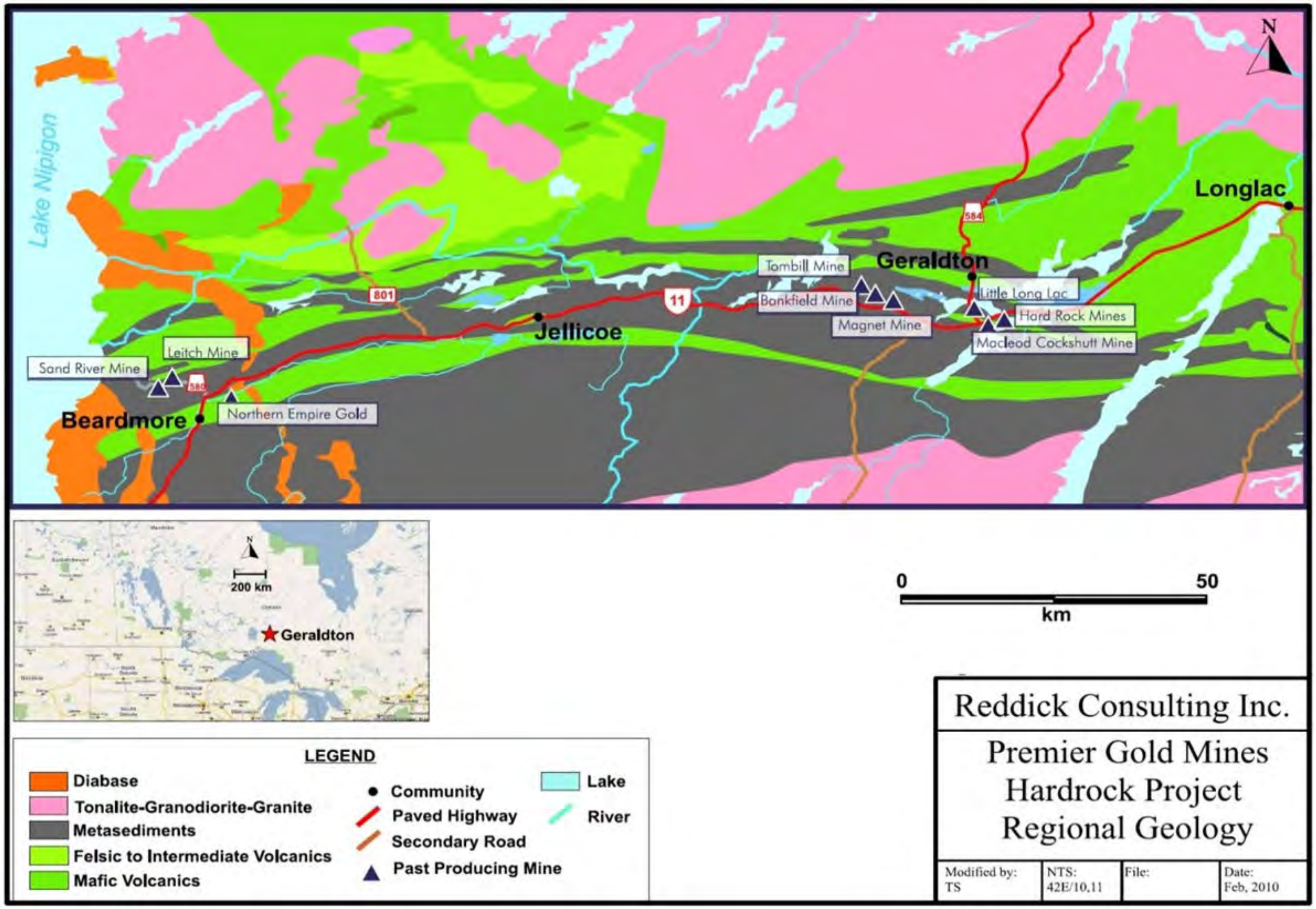
The Hardrock Property is located within the Beardmore-Geraldton Greenstone belt that contains several narrow, east-west striking sequences of volcanic and sedimentary rocks of Archean age. The southern edges of these sequences are spatially related to the through-going, major structural discontinuities thought to be thrust faults that have imbricated the sedimentary sequences. A comprehensive description of the regional geology can be found in Smyk et al., 2005. In the Geraldton area, most of the gold mines and a number of gold showings occur within or in proximity to the Bankfield-Tombill Deformation Zone (also known as the Barton Bay Deformation Zone), a zone of folding and shearing up to 1 km wide. The southern limit of the Tombill-Bankfield Deformation Zone is marked by the Tombill-Bankfield Fault, a zone of intense shearing up to 12 m wide.

In the immediate Geraldton area, the dominant rock types are clastic sediments (greywacke and arenite), oxide facies iron formations (“BIF”) and minor mafic metavolcanics. There are a number of younger intrusives, including an albite-rich porphyry unit (Hard Rock Porphyry) that is spatially associated with much of the gold mineralization on the Hard Rock, MacLeod-Cockshutt and Mosher Mines. Significant gold mineralization is also often spatially associated with BIF. In the case of the Little Long Lac Mine, gold mineralization is primarily hosted by an arkosic unit.

In addition to the belt scale and local faulting, there has been locally intense ductile deformation of the rocks in the Geraldton area which is manifested as tight to almost isoclinal, generally upright, polyharmonic folding of major lithologic units, penetrative deformation, folding and boundinage of veins, lithographic units and local transposition of primary contacts. The degree of deformation is apparent in deformed rocks that are dependent on both primary lithology and proximity to the Bankfield-Tombill Fault.

Gold mineralization in the Hard Rock, MacLeod-Cockshutt, Mosher Mines and the Little Long Lac Mine generally occurs in association with subvertical structures associated with quartz veins or stringers, minor to semi-massive sulphides (associated with replacement zones in BIF), weak to moderate carbonate and weak to strong sericite alteration. The ore zones rake shallowly towards the west in the vicinity of the Hard Rock, MacLeod-Cockshutt and Mosher Mines (15-30° W) and slightly more steeply towards the west at the Little Long Lac Mines (50-60° W), indicative of a strong structural control that post-dates the tight folding of the primary lithological units.

Figure 7.2: Regional Geology Map of the Geraldton-Beadmore Area



The gold mineralization occurs in a variety of host rocks and the style of mineralization is partly a function of the host rock. While the location and overall orientation of the ore bodies appear to have been largely structurally controlled, the deformation of the ore bodies has not been as intense as that of the host rocks. Nevertheless, there are areas where local folding and boundinage of mineralized veins is apparent. Additionally, there are strong secondary controls that influence the extent and intensity of gold mineralization such as the competency contrast between host rocks (e.g. the Hard Rock Porphyry and its contacts with either wacke or BIF) and the chemical character of the host rocks (e.g. oxide facies BIF being replaced by sulphides).

Table 7.1: Summary of Deformation and Gold Mineralization Events – Beardmore-Geraldton Greenstone Belt (Lafrance et al, 2004; Tóth et al. 2013, 2014a, 2014b)

Regional deformation style	Description of structures	
	Folding	Foliation
D_1 thrusting	Gold mineralization	
	Isoclinal, recumbent F_1 folds; up to 1 m in amplitude	Strong; appears in some mafic dikes and QFP; bedding-parallel in sedimentary rocks
D_2 sinistral transpression	Tight upright regional F_{2A} folds; plunge: 20°W-70°W; amplitude up to several km	E-trending, steeply-dipping S_{2A}; axial-planar to F_{2A} folds; parallel or slightly CW/ACW of bedding
	Gold mineralization (or remobilization)	
	Tight to open S-shaped F_{2B} folds; amplitude up to tens of centimetres	E-trending, steeply-dipping S_{2B}; Axial-planar to F_{2B}
D_3 dextral transpression	Gold mineralization (or remobilization)	
	Z-shaped F_{3A} folds; Plunge: 20°W-60°W; amplitude up to several km	ENE-trending, steeply-dipping regional S_{3A}; Axial-planar to F_{3A}; oriented ACW to bedding
	Dextral E-trending shear zones localized along S_2 and lithological contacts	
after Tóth et al., 2015	Z-shaped F_{3A} drag folds overprinting foliation in shear zones	Sinistral slip S_{3B} crenulation clevage; axial-planar to F_{3B}

7.2 Property Geology

A map showing the property geology can be seen in Figure 7.3. The following has been taken from the Hardrock Property 2010 NI 43-101 Report completed by Reddick Consulting Inc. (T. Armstrong, M. Srivastava, and J. Reddick, 2010).

The southern limit of the Property is largely coincident with the Bankfield-Tombill Fault. The fault is described as a variably deformed; largely ductile, high strain zone characterized by strong heterogeneous penetrative strain, narrow shear zones and breccias zones cutting a variety of protoliths. Where it is most highly deformed it is described as a "crush zone" by Smyk et al., (2005) that "has been intensely silicified (Pye, 1952), carbonatized (Anglin and Franklin, 1985) and contains minor amounts of gold (Pye, 1952)."

Horwood and Pye (1951) describe this fault as a “strongly sheared and brecciated zone, which in Ashmore Township attains a width of 40 ft, strikes N. 77° W. and dips at 70° S.”

South of the Bankfield-Tombill Fault the rock are primarily sediments. To the north of the Bankfield-Tombill Fault, the property is dominated by a series of sedimentary units that have an approximate east-west and subvertical orientation. The majority of these units are greywacke/argillite, arenite or oxide facies iron formation. Minor conglomerate units are also found. In the Hardrock area, some of the argillite units also contain 1-5% magnetite, making the distinction between argillite and lean iron formation difficult in places. Individual mm-cm scale bedding is commonly observed in turbidite type sequences within the well bedded units. Massive wacke and arenaceous units are also found. BIF can vary from cm to decimeter scale in thickness, with mm to cm beds common. Although the BIF units are locally tightly folded, attenuated or boundinaged, individual units can in some cases be traced for hundreds to thousands of metres along strike. The greywacke in the vicinity of the Hard Rock and MacLeod-Cockshutt Mines can contain up to 5% mm-cm scale magnetite beds and has been historically referred to as “Lean Iron Formation” in the mine terminology.

Figure 7.3: Hardrock Property Geology

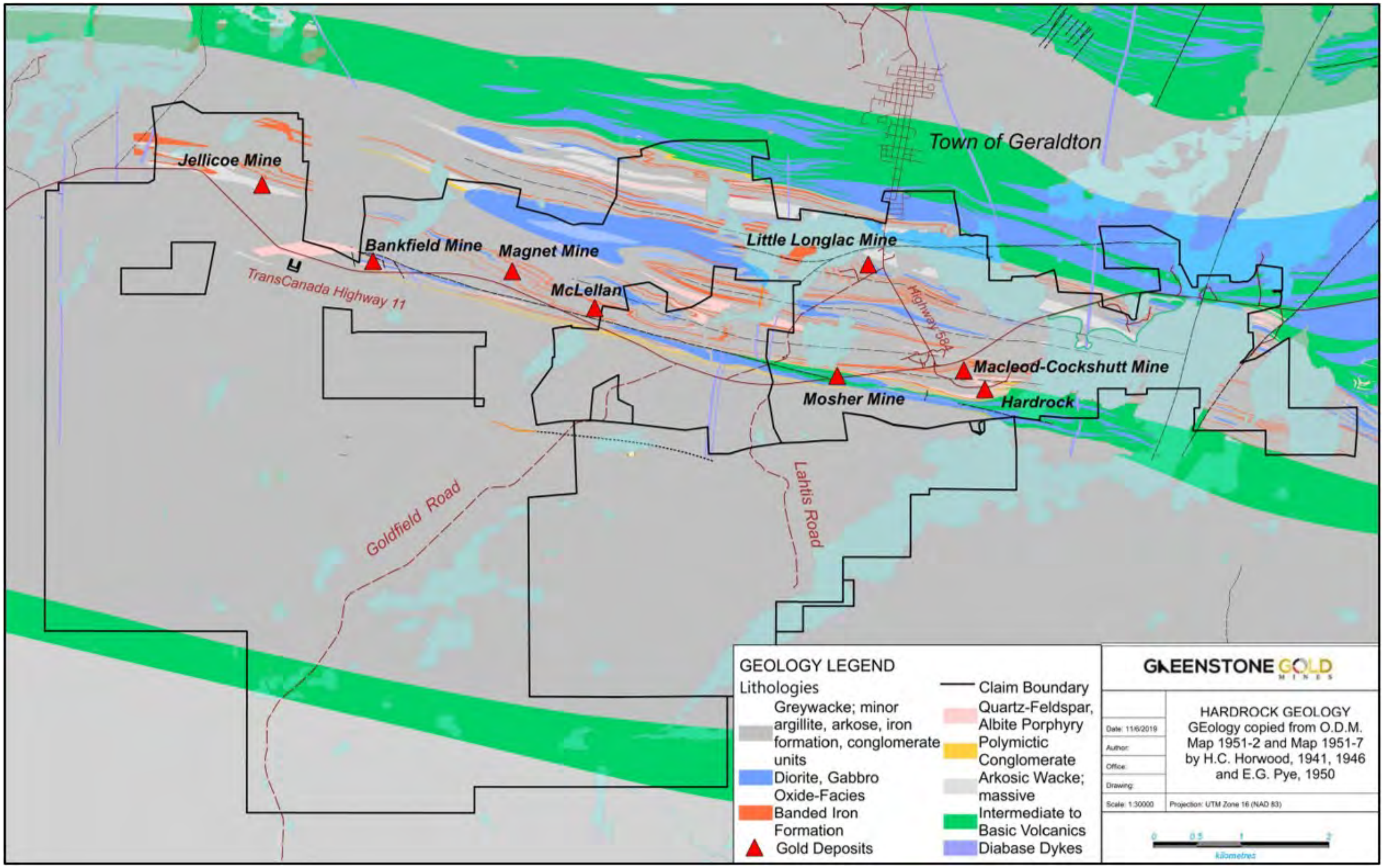


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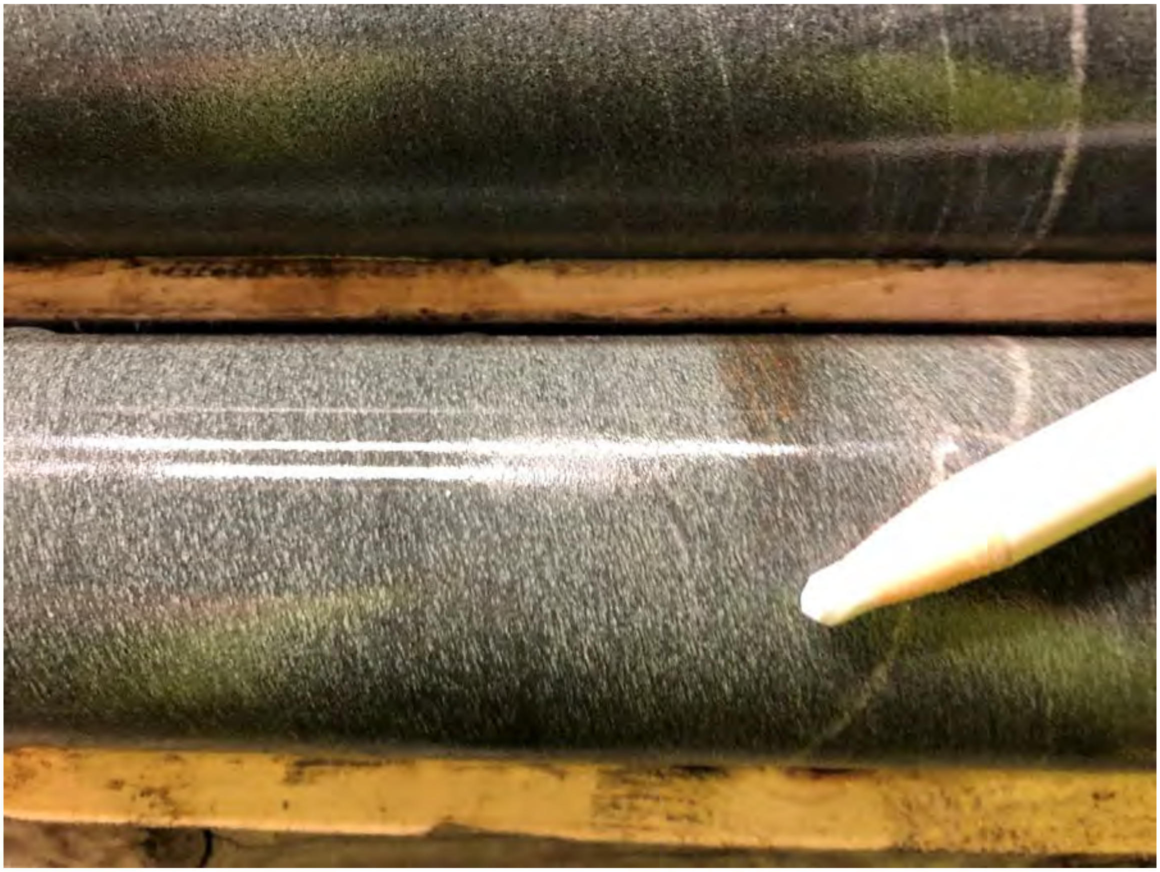
Figure 7.4: Arkosic Unit within Greywackes, Hardrock Deposit Area, DDH 19-21



Intrusive rocks include the Hard Rock porphyry, diorite, gabbro, and diabase dykes. It is of interest that the Hard Rock porphyry seems to be sill-like in nature, even though it is tightly folded and the contacts between it and the sedimentary units are often highly deformed. The general scale and folding pattern of the porphyry very closely matches the geometry of the conglomerate unit that occurs in the vicinity of the Hard Rock and MacLeod Cockshutt Mines.

Figure 7.5: Contact of Hardrock Porphyry and Greywacke. Pen Points North

Note: The folded contact and minor drag folds

Figure 7.6: Contact of Sheared Gabbro (right) with BIF (left) DDH 19-52

7.3 Alteration

The Hardrock Property is underlain by a lithologically heterogeneous package of rocks with anomalous volumes of mafic and felsic intrusions and BIF. Conglomerate occurs along the TBDZ, where most of the gold mines are located. All these rocks are highly strained and have attained lower greenschist facies metamorphism. Despite lithological constraints, it can be demonstrated that chemical alteration near the gold mines often consists of enrichment in Au, Si, K, Ba and CO_2, and depletion in Mg and Ca (Lavigne, 2009). This is manifested as silicification and quartz veining enveloped by sericite-carbonate alteration, accompanied by disseminated pyrite, arsenopyrite and pyrrhotite.

7.7:Quartz Carbonate Veins and Sericite Carbonate Alteration at the F-Zone

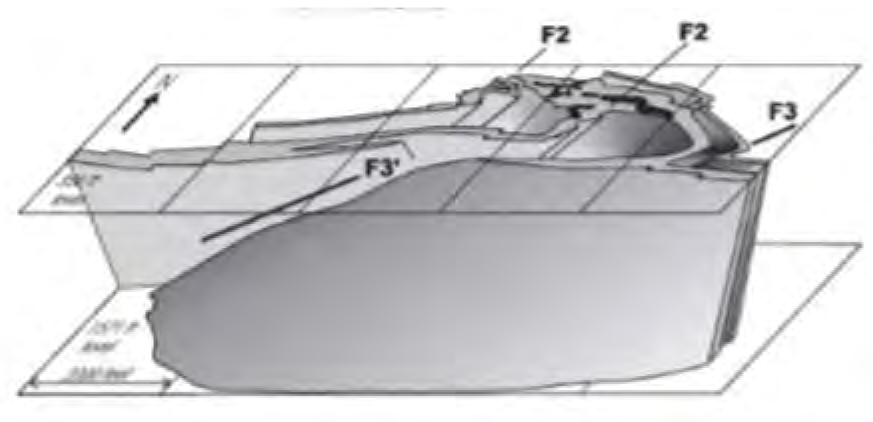
7.4 Mineralization

The following discussion on mineralization was taken from Smyk et al. (2005).

"Gold mineralization in the BGB has resulted from the introduction of hydrothermal fluids in zones of high crustal permeability (Smyk et al., 2005). Permeability was generated by prolonged, multiple periods of deformation, which focused not only fluids, but magmatic activity and intrusions. In the Hardrock Deposit area, a major zone of deformation in which the gold mines are located has been alternatively termed the Bankfield-Tombill Fault Zone (Pye, 1951; Horwood and Pye, 1951) or the Tombill-Bankfield Deformation Zone (Lafrance et al., 2004, and herein).

Most mineralized occurrences in the Hardrock Deposit area lie in a zone of deformation to the immediate north of, and genetically linked to, the Tombill-Bankfield Deformation Zone. This zone of deformation varies from 600 to 100 m in total width (Figure 7.8), while the crush zone of the Tombill-Bankfield Fault proper ranges from metres to hundreds of metres in width. Gold mineralization is associated with D3 brittle shear zones and folds overprinting regional F2 folds (Lafrance et al., 2004). The plunge of the mineralized zones is parallel to F3 fold axes and to the intersection of D3 shear zones with F2 and F3 folds. On a sub-province scale, regional folds cut by D3 dextral shear zones are promising targets for discovering the next generation of large gold deposits."

Figure 7.8: Block Diagram of North Zone at the MacLeod Cockshutt and Hard Rock Mines showing Ore Bods in Black (From Lafrance et al. 2004)



The diagram in Figure 7.9 was drawn using level mine plans published in Horwood and Pye (1951), and shows the overprinting of an F2 S-fold by an F3 Z-fold on the north limb of the Hard Rock Anticline. Ore pods are shown in black. Deformed quartz and quartz-carbonate veins and sulphidized replacement zones occur in BIF host and are spatially related to gold mineralization. Veins are commonly boudinaged and/or folded, while the wallrock is foliated, silicified and sericitized with disseminated pyrite, pyrrhotite and arsenopyrite.

7.4.1 <u>Identification of Gold Mineralization</u>

The interpretation of the mineralized zones by G Mining Services Inc. ("GMS") is based on a litho-structural model developed by InnovExplo but greatly simplifies the domains. As compared to the 2016 Feasibility block model, some wide domains that encompassed significant amounts of internal dilution have been re-interpreted, such that higher grade portions have been made more distinct. In the updated model, lithological domains and mineralized zones are located inside three areas (Figure 7.9 to Figure 7.11).

A North Domain consisting of a refolded (F3 overprinting F2) sequence of BIF and greywacke, with minor porphyry and gabbros. This essentially consists of the large folded "knot" of iron formation and intercalated wackes which plunges to the west at 20 degrees to 35 degrees. Three BIF units are present, denoted by "IF" in the unit names, interlayered with the Mineralized Central Wacke and the undifferentiated greywackes. The North Gabbro is located between the two northernmost BIF units. From top to bottom, the units are as follows:

- North IF 3;
- North Gabbro;
- North IF 2;
- North IF 1.

In the North Domain, mineralization appears to be preferentially spatially associated with the complex refolded area affecting the BIFs and the North Gabbro. Gold mineralization occurs within all rock types but shows a preferential association with the BIFs and gabbro. The three mineralized zones are as follows:

- North 1-Zone;
- North 2-Zone;
- North 3-Zone.

A Central Domain consisting mainly of an undifferentiated greywacke sequence and a mineralized portion of this greywacke, defined as the Mineralized Central Wacke, which are both likely sheared and folded. Three mineralized zones have been defined within the Central Domain to constrain zones of higher-grade gold mineralization inside the Mineralized Central Wacke. From south to north, the three mineralized zones are as follows:

- F-Zone;
- F2-Zone;
- Central-Zone.

A South Domain characterized by a tightly folded (F2) stratigraphic sequence, consisting of the following units from top to bottom:

- Upper Greywacke;
- Mid BIF;
- Upper BIF;
- Porphyry;

- Lower BIF;
- Mid Conglomerate;
- Mid Ultramafic;
- Mid Greywacke;
- Lower Conglomerate;
- Lower Greywacke.

Five mineralized zones have been defined within the South Domain, in which gold mineralization appears primarily associated with the “main” anticline (Hardrock Anticline) and preferentially within both BIFs. These mineralized zones are as follows (from south to north):

- Tenacity Zone;
- SP2-Zone;
- SP-Zone;
- Lower Zone;
- A-Zone.

There are also a number of mineralized intersections within the wackes and near the contacts of the south porphyry. For the most part, this scattered mineralization has not been defined.

The South Gabbro unit marks the southern limit of the deposit and is interpreted to be spatially associated with the Tombill-Bankfield Deformation Zone but shows no evidence of mineralization.

Figure 7.9: Plan View of Litho-Structural Model showing Mineralized Zones at Elevations 300 m and -200 m (Projection: UTM NAD 83 Zone 16)

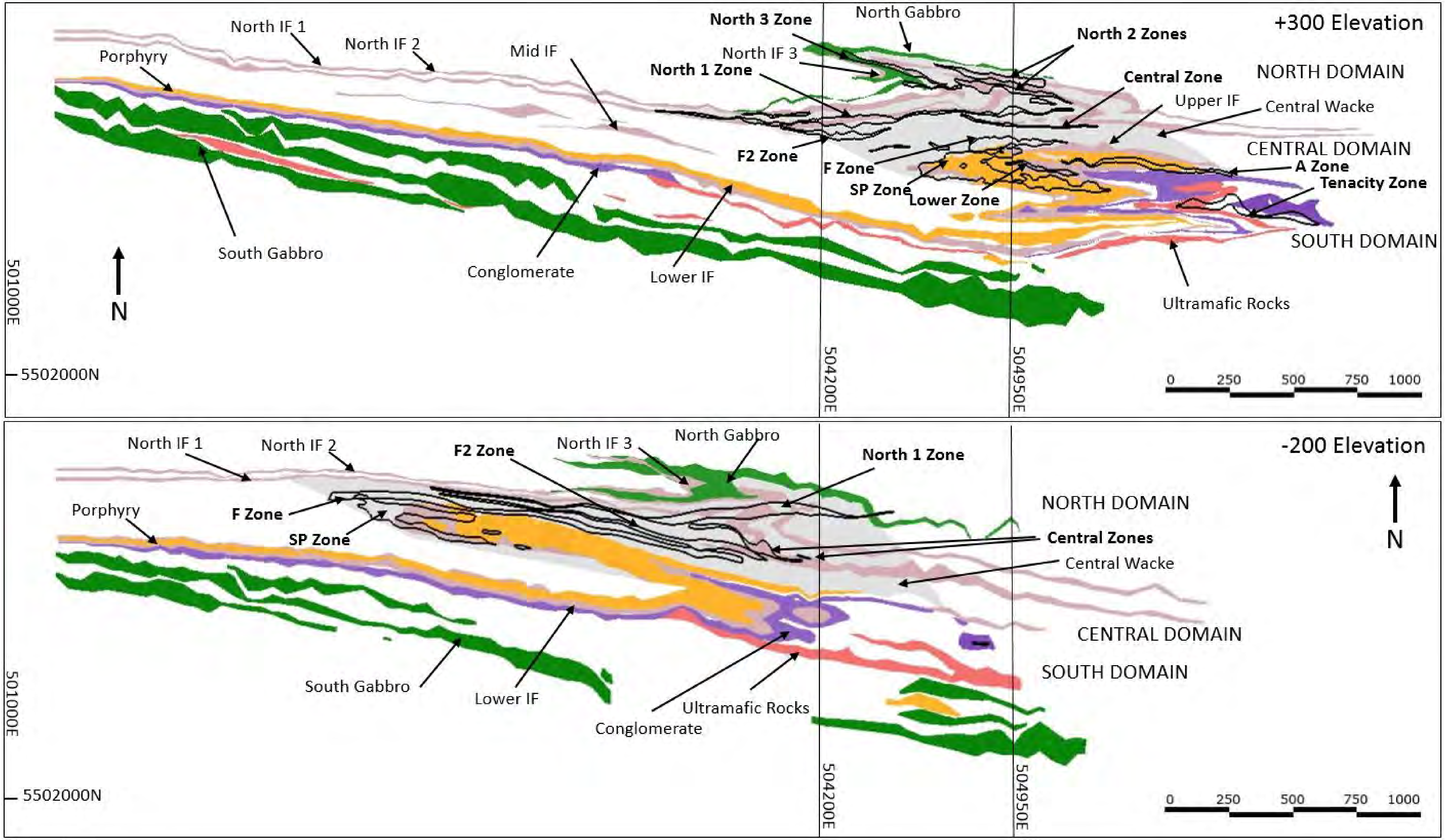


Figure 7.10: Litho-structural Model showing Various Mineralized Zones (Cross section 4200, looking west)

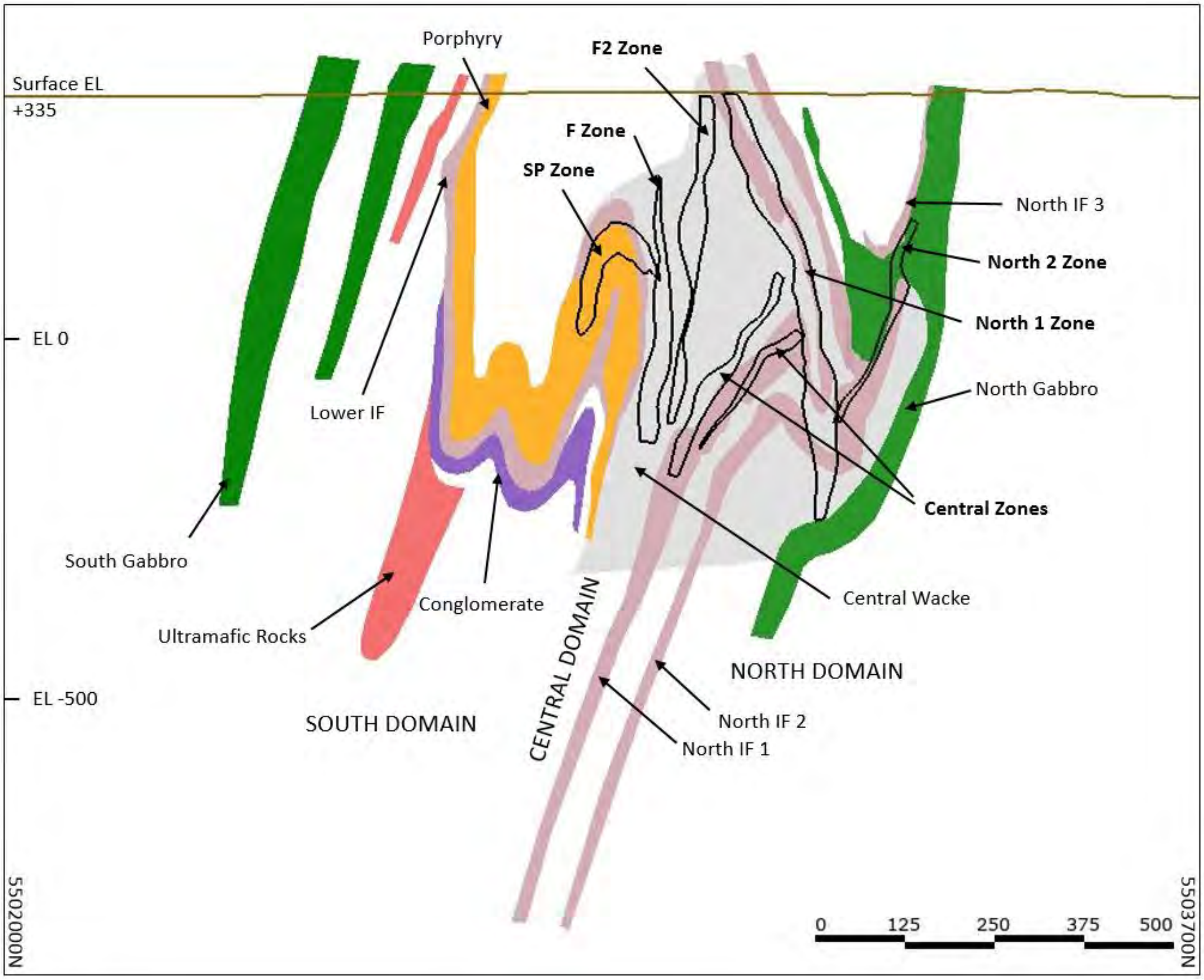
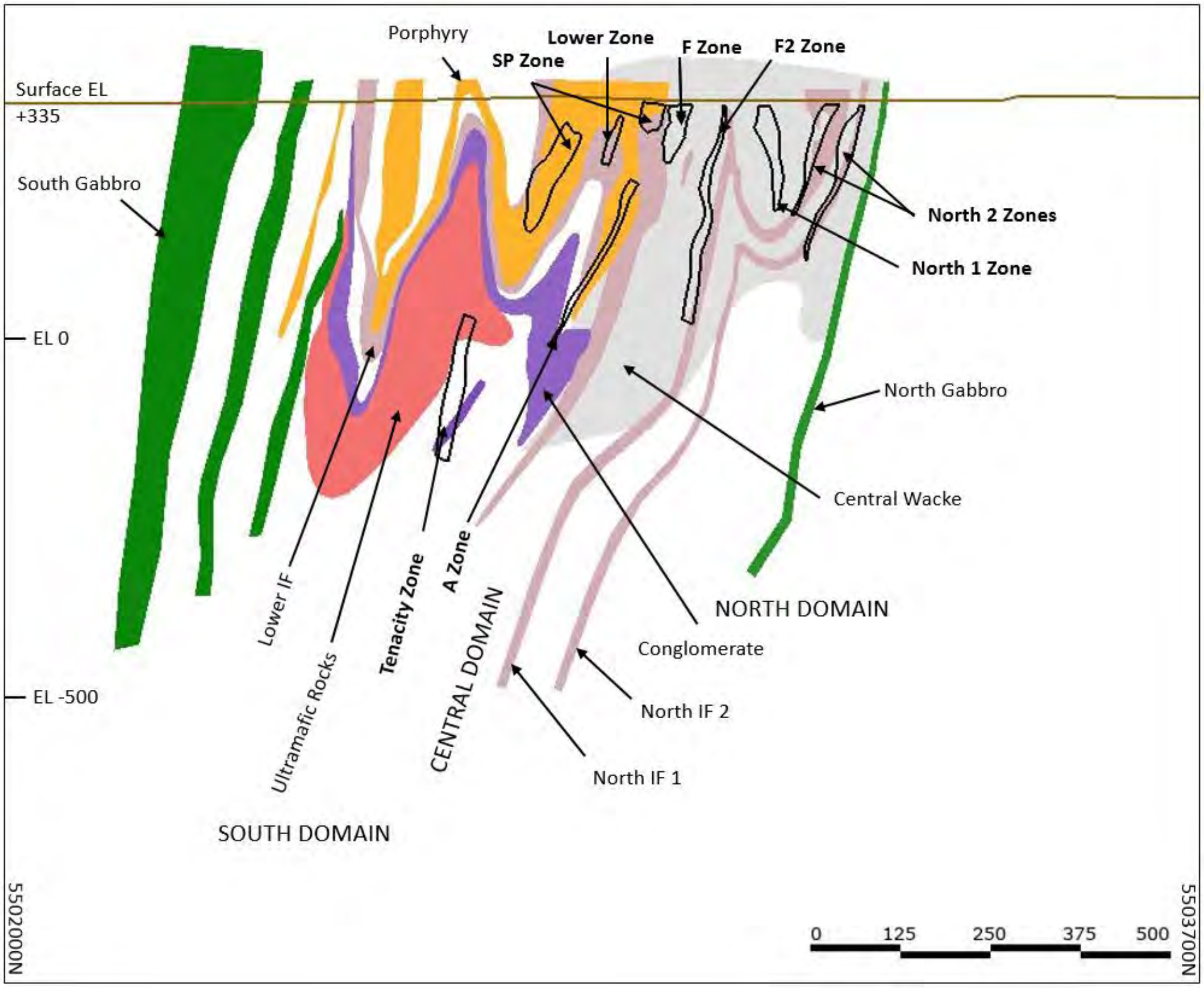


Figure 7.11: Litho-structural Model showing Various Mineralized Zones (Cross section 4950, looking west)



7.4.2 Style of Gold Mineralization

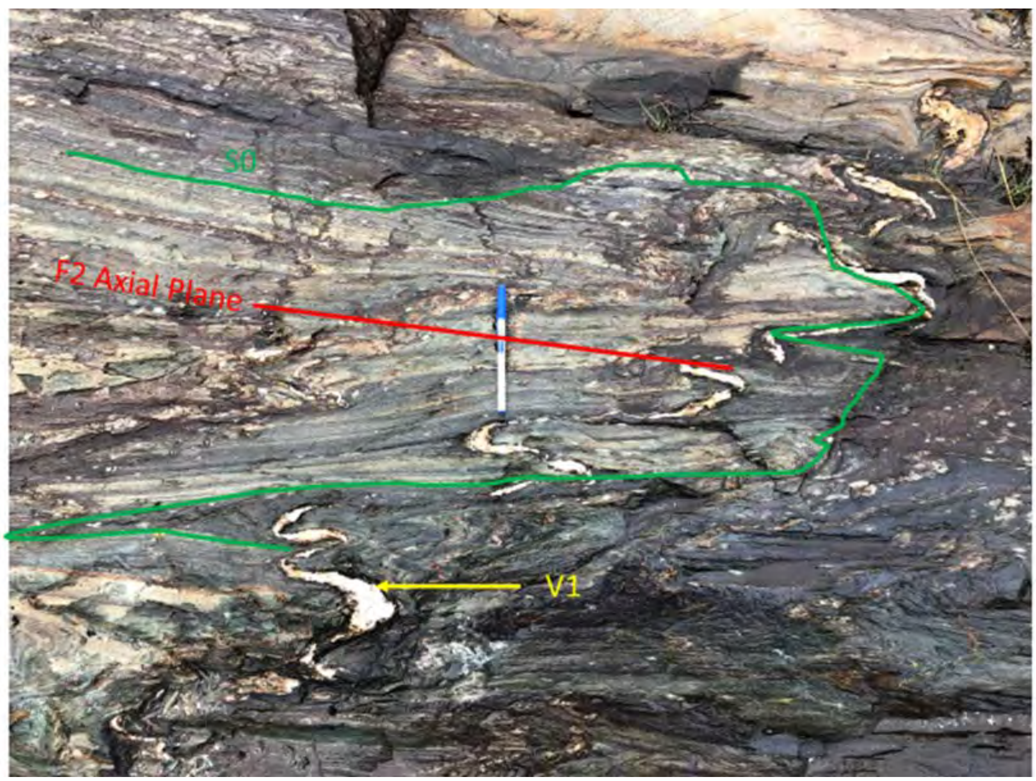
The following discussion on the style of gold mineralization was mostly taken from Davie (1995).

Quartz-carbonate Stringer Mineralization

Zones which are categorized as quartz-carbonate stringer mineralization include F-Zone, F2-Zone, A-Zone, SP-Zone, Central-Zone and Tenacity Zone. Mineralization within these zones generally consists of a series of narrow, tightly asymmetrically folded gold-bearing quartz-carbonate stringers, which are usually attenuated, transposed and dislocated in hook-like segments. The stringers are accompanied by a gold-bearing quartz-sericite-pyrite (±arsenopyrite) alteration halo about the stringers. It is the accumulation of a number of stringers and associated alteration halos that constitutes the zones. Individual stringers and their associated alteration haloes within the mineralized zones are often high grade with minute flecks and clusters of visible gold. Assay results of up to, and often greater than, 30 g Au/t are attainable from some stringers. Overall, zones having average grades of 4 g Au/t as individual stringers are too narrow and discontinuous to consider mining as separate higher grade zones.

The quartz-carbonate stringers and veins display parallel to crosscutting relationships in varying lithologies; however, not unlike the sulfide replacement-type mineralization, they appear to show an affinity towards rocks with higher Fe contents. When in the sediments, the mineralized zones often occur within or proximal to lean iron formations, and variable amounts of pyrite, arsenopyrite and pyrrhotite appear to replace the Fe oxides in the quartz-sericite alteration halos about the stringers. When the mineralization occurs in porphyry, the porphyry displays a similar alteration assemblage with the sulfides having replaced the 0.5 to 1% disseminated hematite content noted in the less altered, hematite-stained porphyry.

Figure 7.12: Deformed Quartz-carbonate Stringers in BIF, Headframe Outcrop



All evidence indicates that the mineralized zones have undergone identical deformation to that displayed by the lithologies and individual veins. As a result, the mineralized zones appear to be the preserved portions of isoclinally and asymmetrically folded mineralized zones occurring at or near the hinge lines of major and minor fold axes. An understanding of this deformation is critical in determining which drill hole extrapolations have the best probability of intersecting mineralization.

Figure 7.13: Folded Quartz-carbonate Veins within Altered Quartz-porphyry, Porphyry Hill

Sulfide Replacement Mineralization

Zones that are categorized as sulfide replacement mineralization include the North 1, North 2 and North 3 zones, and the SP-Zone. The nature of the mineralization within these zones is best understood from the historical work completed on the North 1-Zone. Mineralization within these zones occurs as variable pyrite, arsenopyrite and pyrrhotite replacement of Fe oxide at the margins of quartz veins, within the hinge zones of folded BIFs. The auriferous sulfide replacement appears to have migrated outwards along the iron oxide bands from gold-bearing quartz-carbonate stringers occupying brittle axial planar tension fractures. This replacement mineralization yields grades of 7 g Au/t or greater.

7.4.3 Mineralization by Zone

The following descriptions of mineralization have been copied from the NI 43-101 Technical Report prepared by GMS on December 22, 2016.

Following the initial discovery of gold at the Hard Rock Mine in 1934, and during subsequent exploration and mining over the next 80 years, many different naming systems have been used for the mineralized zones. Table 7.2 summarizes the evolution of the nomenclature.

Table 7.2: Historical and Current Nomenclature of Mineralized Zones

<table>
<tr><th></th><th>2019 Name</th><th>Former Names</th><th>Historical Name</th><th>Description</th></tr>
<tr><td rowspan="4">NORTH DOMAIN</td><td rowspan="2">North 1-Zone</td><td>High Grade North Zone</td><td rowspan="2">North Zone</td><td rowspan="2">Iron formation sulfide replacement</td></tr>
<tr><td rowspan="2">North Zone</td></tr>
<tr><td>North 2-Zone</td><td>n/a</td><td>Iron formation sulfide replacement</td></tr>
<tr><td>North 3-Zone</td><td>North Wall Zone</td><td>n/a</td><td>Iron formation sulfide replacement</td></tr>
<tr><td rowspan="3">CENTRAL DOMAIN</td><td>F-Zone</td><td>F-Zone</td><td>F-Zone</td><td>Quartz-carbonate stringers in greywacke</td></tr>
<tr><td>F2-Zone</td><td>Fortune (F2) Zone</td><td>n/a</td><td>Quartz-carbonate stringers in greywacke</td></tr>
<tr><td>Central-Zone</td><td>n/a</td><td>n/a</td><td>Quartz-carbonate stringers in greywacke</td></tr>
<tr><td rowspan="5">SOUTH DOMAIN</td><td>Tenacity Zone</td><td>Tenacity Zone</td><td>B-Zone</td><td>Quartz-carbonate stringers in greywacke and conglomerate</td></tr>
<tr><td>SP2-Zone</td><td rowspan="2">SP-Zone</td><td>n/a</td><td>Quartz-carbonate stringers in greywacke and minor Iron formation sulfide replacement</td></tr>
<tr><td>SP-Zone</td><td>South Zone / Trench Zone</td><td>Quartz-carbonate stringers in porphyry and greywacke and minor Iron formation sulfide replacement</td></tr>
<tr><td>Lower Zone</td><td>P-Zone</td><td>P-Zone</td><td>Quartz-carbonate stringers</td></tr>
<tr><td>A-Zone</td><td>A-Zone</td><td>A-Zone</td><td>Quartz-carbonate stringers in greywacke and lesser porphyry</td></tr>
</table>

North 1 and 2 Zones

The North 1 and North 2 zones both represent two main types of mineralization, fracture filling and replacement. They are characterized by the presence of massive sulfides, but the fracture filling type contains greater amounts of quartz and carbonate.

The North 1-Zone is an amalgamation of mineralized areas of the historical North Zone located at the Z fold hinge of the main iron formation, and the High Grade North Zone located further west. The North 2-Zone is located along the northern synclinal limb of the historical North Zone and encompasses the majority of its mined resource.

North 3-Zone

Mineralization is primarily quartz-carbonate stringers concentrated at the synclinal hinge contact between the upper iron formation and the northern gabbro and enveloping greywacke. Gold mineralization is focused in areas with intercalated bands (1 to 50 cm wide) composed of all three lithologies, indicating tight isoclinal folding. Mineralization is accompanied by moderate chlorite and sericite alteration in the gabbro and greywacke, and weak to moderate fuchsite alteration in the gabbro. Mineralization is associated with arsenopyrite and pyrite sulfides in all three lithologies.

F-Zone

The F-Zone mineralization lies proximal to the northern contact between the quartz-feldspar porphyry and greywacke. Gold mineralization is associated with trace to 5% pyrite and lesser arsenopyrite and pyrrhotite and moderate to minor sericite, chlorite and carbonate alteration.

F2-Zone

The F2-Zone horizon is composed of multiple, en-echelon, narrow vein zones located between the F-Zone to the south and the North 1 Zone to the north. Gold mineralization is associated with trace to 5% pyrite, with lesser arsenopyrite and pyrrhotite, and moderate to minor sericite, chlorite and carbonate alteration.

Central-Zone

The Central-Zone is a lens within the greywacke envelope adjacent to the North 1-Zone and subparallel to the south limb of the North IF-1 unit. Similar to the F2-Zone, the Central-Zone is characterized by quartz carbonate stringers with trace to 2% pyrite and lesser arsenopyrite, hosted in greywacke with moderate to minor sericite, chlorite and carbonate alteration.

Tenacity Zone

The Tenacity Zone is marked by moderately to intensely silicified and veined greywacke host rocks, adjacent to folded altered ultramafic and conglomerate units. Gold mineralization is associated with traces to 5% pyrite and lesser pyrrhotite and arsenopyrite, accompanied by sericite and chlorite alterations in sediments or talc and serpentine alterations in ultramafics.

SP and SP2-Zones

The mineralization is partly quartz-carbonate stringer and partly sulfide replacement and occurs at the contact between the porphyry and the lean iron formation/greywacke unit of the southern limb of the main porphyry anticline. The mineralization is located along the southern limb, proximal to the hinge of a parasitic asymmetrical Z-fold of the contact. Quartz-carbonate stringer mineralization is predominantly found in the porphyry and greywacke and is associated with trace to 5% pyrite and lesser arsenopyrite. Sulfide replacement mineralization is localized at the contact margins between porphyry and iron formation and consists of 2 to 10% blebby pyrite.

Lower Zone

Mineralization is primarily quartz-carbonate stringers located in the hinge of the Lower BIF with intercalated greywacke. Gold mineralization is associated with trace to 5% pyrite as stringers and blebs, contained in veinlets with 10 to 30% quartz and carbonate. Alteration is strong to moderate chloritization. The mineralized zone is often crosscut by moderately chlorite- and fuchsite-altered gabbro.

A-Zone

The mineralization consists mainly of gold-bearing, irregularly folded, quartz-carbonate stringers that are generally less than 10 cm wide. Most of this gold occurs freely in the quartz-carbonate stringers, although some is associated with pyrite. The mineralization occurs within a folded and fractured greywacke and conglomerate and stops in the northern limb of the porphyry. Gold mineralization is associated with trace to 10% pyrite and lesser arsenopyrite, accompanied by carbonate and sericite alteration.

7.5 Other Greenstone Gold Property Deposits (Brookbank, Key Lake and Kailey)

The following is largely derived from the Technical Report prepared by GMS on December 22, 2016.

7.5.1 Regional Geological Setting

The regional geological setting described in Subsection 7.1 for the Hardrock Project and summarized in Table 7.1 is applicable to the Greenstone Gold Property (formerly the Trans-Canada Property) as a whole, which includes the Brookbank, Key Lake and Kailey Projects.

7.5.2 Brookbank Project Local Geology

The Brookbank Project is underlain predominantly by east-west trending and steeply south to vertically dipping meta-volcanic and metasedimentary rocks. Meta-volcanic rocks consist of massive and pillowed, locally amygdaloidal, flows of basaltic composition along with related tuffaceous rocks. Pillowed flows exhibit tops to the north. They are locally intercalated with coarser-grained rocks of similar composition that have been interpreted as either intrusions or coarse-grained volcanic phases at the center of thicker basaltic flows. The meta-volcanic rocks are locally intruded by quartz-feldspar porphyritic dykes. See Figure 7.14 for a geological map of the prospect.

Mafic meta-volcanic rocks are fault-bounded against domains of metasedimentary rocks. The northern domain consists of a polymictic conglomerate with pebble- to boulder-sized, rounded to sub-rounded clasts in a feldspar-quartz-sericite matrix. Clasts consist of volcanic and intrusive rock types of various compositions, quartz pebbles and jasper, the latter suggesting affinity with Timiskaming Formation conglomerates in the Timmins (Porcupine) Mining District.

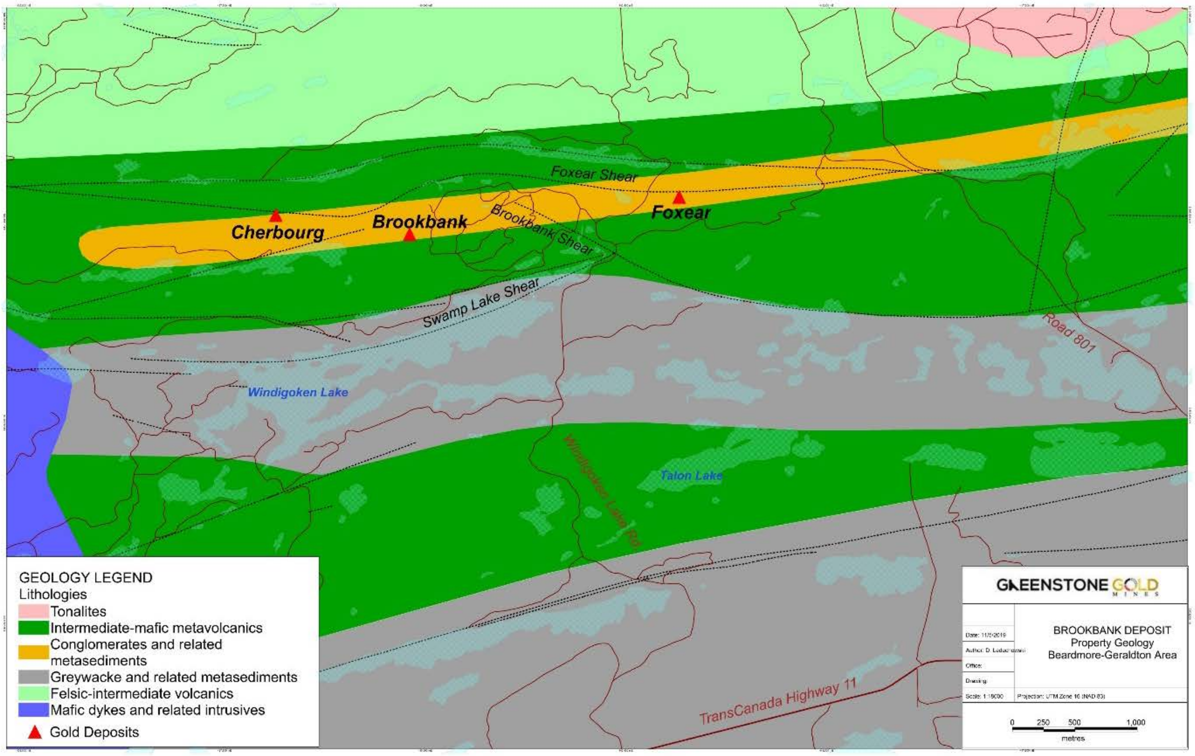
Metasedimentary domains south of Windigokan Lake also contain polymictic conglomerate as well as feldspathic and quartzose sandstone and wacke, siltstone, minor argillite and hematitic iron formation.

Felsic to intermediate pyroclastic rocks and flows occur in the north part of the property and are fault bounded with mafic meta-volcanic rocks across the Paint Lake Fault. They consist of tuff breccia, pyroclastic breccia and tuff, and massive to porphyritic rhyolite flows.

Intermediate to mafic intrusions cut the meta-volcanic and metasedimentary rocks in the central part of the Brookbank Property. They consist of quartz diorite, diorite and gabbro. North-trending, flat-lying, locally porphyritic diabase dykes of Keweenawan age cut the meta-volcanic and metasedimentary rocks along the western boundary of the property in Sandra Township and along the western boundary of Irwin Township.

The Brookbank Project is transected by an east-west trending zone of extensive heterogeneous brittle and ductile deformation and hydrothermal alteration, which is referred to as the “Brookbank shear zone” (Figure 7.14). Deformation is locally in excess of one kilometre wide and consists of anastomosing bands of intense fissile shearing, quartz veining and fracturing with associated ductile deformation around domains of less deformed metavolcanic and metasedimentary rocks. The deformation can be traced for a minimum of 10 km along strike through Irwin Township and remains open in either direction.

Figure 7.14: Brookbank Project Geology



7.5.3 Brookbank Project Mineralization

The 6.5 km long Brookbank shear zone hosts the Brookbank, Cherbourg, and Foxear Deposits (Figure 7.14). The deposits occur along lithological contacts between mafic volcanics and metasediments.

Other areas of gold mineralization are present in one or more of the localized deformation bands within the hanging wall mafic volcanics, which are generally parallel to the Brookbank main zone within the Brookbank shear zone structure.

The zones of mineralization at Brookbank, Cherbourg and Foxear occur within one of several bands of intense deformation and hydrothermal alteration at or near the contact between domains of mafic flows and polymictic conglomerates. Hydrothermal alteration accompanying the mineralization consists of silicification, carbonatization, sericitization, chloritization, hematization and sulfidization (Figure 7.15). This alteration is commonly marginal to the mineralized quartz-carbonate veins, fractures and stockworks and may exceed 50 m in width locally.

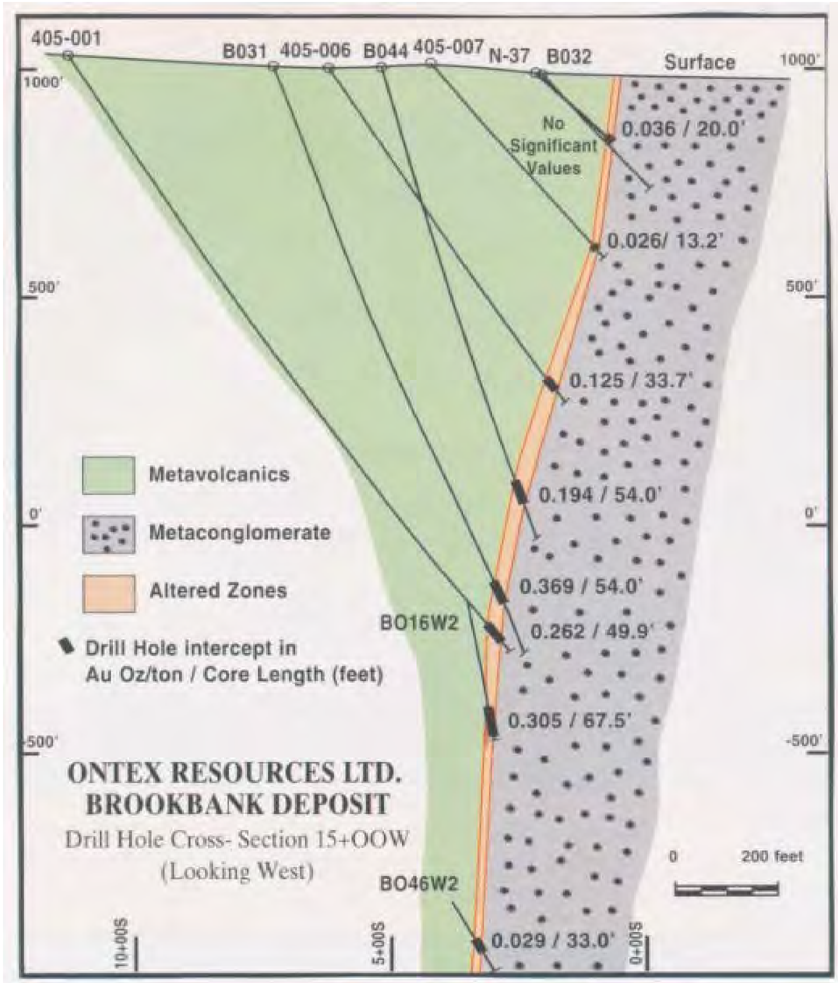
Figure 7.15: Exposure of the Brookbank Mineralized Corridor showing Intense Hydrothermal Alteration

Micon, 2013

Mineralogical studies indicate that the precious metal mineralization consists of gold-silver particles with an approximate gold to silver ratio of 80:20. The gold occurs primarily as late fracture-controlled mineralization. The mineralization forms elongate lenticular particles associated with grain boundaries and possibly crystallographic planes. The gold generally consists of fine grained free gold particles, although there is very little visible gold even in areas of plus 30 g Au/t assays. Gold values are highest in the quartz–carbonate veinlets/stringers.

Sulfide mineralization (pyrite and minor chalcopyrite) is also present within the sheared host rock and quartz veinlets.

Figure 7.16: Cross Section 1500W



After Ontex, 2009.

7.5.4 Key Lake Project Local Geology

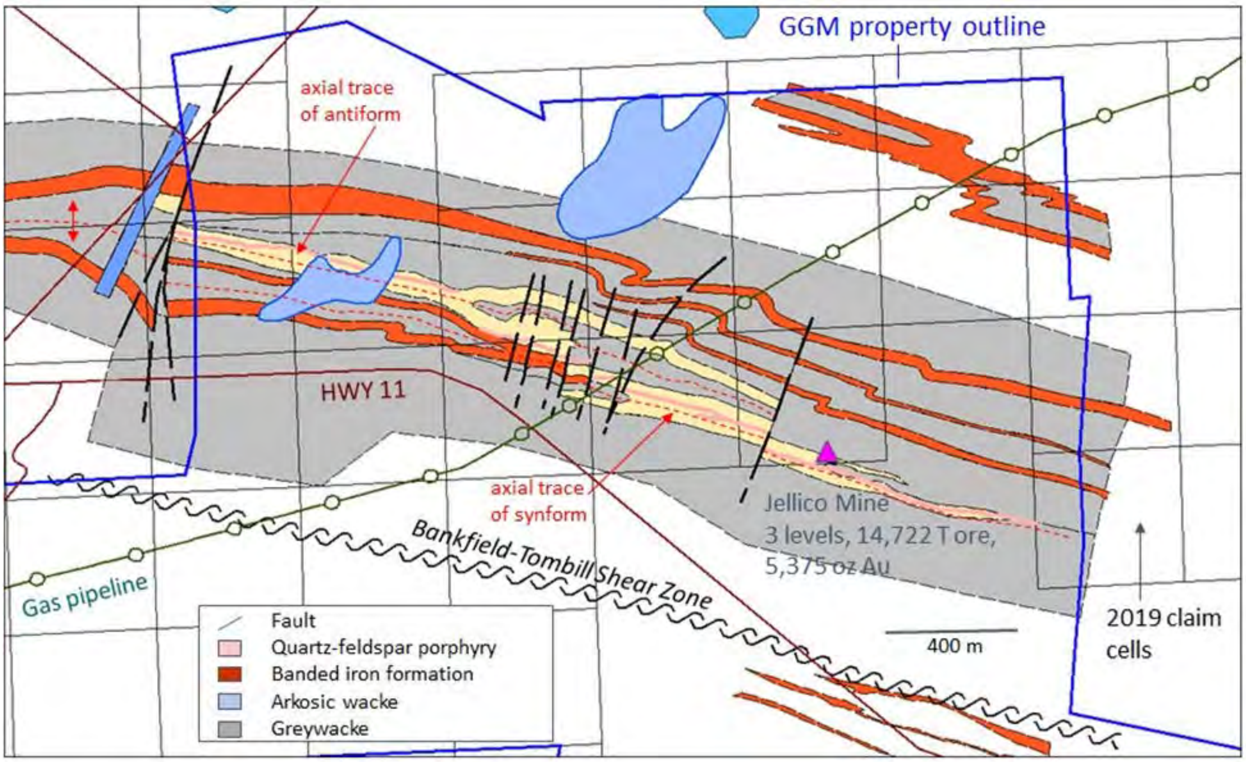
The Key Lake Project is located within the Beardmore-Geraldton Greenstone Belt of the Wabigoon Subprovince of the Superior Province. The project area is within the southern metasedimentary sub-belt on the southern limb of a west-plunging syncline. The mineralized zone at Key Lake is 550 to 800 m northeast of the Tombill-Bankfield Fault and diverges from it toward the west. It is about 2.5 km south of the contact with the central meta-volcanic sub-belt.

Meta-greywacke is the predominant rock type in the area and occurs in a series of turbidites. A thick section of fine to coarse-grained altered wacke hosts most of the gold mineralization. A bed with granule- to pebble size clasts may be a matrix-supported meta-conglomerate or a vitric lapilli tuff. Magnetite-rich argillite occurs to the north and south of the mineralized zone. Banded Iron Formations (“BIF”) occur further north.

The metasedimentary rocks have been intruded by one or more thin (0.5 to 3 m) porphyritic aphanitic felsic dykes which are spatially related to gold mineralization and is a useful marker horizon. Gabbro and diorite dykes occur in some areas and Proterozoic diabase dykes crosscut all other rock units.

Figure 7.17 illustrates the interpreted geology taken from Dome Exploration maps and is based on magnetics survey data and drill cores from ~18,300 m of drilling (Burk, 2019).

Figure 7.17: Generalized Geology of the Key Lake Property



7.5.5 Key Lake Project Mineralization

Gold occurs in altered meta-greywacke (arkose), felsic dykes and in thin veins cutting these rocks. Gold bearing altered rocks typically have more than trace amounts of pyrite and/or arsenopyrite. Mason and White (1986) reported sphalerite and silver. Accessory chalcopyrite has been identified in some holes. A variety of veins are present including quartz with angular bits of white carbonate typically along vein margins, white and grey massive quartz, and dark grey veinlets usually less than 3 mm thick composed of quartz and/or very fine grained arsenopyrite. Visible gold occurs in veins in both meta-greywacke and felsic dykes but is not common and rarely occurs in wall rock.

Alteration occurs within and extends beyond the zone of gold mineralization. Widespread dolomite/ankerite alteration was detected by staining (Gasparetto and Stevenson, 1996).

Zones of greenish, brownish, and rarely yellowish sericitization are more limited and envelope all but a small fraction of the gold mineralization. Silicification is more limited still and is a very good indicator of gold mineralization. However, a significant proportion of the gold mineralization does not occur in silicified rocks.

Two examples of mineralization observed on the Key Lake Deposit are shown in Figure 7.18 and Figure 7.19.

Figure 7.18: Example of Fracture-controlled Pyrite Mineralization in Sericite-silica Arkosic Wacke, 0.54 ppm Au

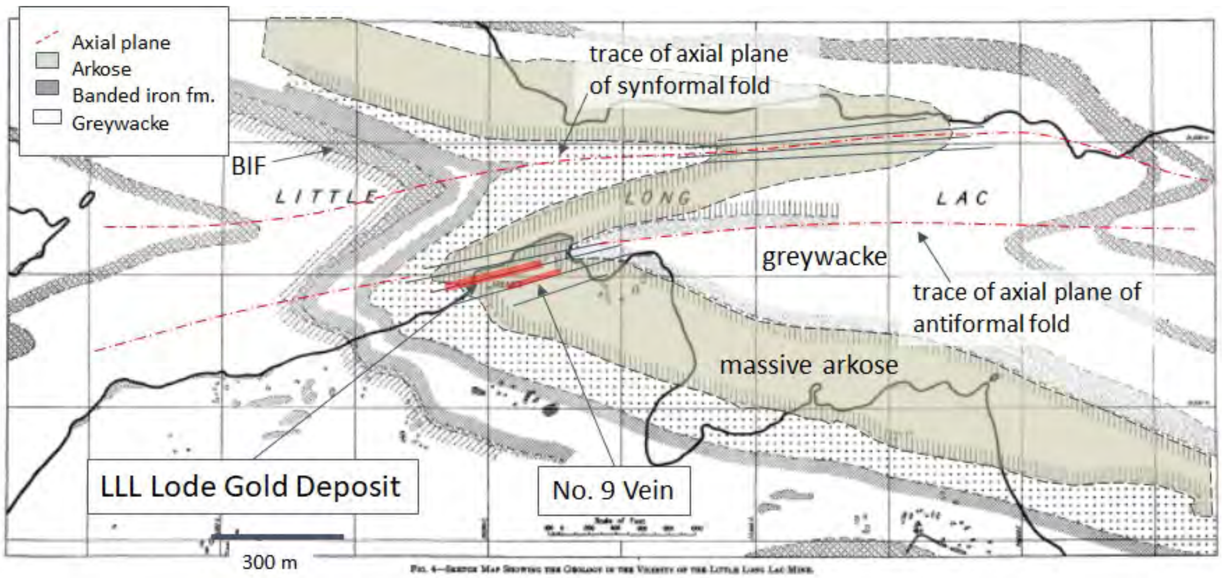
Figure 7.19: Example of Sericite-altered Quartz-feldspar Porphyry; 7.75 ppm Au. Dyke Intrudes Arkose (top core)

7.5.6 Kailey Project Local Geology

The local geological setting described in Subsection 7.3 for the Geraldton area and summarized in Figure 7.1 is applicable to the Kailey Project.

The Kailey Deposit is located at the former Little Long Lac Gold Mine, about 1.7 km north of the Hardrock Mineralized Corridor. It lies within a broad synclinal belt of greywacke, slates, conglomerates and iron formation that extend westwards to Lake Nipigon. The sediments overlie a thick series of lavas, and both are intruded by igneous rocks of various ages and types. At Little Long Lac Gold Mine, the sediments follow a westerly pitching drag fold on the northern limb of the syncline. Subsequent to the folding, east-west zones of shearing developed and formed channel ways for gold-bearing solutions.

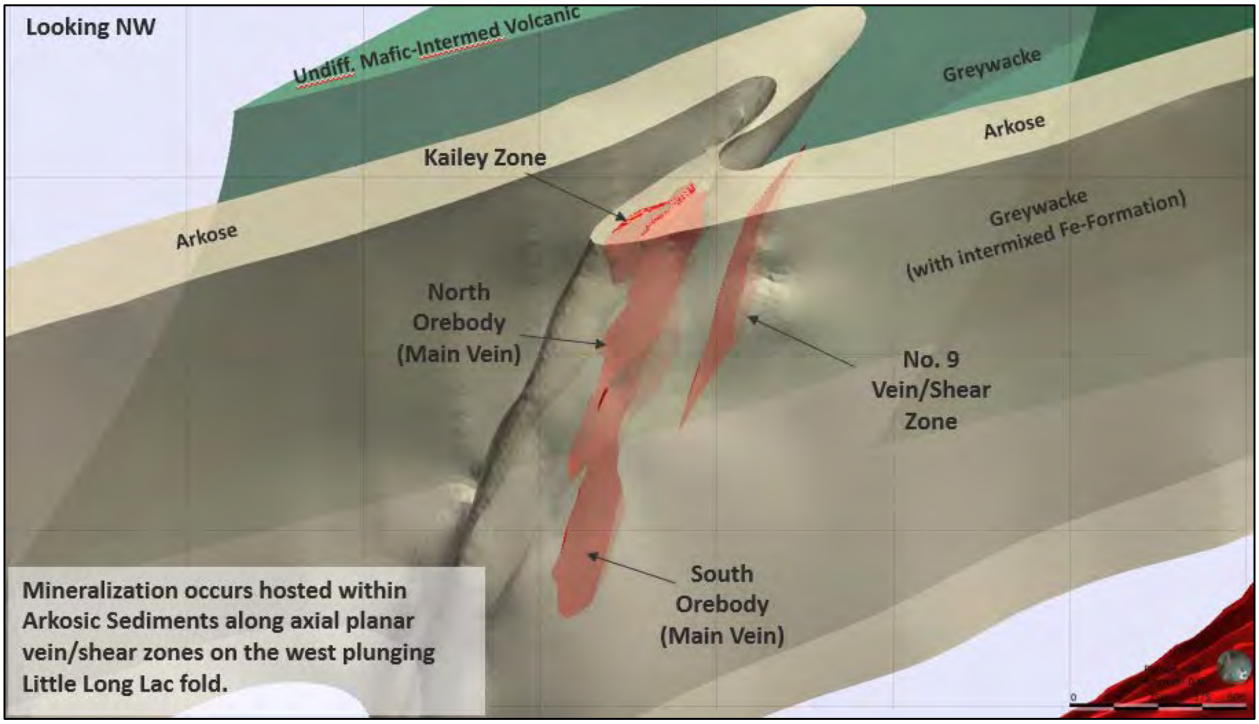
Figure 7.20: Kailey Project Geology



Deposit-forming quartz veining is localized along a N70E-striking, subvertical axial planar shear structure within a west-plunging antiformal fold that is part of a Z-shaped drag fold developed on the northern limb of the west-plunging Barton Bay Syncline. Closure of this antiformal fold is obscured by the waters of Barton Bay, Lake Kenogamisis (Little Long Lac). The No.9 Vein is also controlled by an axial planar fracture that parallels the main lode. The uniform, massive nature of the arkose unit favoured the development of the through-going fractures (Burk, 2019).

An isometric view of the various mineralisation and host unit is shown in Figure 7.21.

Figure 7.21: Isometric view of the Kailey, North, South and No.9 Vein Mineralization with the Folded Arkosic Host Unit



7.5.7 Kailey Project Mineralization

The Main Zone mineralisation typically consists of two parallel quartz veins, 2-20 cm thick with sheared arkose at their margins and separated by 100-150 cm of fractured arkose containing multiple quartz stringers; the larger veins pinch and swell but were remarkably continuous along strike and down plunge.

The Kailey zone is a shallow and low-grade mineralized domain which is north to historical Main Vein. The south limb of the zone appears to represent the upper extension of the North orebody (Main Vein). Mineralization is characterized by a network of narrow quartz-carbonate veins and stringers hosted in an altered arkose (sericite + lesser carbonate) which contains a lower-grade halo.

The No.9 zone is located approximately 150 m south of the Main orebody. It is commonly about 0.5 to 2 m wide and contains a relatively high-grade shear zone, within a lower grade halo (up to 30 m wide). The lower grade halo is characterized by strongly altered and moderately sheared arkose.

Mineralization is accompanied with predominantly pyrite, arsenopyrite and minor base metal sulphides. Scheelite is also present in varying degrees.

8. DEPOSIT TYPES

8.1 Hydrothermal IF-Hosted Gold Deposits

The gold orebodies at the MacLeod-Cockshutt Mine are one of the type examples of BIF-hosted gold deposits. Other well-known Canadian examples include the Central Patricia Mine, ON and Lupin Mine, NT. In these types of deposits, ductility contrasts between the iron formation units and enclosing sedimentary units create structural traps that encourage the flow of hydrothermal fluids. The iron formation also acts as a chemical trap, precipitating Sulphur, arsenic and attendant precious metals from the hydrothermal fluids. In Archean terranes, this usually occurs in a brittle-ductile structural regime, depositing mesothermal mineralization. Quartz-carbonate veins and sulphide replacement zones are common. It should be noted, however, that the bulk of the "iron formation-hosted" mineralization at Hardrock actually occurs within the interbedded wacke and argillite. This suggests that during deformation, the iron formation deformed ductilely, while the wacke units were more brittle. Alteration includes the addition of silica, K, CO2, S, +/- As and the depletion of Ca and Mg. This is manifested in the rock as quartz-carbonate veining, silicification and/or semi-massive to massive sulphides (pyrite, pyrrhotite +/- arsenopyrite) surrounded by a halo of sericite-carbonate-pyrite alteration.

The following sub-types are recognized and are present in the Hardrock Project. The following descriptions are copied from the 2016 FS report, which in turn is quoted from Kerswill (1993).

8.1.1 Non-Stratiform Type

In non-stratiform deposits, gold is restricted to late structures (quartz veins and/or shear zones) and/or sheared sulfide BIF immediately adjacent to such structures. Mineralization is confined to discrete, commonly small, shoots separated by barren (gold- and sulfide-poor), typically oxide BIF. Mineralized rocks are generally less deformed than associated rocks. Iron-sulfide minerals are in many cases relatively undeformed and unmetamorphosed. Pyrite and/or sheared pyrrhotite have clearly replaced other pre-existing iron-rich minerals, notably magnetite. Arsenic-bearing minerals are common, but not always present. If they are present, a strong positive correlation generally exists between gold and arsenic. Alteration is usually typical of that associated with "mesothermal vein" gold deposits. Mineralization is relatively silver-poor, and gold grains generally have gold/silver ratios of >8.0. Non-stratiform deposits are relatively common, typically small and, compared with stratiform deposits, difficult to evaluate and mine.

Non-stratiform deposits contain sulfide-rich alteration zones immediately adjacent to late structures and are similar to mesothermal vein-type gold deposits. Late quartz veins and/or shear zones are present in most known BIF-hosted gold deposits. The distributions of gold-bearing veins and sulfide-rich zones are commonly controlled by fold structures. Major faults ("breaks") of regional scale have been recognized near many non-stratiform deposits.

Irregular, massive lenses of sulfides and quartz occur in a folded series of greywacke and iron formation in the Hard Rock and MacLeod-Cockshutt mines (Horwood and Pye, 1951). These massive replacement lenses (up to 65%, sulfides) cut the Z-folded iron formation and are related to quartz-carbonate veins up to 0.6 m wide. Veins are usually barren of gold mineralization except where they contain sulfides, consisting primarily of pyrite, arsenopyrite and pyrrhotite. Mineralization is preferentially concentrated in the wall rocks outward from the quartz veins and ore is locally banded due to the selective replacement of the less competent wacke laminae in the iron formation by sulfides.

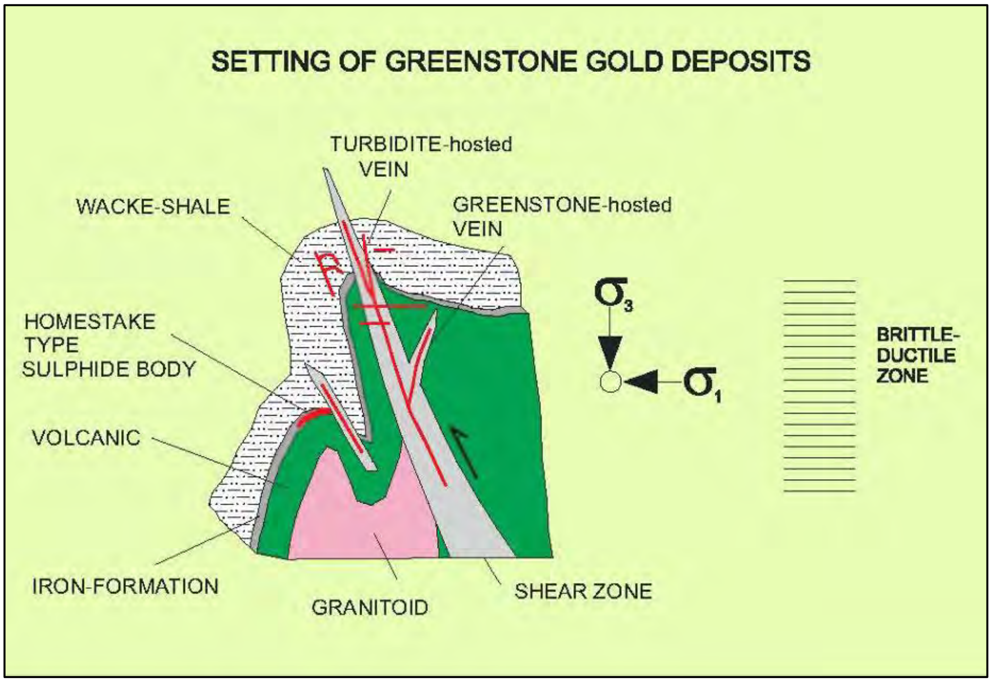
Examples of this type of deposit at the Hardrock Project include the North Zones, as well as parts of the F, F2 and Central Zones.

8.1.2 Greenstone-Hosted Quartz-Carbonate Vein Deposits

Greenstone-hosted quartz-carbonate vein deposits occur as quartz and quartz-carbonate veins, with valuable amounts of gold and silver in faults and shear zones located within deformed terrains of ancient to recent greenstone belts commonly metamorphosed at greenschist facies (Dubé and Gosselin, 2007). Greenstone-hosted quartz-carbonate vein deposits are a subtype of lode gold deposits (Poulsen et al., 2000) (Figure 8.1). They are also known as mesothermal, orogenic. They consist of simple to complex networks of gold-bearing, laminated quartz-carbonate fault-fill veins in moderately to steeply dipping, compressional brittle-ductile shear zones and faults, with locally associated extensional veins and hydrothermal breccias. They can coexist regionally with iron formation-hosted vein and disseminated deposits, as well as with turbidite-hosted quartz-carbonate vein deposits (Figure 8.2). They are typically distributed along reverse-oblique crustal-scale major fault zones, commonly marking the convergent margins between major lithological boundaries such as volcano-plutonic and sedimentary domains. These major structures are characterized by different increments of strain, and consequently several generations of steeply dipping foliations and folds resulting in a fairly complex geological collisional setting.

The crustal scale faults are thought to represent the main hydrothermal pathways towards higher crustal level. However, the deposits are spatially and genetically associated with higher order compressional reverse-oblique to oblique brittle-ductile high-angle shear zones commonly located less than 5 km away and best developed in the hanging wall of the major fault (Robert, 1990). Brittle faults may also be the main host to mineralization as illustrated by the Kirkland Lake Main Break; a brittle structure hosting the 25 Moz Au Kirkland Lake Deposit.

Figure 8.1: Setting of Greenstone Hosted Gold Deposits



From Poulsen et al., 2000

Stockworks and hydrothermal breccias may represent the main host to the mineralization when developed in competent units such as granophyric facies of gabbroic sills. Due to the complexity of the geological and structural setting and the influence of strength anisotropy and competency contrasts, the geometry of the vein network varies from simple (such as the Silidor Deposit, Canada) to more commonly complex with multiple orientations of anastomosing and/or conjugate sets of veins, breccias, stockworks and associated structures (Dubé et al., 1989; Hodgson, 1989, Robert et al., 1994, Robert and Poulsen, 2001).

Economic grade mineralization also occurs as disseminated sulfides in altered (carbonatized) rocks along vein selvages. Deposit shoots are commonly controlled by: 1) the intersections between different veins or host structures, or between auriferous structures and an especially reactive and/or competent rock type such as iron-rich gabbro (geometric ore shoot); or 2) the slip vector of the controlling structure(s) (kinematic ore shoot). For laminated fault-fill veins, the kinematic ore shoot will be oriented at a high angle to the slip vector (Robert et al., 1994; Robert and Poulsen, 2001).

At the district scale, the greenstone-hosted quartz-carbonate-vein deposits are associated with large-scale carbonate alteration commonly distributed along major fault zones and associated subsidiary structures (Dubé and Gosselin, 2007). At the deposit scale, the nature, distribution and intensity of the wall-rock alteration is largely controlled by the composition and competence of the host rocks and their metamorphic grade. Typically, the alteration haloes are zoned and characterized, at greenschist facies, by iron carbonatization and sericitization with sulfidation of the immediate vein selvages (mainly pyrite, less commonly arsenopyrite).

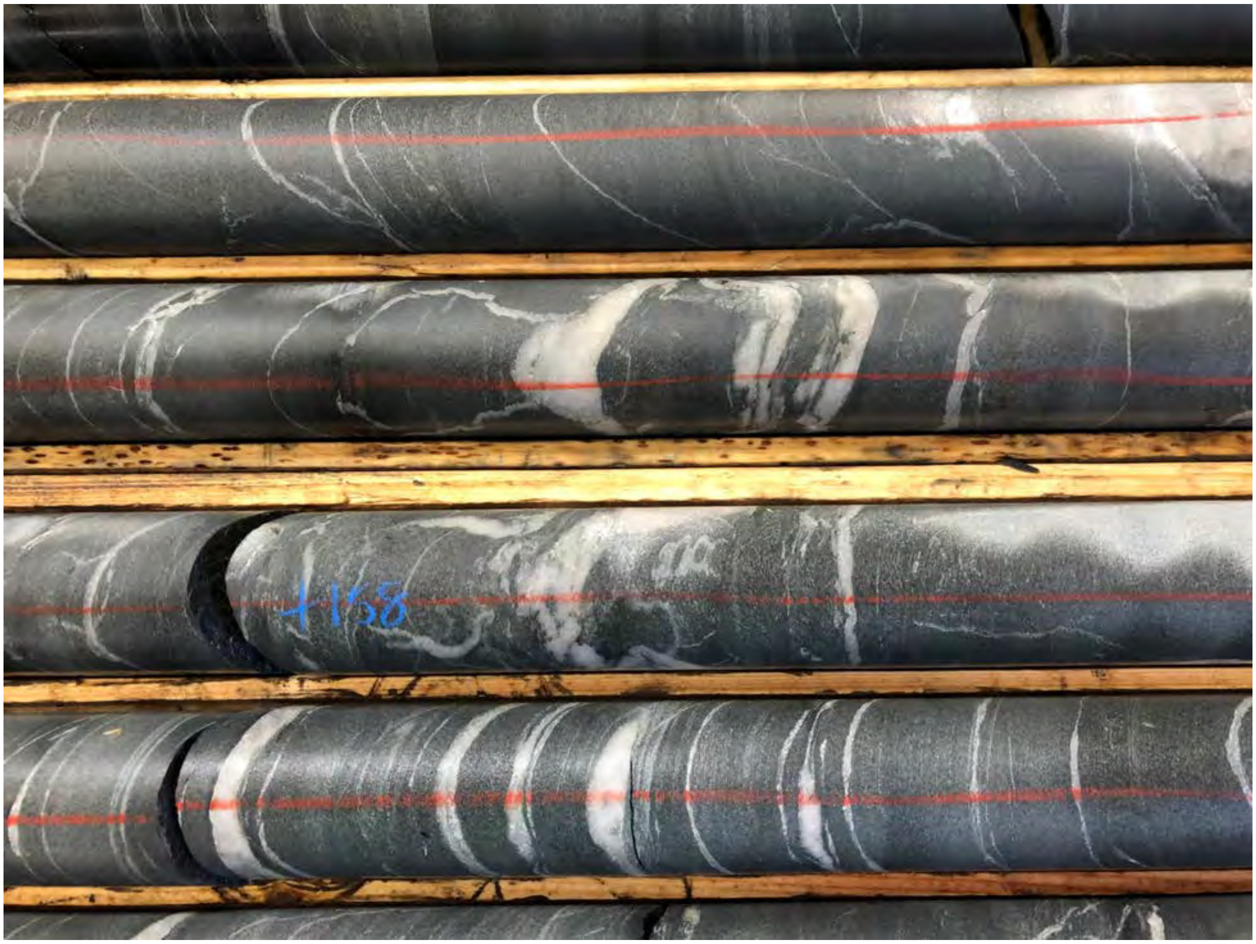
The main gangue minerals are quartz and carbonate with variable amounts of white micas, chlorite, scheelite and tourmaline. The sulfide minerals typically constitute less than 10% of the mineralization. The main ore minerals are native gold with pyrite, pyrrhotite and chalcopyrite without significant vertical zoning (Dubé and Gosselin, 2007).

The structurally controlled, high grade veins spatially related to the Hard Rock Porphyry in the Hard Rock and MacLeod-Cockshutt mines are similar to quartz-carbonate-sericite veins that host gold within many gold camps in Ontario (Porcupine, Kirkland Lake and Red Lake). The veins related to the Hard Rock Porphyry do not host significant tonnages of ore from past production despite their locally high grades. Numerous thin, gold-bearing quartz stringers occur along shear fractures in zones of faulting, folding and shearing at the contact with wacke and Hard Rock Porphyry. When stringers merge, elongate replacement or blow-out lenses up to 1 m long are formed. Normally, they occur as thin highly contorted veinlets that follow both shear and tension fractures and locally have a gash-like character. Carbonate (ankeritic-dolomite), sulfides (pyrite, pyrrhotite, arsenopyrite and chalcopyrite) and tourmaline are found to be associated with the quartz. Zones A through H were of this type (Horwood and Pye, 1951).

The greywacke (turbidite) associated mineralization is typically characterized by narrow, often sheeted, millimetre- to centimetre-scale veins with attendant but highly variable degrees of carbonate-sericite-pyrite alteration. This style of mineralization forms wide, low grade zones in the former Hard Rock, MacLeod-Cockshutt and Mosher mines. The F-Zone was the most spectacular zone, accounting for an orebody of some 10,000,000 t at 0.15 oz/ton Au (Macdonald, 1983b). The F-Zone produced the bulk of the tonnage that came from these mines from the 1950s to 1970.

Several diamond drill holes on the current program intersected significant widths of the F-Zone style mineralization. Figure 8.2 is an example of sheeted quartz-carbonate veinlets within weakly to moderately sericitized and carbonatized greywacke containing 1% to 5% fine grained arsenopyrite.

Figure 8.2: Sheeted Quartz-Carbonate Veins Hosted in Greywacke, DDH-19-54



8.2 Other Greenstone Gold Deposits

8.2.1 Brookbank

Economic concentrations of gold in the Beardmore-Geraldton area are typical of Archean epigenetic hydrothermal gold deposits normally considered to be mesothermal lode gold deposits. The gold mineralization is primarily located in areas of high strain and deformation with brittle structures providing a pathway and also hosting mineralization as veins or replacement zones with associated alteration. There are also low grade zones that locally have less obvious structural control, less veining, and less intense hydrothermal alteration on a hand specimen scale, but these clearly have strong deposit-scale structural controls.

Gold mineralization on the Brookbank Deposit is hosted within bands of intense deformation at the contact zone between domains of mafic flows and polymictic conglomerate. This contact zone straddles the 6.5 km east-west trending Brookbank shear zone. The mineralization occurs within quartz-carbonate veinlets/stringers, fractures and/or stockworks associated with hydrothermal alteration (Figure 8.3).

Figure 8.3: Exposure of the Brookbank Deposit Quartz Carbonate Veins/Stringers, Fractures/Stockworks

Micon, 2013.

Taking into account the deposit model discussed above, previous and current exploration activities have been focused on the contact zone between the sedimentary and volcanic assemblages within the confines of the Brookbank shear zone.

8.2.2 Key Lake

The Key Lake Deposit consists of an altered and mineralized felsic dyke in contact with sericitized and mineralized arkose. Higher grades appear to plunge to the west, following the plunge of folds in the area. This is a good example of stockwork quartz-carbonate with disseminated gold style of mineralization.

8.2.3 Kailey

Kailey is located within the area of the former Little Long Lac Gold Mine. The Little Long Lac Deposit occurs in the large Z-shaped minor fold on the north limb of the Barton syncline. The fold plunges 45 to 55° to the west. Numerous smaller flexures are superimposed, some of which are believed to have been formed during a later period of deformation. The deposit consists of more or less parallel quartz veins and stringers within fracture zones in massive arkose. For the most part, the sulfides are confined to narrow selvages and books of altered wall rock along and within the individual veins, although small amounts are commonly enclosed by the vein quartz itself. The quartz veins have, along their walls, narrow selvages, generally less than half an inch thick, of highly sheared and sericitized arkose impregnated with small amounts of finely divided sulfides, consisting mainly of pyrite and arsenopyrite.

In 1935, Bruce proposed three types of gold deposits for the Little Long Lac Gold Mine area:

- Shear zones in sedimentary rocks, along which narrow, but closely spaced quartz veins occur in parallel planes;
- Irregular veins of quartz accompanied by pyrite, filling fractures in iron formation;
- Zones of pyritization and silicification in both in sedimentary and intrusive rocks.

The most favorable type of sediments for deposits of the first type is the belt of Timiskaming sediments of massive and thickly-bedded greywacke or arkose that lies between the two northern bands of iron formation. It is in this belt that the veins of the Little Long Lac mine are located, and ae almost exclusively hosted within the Arkose.

9. AEXPLORATION

9.1 Hardrock Property

The first gold discovery in the area of the Property was made between 1916 and 1918. Since then, the Hardrock Project has been the subject of extensive exploration by a number of companies. In 1931, W.W. "Hard Rock" Smith discovered gold-bearing quartz stringers near the location of the Hard Rock Number One Shaft and Tom Johnson and Robert Wells discovered gold on Magnet Lake, which later hosted the Bankfield Gold Mine.

In 1934, the period of mine production in the area began with the Little Long Lac Mine, which was the first successfully producing mine in the area. To the west of the 1931 Hard Rock discovery, F. MacLeod and A. Cockshutt staked claims and continually explored the area throughout the 1930s and 1940s. By the late 1940s the F-Zone, a low-grade, large-tonnage orebody in greywacke, was identified on both the MacLeod-Cockshutt and Hard Rock Properties.

Production on the Mosher Long Lac Mine (located west of, and immediately down plunge of the same mineralized zones exploited in the MacLeod-Cockshutt Mine) began in 1962, then in 1967, the MacLeod-Cockshutt, Mosher and Hard Rock mines amalgamated and remained in production until 1970. The consolidated Hard Rock, MacLeod-Cockshutt and Mosher Mines had produced 2,075,074 ounces of gold at an average grade of approximately 0.14 ounces of gold per ton (~13 Mt @ 4.9 g Au/t) in the period from 1934 to 1970.

In the 1980s, Lac Minerals Ltd. reviewed the remaining underground reserves and conducted ground geophysical work and diamond drilling to target areas with open-pit potential.

In 1993 and 1994, Asarco conducted various types of drilling to evaluate the potential of the near surface portion of the F-Zone. Subsequently, Cyprus Canada Inc. signed various agreements with Lac Minerals Ltd and Roxmark Mines Ltd. to earn an interest in and acquire ground in the area. Cyprus then drilled 25 holes in 1996 and 1997 to help in better understanding and assessing the open-pit potential on the Property.

In 2007, Premier Gold Mines Limited ("Premier") began assembling the current property. Results of 1,629 drill holes were included in the 2016 Feasibility Study ("FS").

In February 2015, Premier and Centerra Gold Inc. formed a definitive 50/50 partnership for development of the Hardrock property. In July 2015, the joint partnership was named Greenstone Gold Mines GP Inc ("GGM").

In 2016, GGM conducted 34 km of induced polarization ("IP") surveys over the Little Long Lac, MacLellan, Magnet, Bankfield and Bankfield West deposits and 23 km of IP over the Hardrock Deposit. This was done mainly to test the geophysical response and establish a signature for these deposits. Outcrop stripping and channel sampling were also conducted at the F-Zone, Porphyry Hill, Headframe, Headframe East and "OPP" exposures. Results of this work are detailed in the 2016 Technical Report.

In 2018, 405 RC holes totaling 19,995 m and 62 blast holes totaling 535 m were drilled to provide further definition of near surface gold mineralization in 5 different areas on the Property.

A total of 76 RC holes (5,946 m) and 54 diamond drill holes (12,108 m) were drilled on the Hardrock Deposit in 2019.

9.2 Other Properties

Since Premier started his acquisition on the other projects within the Hardrock Property in 2007, mostly of the exploration work have been focused on diamond drilling. The details of any historical exploration program performed before Premier have been summarized in Section 6 - History.

9.2.1 Brookbank

Between 2012 to 2013, two (2) hole drill program was completed on the Brookbank Project by Premier, totalling 1,393 m. These holes were designed to target IP anomalies near the known gold deposit at Brookbank.

In 2016, GGM completed 14 diamond drill holes for a total of 6,377.4 m of drilling on the project. In addition, orientation till and soil surveys were conducted over the Brookbank, Brookbank East and Patter Lake areas, while limited ground magnetics and outcrop channel sampling was conducted over Brookbank East, along with selected re-logging/re-sampling of diamond drill holes.

9.2.2 Kailey and Key Lake

2011 exploration activities had a focus on in-fill drilling proximal to the historic Little Long Lac and Brookbank Gold Mines to delineate previously discovered high-grade zones in the main resource areas estimated by previous operators. The details of each drilling program completed by Premier and GGM are fully described in Section 10 - Drilling.

There has been no new exploration field work conducted on the Brookbank, Key Lake, Kailey or other satellite properties since the 2016 Feasibility Study. Work during this time has consisted mainly of data compilation and selected re-logging of diamond drill holes.

10. DRILLING

10.1 Hardrock

Information in this section was provided by the Greenstone Gold Mines ("GGM") exploration team who has conducted the drilling programs in 2018 and 2019 under the supervision of G Mining Services Inc. ("GMS"). GMS has reviewed this information and compiled the following section regarding drilling practises at the Project. All drilling carried out prior to the 2016 National Instrument 43-101 ("NI 43-101") Technical Report are described in Section 6 - History of this Report.

Over the years, different drill core diameters have been used on the Hardrock Deposit. Recent drill holes at the Hardrock Project are mostly drilled with NQ diameter core. Table 10.1 summarizes the core diameter used in the different years, with recent reverse circulation ("RC") drilling also shown.

Table 10.1: Number of Drill Holes and Core Size per Year

Year Drilled	DDH Count	Core Size
1987	34	BQ
1988	33	BQ
1993	27	BQ
1994	76	BQ
1995	7	BQ
1996	24	Unknown
2009	340	NQ
2010	243	NQ
2011	166	NQ
2012	126	NQ
2013	278	NQ
2014	128	NQ
2014	1	PQ
2015	117	NQ
2018	405	RC
2019	76	RC
2019	54	NQ
Unknown	29	Unknown
Total	**2163**	

10.1.1 Collar Locations, Orientations and Down Hole Surveys

Collar locations for the drill holes on the Hardrock Project were determined using a cut grid or a hand-held GPS. Subsequent to completion, the collars were located, depending on the years drilled, using either a GPS, a Trimble and more recently since 2014 the more precise Trimble RTK survey instrument. A total of 55% of the holes drilled prior to 2013 have been surveyed using a hand-held GPS.

Whenever it was possible, casings were left in the ground after drilling operations. A collar re-survey campaign, using the Trimble RTK survey instrument, took place in the summer of 2014 for a total of 536 drill holes for which casing was found. Of these 536 resurveyed collars, 489 were previously surveyed by a handheld GPS. Following the ranking system described below, the Trimble survey replaced the original survey improving the precision of the collar location for 30% of the drill holes in the database.

Once the holes were drilled, the drill hole azimuth and precise UTM coordinates were determined by placing an APS unit on the drill casing. The downhole dip and drill hole orientations were surveyed using a gyroscope unit (REFLEX Gyro™). The UTM Coordinate System, NAD 83, Zone 16, is used to record the locations (x, y, and z) of the drill collars.

For the 2018 and 2019 drilling programs, the site surveyor and geologists spotted the reverse circulation grade control ("RCGC") and blast holes using a Trimble device with RTK base station using the coordinates planned by GMS or GGM. In the event of unstable or poor ground access, the hole was moved a few metres. The drill is aligned to the proper azimuth and dip using a Reflex Astronomic Positioning System ("APS"). Down-hole surveys were taken every 30 m in the diamond drill holes using a Reflex EZ-Gyro™ instrument.

10.1.2 Core Marking and Logging Procedures

The first time the core is handled is at the drill rig by the driller helper who takes the core from the core tube and places it in core boxes, marking off every 3 m. Once a core box is full, the helper wraps the box with tape or wire depending on the preference of the drilling company. At the end of each shift, the core is delivered to the core shack. GGM personnel remove the wire or tape and bring the boxes to the logging trailers. The technicians rotate the core so that all the pieces slant one way, at about a 45° angle. They also check that distances are correctly indicated on the wooden blocks placed every 3 m. If there is a mistake on any of the blocks, the Project Manager is informed, and the Drill Foreman is brought in. The core is measured in each box and the box labelled. Red lines are drawn along the centre of the core to provide a reference for the core cutters. Geological technicians and geologists are then responsible for taking photographs of the core.

Rock quality designation ("RQD") is done by either geologists or the geological technicians. Any breakage under 10 cm is recorded. Core from the Hardrock Deposit is of very good quality and recovery is high.

Logging of diamond drill core was performed on site by geologists contracted to GGM. Logging was typically recorded by hand onto paper or a notebook and then transcribed later into LogChief software. Geologists note intervals of varying lithology, accessory minerals, the type and style of any veins (e.g. quartz-carbonate veins), the type and habit of sulphide mineralization (pyrite, arsenopyrite, pyrrhotite) and whether the unit appears folded and relevant structural measurements (e.g. bedding, foliation, fracture or vein orientation, fold axes). Visible gold ("VG") is also noted, if present.

Samples were generally taken along the entire length of the holes (continuous sampling) and are entered in the related Datashed software. Sample length typically ranged from 0.5 m to 1.5 m. In clearly mineralized zones where visible gold is present, the geologist will place a piece of colored ribbon in the core box. The core cutters, upon seeing this, will cut a piece of brick after the sample, in order to clean any residual gold from the saw. Cutting was accomplished with two Vancon core saws. Cuttings are allowed to decant in a series of settling tanks and the water is recirculated back to the cutting area. These tanks are cleaned periodically to avoid contamination. The individual cut samples were placed into polyethylene bags, along with the sample tag and sealed. Samples were then placed into rice bags (approximately 8 to 10 samples per bag) and taken Activation Laboratories Ltd. ("Actlabs Geraldton") facility in Geraldton.

10.1.2.1 RC Chip Logging Procedures

During sampling of the RCGC drill holes, approximately 100 g of the 10 kg representative samples, were collected and put into wet and dry chip trays (50 g in each) for the geologists to log. The dry samples were placed directly into the tray. An equal amount was sieved and cleaned and placed into the wet tray. The wet samples were subsequently logged by geologists using a SciOptic fiber optic microscope. Information on lithology, alteration and mineralization was recorded in DataShed by the onsite geologists.

The off-site storage of core, RCGC duplicate samples, assay pulps and rejects are located at the Magnet Mine Site.

10.1.3 Drilling and Re-sampling Included in the 2016 Mineral Resource Estimate

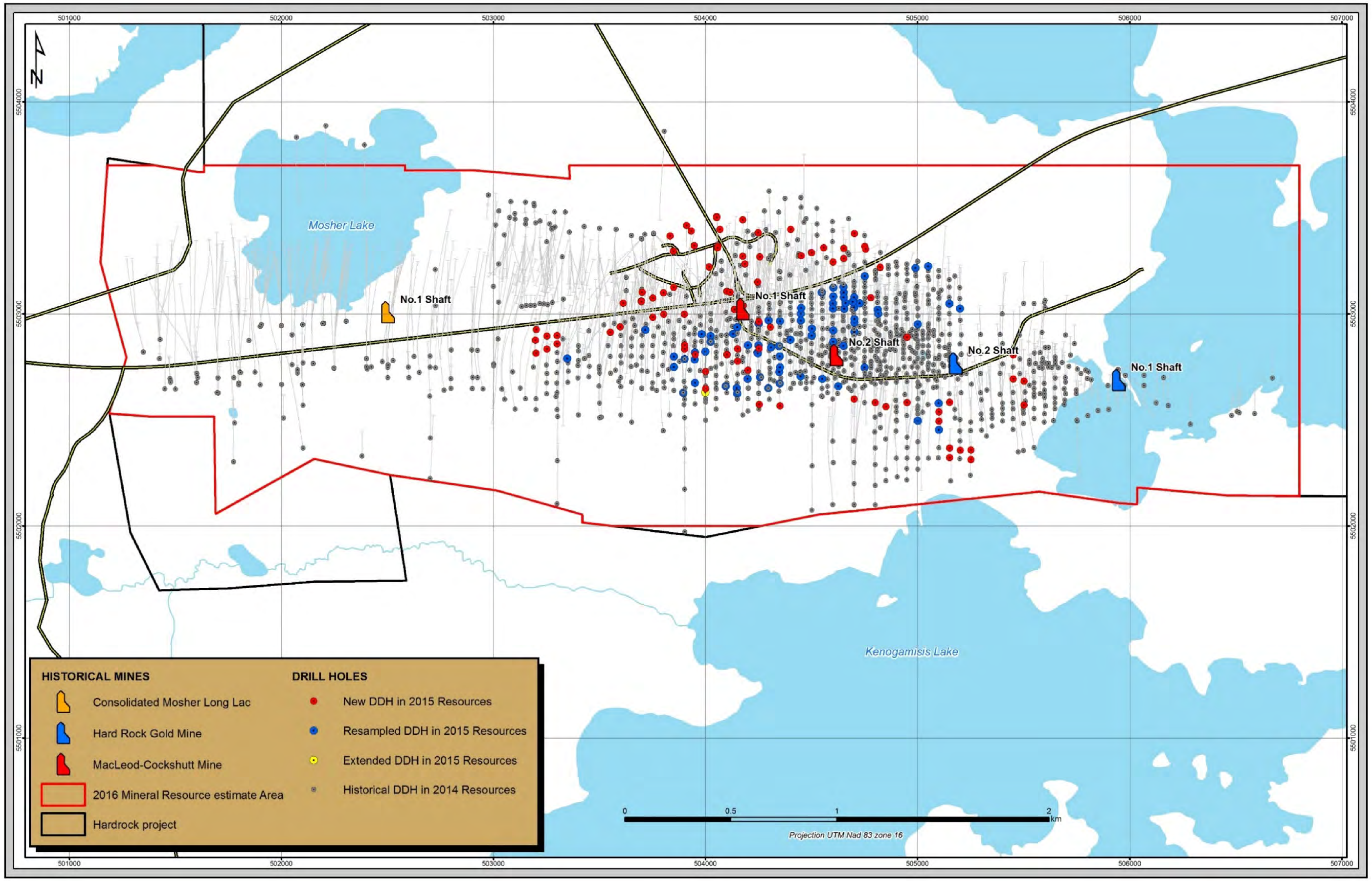
Between May 26, 2014 and November 18, 2015, GGM added 157 diamond drill holes ("DDH") on the Hardrock Deposit for a total of 54,027 m. One diamond drill hole (MM043) included in the 2014 Mineral Resource Estimate ("MRE") was also deepened, from 456 m to 655 m, representing a total of 199 m of new meterage.

Seventy-nine (79) historical DDH were re-sampled to add new assay results in the 2016 MRE. These holes represent a total of 8,733 m and 6,411 samples included in the 2016 Project database.

Figure 10.1 shows the locations of the drill holes included in the 2016 MRE. The new drill holes (red), the re-sampled DDH (blue) and extended drill hole (yellow) that are included in the 2016 MRE are presented in Figure 10.1.

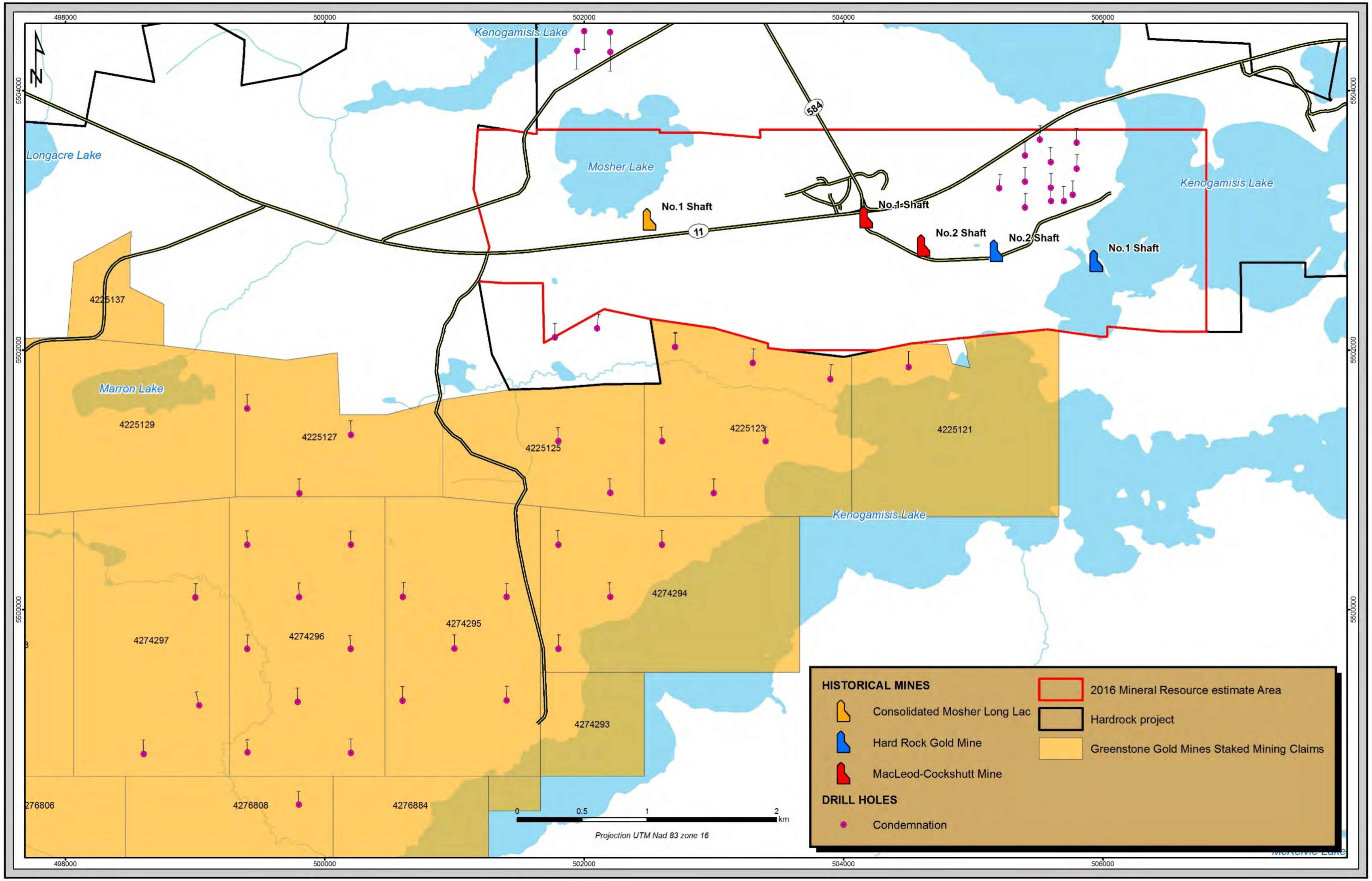
Figure 10.2 shows the locations of the condemnation drill holes drilled in the area of the Hardrock Deposit. A total of 55 condemnation DDH totalling 8,512 m were drilled by GGM.

Figure 10.1: Location of Greenstone Drill Holes used in the 2016 MRE, Prior to the 2018 and 2019 Drilling Programs



Source: Innovexplo, 2015 with modifications by GGM, 2016

Figure 10.2: Location of Greenstone Gold Mines Condemnation Drill Holes in the Hardrock Deposit Area



Source: GGM, 2016

10.1.4 2018 RC Grade Control and Blasthole Drill Program

The 2018 RCGC and down-the-hole ("DTH" or Blast hole) drilling campaigns were resource definition programs, designed to de-risk the project and to focus on increasing the confidence level in the mineral resources in the initial years of production. The drilling took place on five key areas, outlined in Table 10.2. Area 1 (not shown) was not accessible due to flooding.

Table 10.2: Targeted Areas for the RCGC and DTH Drilling at Hardrock

Area	Zone	Lithology
Area 2	Headframe	Interbedded BIF and greywacke
Area 3	F-Zone	Primarily greywacke with lesser BIF and porphyry
Area 4	Porphyry Hill	Primarily porphyry with lesser greywacke and BIF
Area 5	Headframe East	Interbedded BIF and greywacke
Area 6	SP-Zone	BIF, porphyry and greywacke

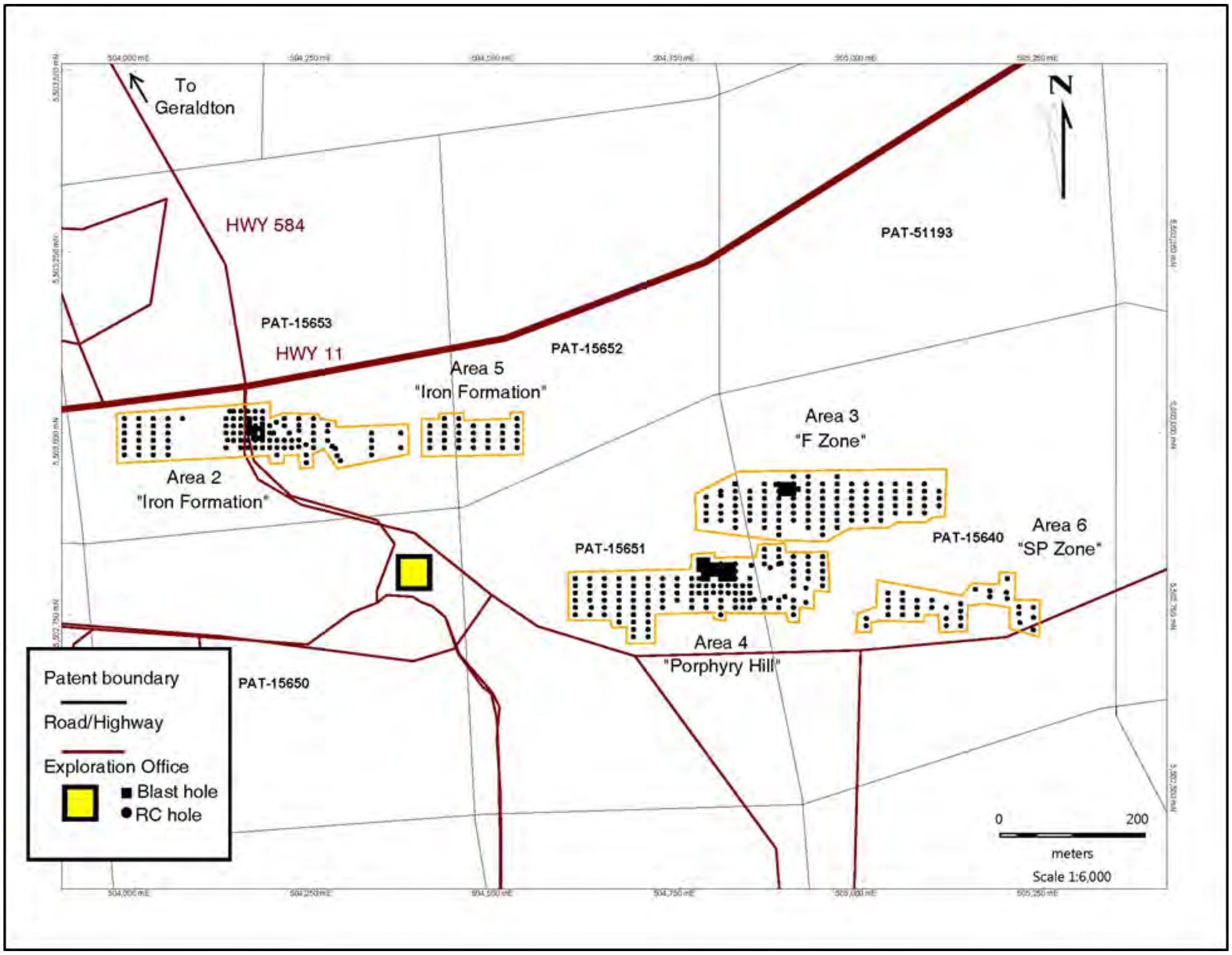
From May 24, 2018 to September 6, 2018, 405 RCGC drill holes were completed totaling 19,995 m on the Property based in Geraldton, Ontario. All RCGC drilling was completed by NPLH Drilling based in Timmins, Ontario. The program targeted five areas which were defined by their geographic and lithological properties (Table 10.2).

All RCGC drill holes were planned by Mr. Réjean Sirois of GMS and qualified person ("QP") of this current MRE. RCGC holes were planned with a spacing of 10 m in the North to South direction and 20 m in the East to West direction. On average, the RCGC holes were 50 m deep and had a dip of -50 degrees, oriented due North or South and planned within five significant mineralized areas. The results obtained from the RCGC drilling program confirmed the grade continuity in all areas. All RCGC material (chip trays from logging, rejects and representative samples) are stored on site in sea containers at GGM's Magnet Property.

During four days in early July, 62 blast holes totaling 535 m were drilled by Epiroc. The program occurred concurrently with the RCGC drilling program and aimed to further increase the confidence in the mineral resources in the F-Zone, headframe and porphyry hill area (see Table 10.2) and to test the performance and viability of blasthole drilling for the Hardrock Deposit.

The blast holes were planned with a tighter spacing of approximately 6 m in the North to South and East to West directions. The blast holes were on average 10 m deep and drilled vertically. Figure 10.3 presents the surface plan of RCGC drilling and blastholes from 2018.

Figure 10.3: 2018 RCGC and Blast Hole Drilling Locations at the Hardrock Project



10.1.5 2019 Drill Program

The 2019 drilling program consisted of 76 RCGC drill holes totaling 5,946 m of which 5,527 m were assayed, and 54 NQ size DDH for 12,108 m of which 10,470 m was assayed. The first drill was mobilized on February 12 ,2019, drilling commenced February 19, 2019 and continued through to April 25, 2019. Both RCGC and DDH drilling were contracted to NPLH Drilling, based in Timmins, Ontario. These were resource definition and grade control programs, designed to provide better definition in high potential areas of the Project and to increase the confidence level in the Mineral Resource in the initial years of production.

In 2019, all RCGC and DDH holes were planned in conjunction with the representatives of the joint-venture partners of the Project (Centerra Gold and Premier Gold). RCGC holes were planned with a spacing of 20 m in the North to South direction and 20 m in the East to West direction. On average, the RCGC holes were 100 m deep and had a dip of -50 degrees, oriented due North or South. Figure 10.4 illustrates the RCGC and DDH programs performed by GGM in 2019 with the 2016 FS Pit Design in the background for scale.

The 2019 drilling program outcomes are detailed below:

- RCGC drilling was spatially limited to the SP-Zone and F-Zone to confirm grade continuity for benches 4 to 7;
- 70 m vertical (or 7 benches) were drilled at an average spacing of 20 m x 20 m inside an area already drilled in 2018;
- Diamond drilling intersected the majority of mineralized domains, and infilled gaps in the drill spacing in the central portion of the pit;
- Grades in drilling compared well with block model grades predicted in a 2018 interim block model.

Figure 10.4: 2019 Drilling Programs - Hardrock Project

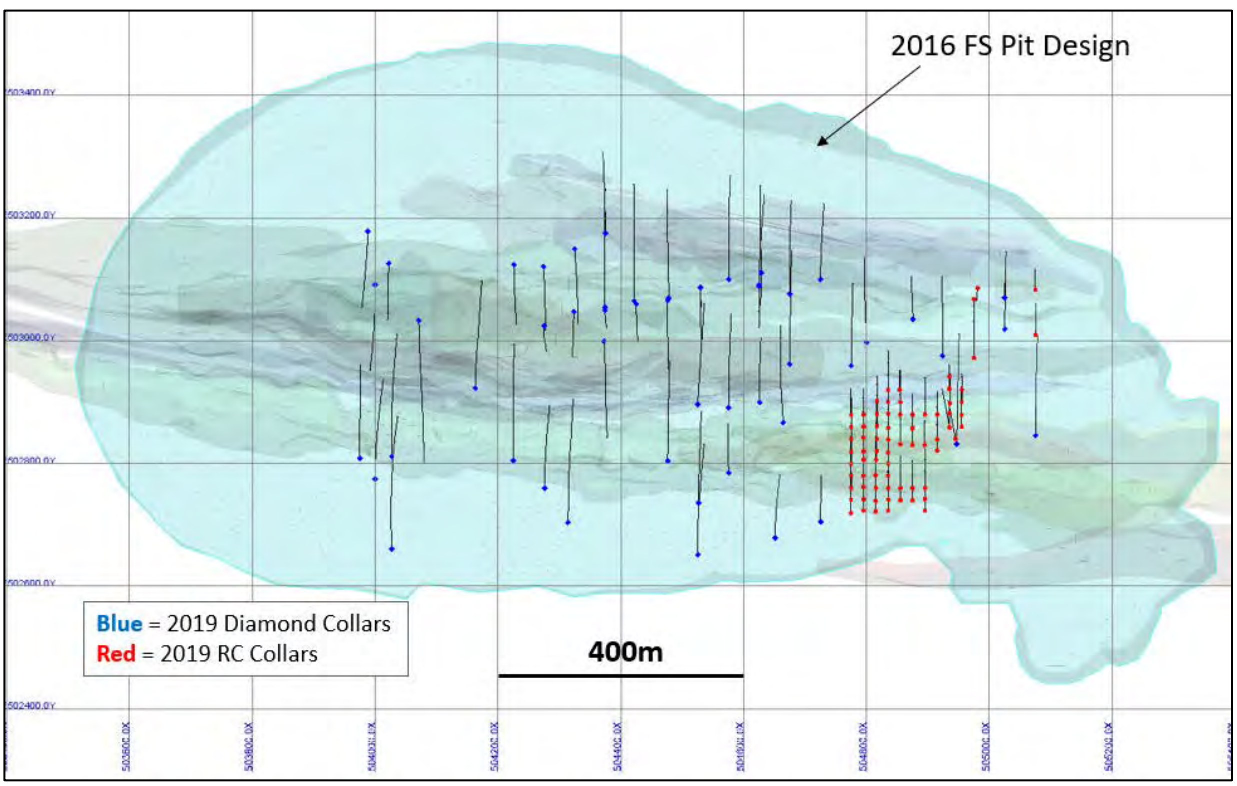


Figure 10.5 and Figure 10.6 show both drill rigs used to carry out the resource definition drilling campaigns in 2019.

Figure 10.5: Hardab 7000 Maxidrill RC Drill Rig

Figure 10.6: Diamond Drill Rig

10.1.6 QP Opinion on Drilling – Hardrock Project

During the Hardrock site visit, GMS reviewed drilling procedures, observed RC drilling, and inspected sampling facilities and core storage facilities. Core recovery is excellent throughout the deposit, and recoveries from near-surface RC drilling were deemed acceptable. Drilling methods (both diamond drilling and RC drilling) adhered to industry standard practises, and representative samples were obtained.

Overall, GMS considers the data obtained from the exploration and drilling programs carried out at the Hardrock Property to be reliable and meet industry standards

10.2 Other Greenstone Gold Property Deposits Brookbank, Kailey (Little Long Lac) and Key Lake

The drilling information described in the following section was obtained from the GGM exploration team, the 2009 Technical Report prepared by Scott Wilson RPA and the 2016 Technical Report prepared by GMS in 2016.

10.2.1 Drilling Procedures

All Ontex diamond drilling was completed from surface using NQ wire-line stabilized hexagonal core barrels with an 18 in. long shell. For deep holes hexagonal core barrels are first utilized. Wedge cuts were completed on parent holes. Hole collars are well marked with casings generally left in the hole. Core diameter was usually NQ.

Diamond drilling for 2008 was contracted to Chibougamau Diamond Drilling ("Chibougamau Drilling") based in Chibougamau, Québec and Major Drilling out of New Brunswick. The drill rigs were mounted on skids and dragged into position using a skidder or bulldozer

During the 2016 Brookbank drilling program, one (1) hole was drilled by Confederation College and the other 13 holes by ForageG4 drilling. Drill holes were surveyed approximately every 10 m using a TN14 instrument for single shot surveys and an EZ-Gyro for multishot surveys.

Core diameter for Premier Gold Mines Limited ("Premier") drilling is NQ size (48 mm in diameter) and all drilling was recorded in metres. The core was placed in three-row wooden core boxes provided by the contractor. The boxes and distance (in metres) were labelled by the drill crews. Upon receipt at the Premier core shack, the boxes were then labelled with permanent metal tags according to drill hole number, box number, and metres down-hole. After the core was logged and sampled, it was permanently stored in core racks at the Magnet mine site or at a site constructed in 2009 on Old Arena Road near the Premier core shack in Geraldton.

10.2.2 Collar Locations, Orientations and Down Hole Surveys

Collar locations for the Ontex drilling for the Brookbank Property were located using a hand-held GPS ("Garmin Etrex GPS"). Holes were later resurveyed by a professional land surveyor using differential GPS.

At Brookbank, a large collar resurveying campaign was undertaken in 2008 by JDB using a differential GPS. The objective was to resurvey drill holes in NAD83 as previous drill collars were surveyed in local coordinates. A total of 157 drill collars were found and resurveyed. The remaining 150 drill holes were converted from local grid to NAD83 using the prior surveys to control the grid transformation.

Collar locations for the Premier drilling were located using a cut grid or by using a hand-held GPS. Subsequent to completion, most collars were located using a Trimble GPS survey instrument. Some hole locations are only recorded to the nearest metre, even though more accurate measurements were possible.

In 2007, a Reflex Instruments down-hole survey tool provided by the drill contractor was used with surveys typically taken every 50 m. A Reflex Instruments Maxibore tool was also used for down-hole surveys starting in November 2007 as standard practice even though azimuth readings were still an issue. This survey tool was operated by Premier employees and has been used for approximately 95% of the holes since it became available. In May-June 2010, Premier changed to an Icefields Gyro survey tool to achieve more efficient and more accurate survey data. In late 2009 and again in October 2010, a survey determined drill hole orientations using a gyroscope at surface on casings for 79 historic holes and 310 holes drilled by Premier.

10.2.3 Core Logging and Sampling

The geologist prepared a detailed geological log including lithology, veining, alteration, mineralization, structures (oriented core), surveying, assays (gold and trace elements) and magnetic susceptibility. Magnetic susceptibility was collected every meter downhole using the MPP susceptibility meter from GDD Instruments based out of Quebec City.

The drillers provided the “ori-marks” and core was then oriented by geotechnicians and geologists at the logging site to obtain alpha and beta measurements and ultimately strike and dip of geological structures. Although core recovery for the program was very good, the high fracture zones (“HFZs”) encountered in every hole meant that only approximately 60% of the core could be oriented.

The geologist then identified and marked the beginning and the end of the sampling intervals. Upon completion of the logging and demarcating the sample intervals, technicians sawed the core longitudinally in half with a diamond saw, except for material which was highly fractured and contained clay minerals, which was divided manually with hammer and chisel. One half of the core was bagged, tagged with a sample number and then sealed; the other half was put back in the core boxes and kept as a reference and check sample in the event that duplicate assays are required. Generally, samples of 1 m length were taken in longer sections of similarly mineralized rocks; however, sample size was reduced to as low as 0.4 m in areas of particular interest, or where lithology and mineralization were distinct.

Premier re-sampled and analyzed the holes drilled by their predecessor as part of their validation of previous work.

10.2.4 Brookbank

10.2.4.1 Summary of Drilling Campaigns

Since the mid-1940s, numerous drill programs have been carried-out on the Brookbank, Cherbourg, and Foxear areas. The details of all historical drilling programs performed before Premier have been summarized in Section 6 - History of this Report.

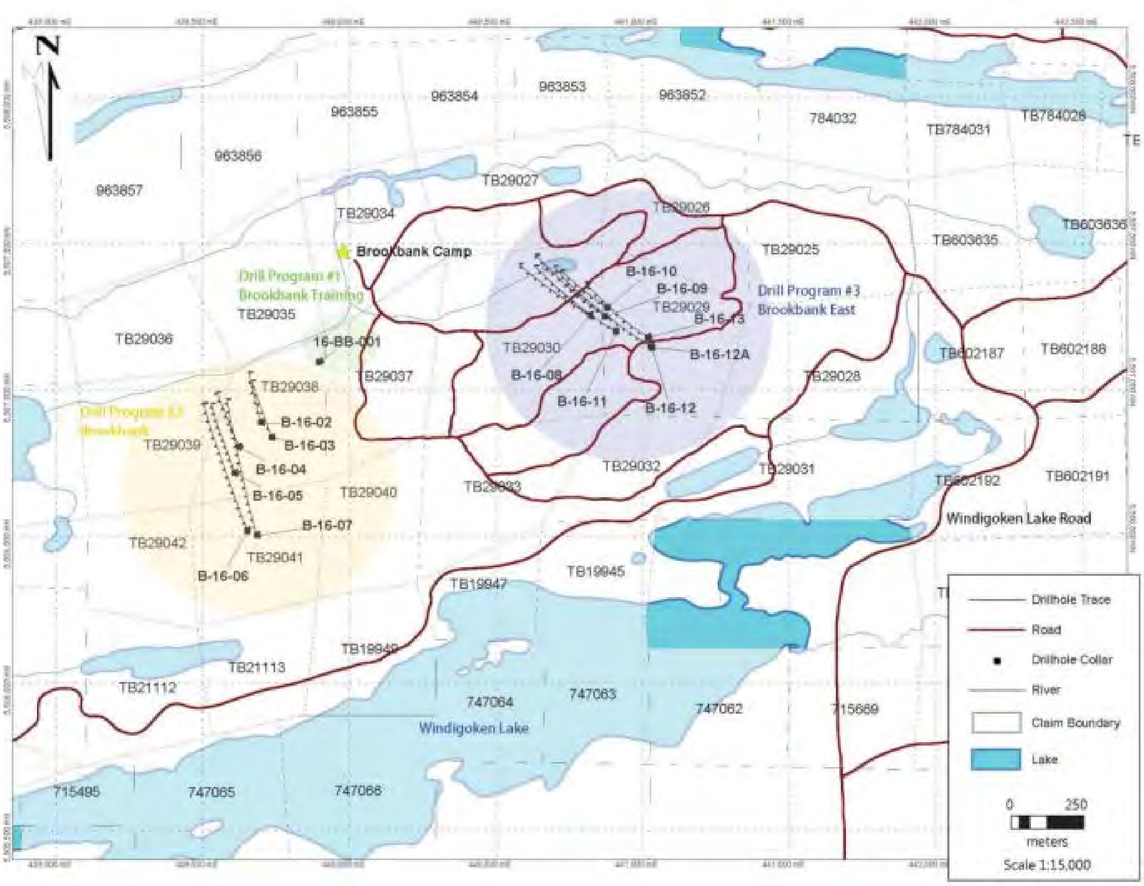
During 2016, the Brookbank Deposit was operated by the GGM. and Premier joint venture.

Two drilling programs were completed on the Brookbank Property between July 17th and December 16th, 2016; the Brookbank Deposit infill program and Brookbank East exploration drilling program. In total, 14 holes (1 abandoned, therefore 13 exploration holes) were drilled during the two drilling programs (Figure 10.7).

Between October 24th, 2016 and December 4th, 2016, the infill program consisted of six holes which targeted the Brookbank Mineral Resource area. The program was designed to increase confidence in the resource and to test the interpretation of the geological model. Results obtained from the drilling program were encouraging, increasing confidence in the high-grade portion of the resource and defined targets at depth.

Exploration drilling at Brookbank East, approximately 1 km east of the second exploration program, was performed between November 8th to December 16th, 2016. The purpose of the program was to test the intersection of the main mineralized iron-carbonate shear zone and many oblique structures observed at outcrop and interpreted from the detailed magnetics surveys.

Figure 10.7: Six (6) Drill Traces of Two Drilling Campaigns Performed at Brookbank in 2016 (green – Brookbank training, blue – Brookbank East, yellow – Brookbank)



A summary of all the diamond drilling completed on Brookbank Property between 1944 to 2016 is shown in Table 10.3.

Table 10.3: Summary of Brookbank Project Drilling Programs

Year(s)	Company	Zone								Total			
		Brookbank		Cherbourg		Foxear		Other					
		No. of Holes	Metres	No. of Holes	Metres	No. of Holes	Metres	No. of Holes	Metres	No. of Holes	Metres	No. of Samples	Metres Assayed
1944	Noranda Exploration	40	1,860							40	1,860	470	575
1975	Lynx Canada Exploration	6	376							6	376	87	45
1981	Metalore Resources Ltd.	30	3,569							30	3,569	1,765	2,174
1982	Metalore Resources Ltd.	1	106			4	453			5	559	116	133
1983	Metalore Resources Ltd.	40	3,785	7	597					47	4,382	1,646	1,734
1984	Metalore Resources Ltd.	31	9,795			14	963			45	10,758	1,018	1,063
1985	Metalore Resources Ltd.					12	1,242			12	1,242	65	42
1986	Metalore Resources Ltd.	8	3,499	49	5,101					57	8,600	586	397
1987	Hudson Bay Mining	42	19,359							63	25,193	2,723	2,303
	Metalore Resources Ltd.	2	560	9	2,487	10	2,787					290	183
1989	Placer Dome Inc.	19	8,354	5	1,538	2	984	3	934	29	11,810	1,197	1,460
1993	Metalore/Ontex			6	1,546					6	1,546	38	29
1994	Metalore Resources Ltd.			9	1,109	4	1,376			15	2,810	81	65
	Metalore/Ontex					2	325					13	7
1995	Metalore Resources Ltd.					5	2,774			5	2,774	44	39
1999	Ontex Resources Ltd.	16	4,738	13	6,706	3	1,295			32	12,738	2,082	2,077
2000	Ontex Resources Ltd.	34	17,120	5	1,564	13	4,792			52	23,476	2,148	2,185
2001	Ontex Resources Ltd.			9	2,523	12	4,530			21	7,053	392	324
2006	Ontex Resources Ltd.	14	3,000							14	3,000	870	900
2007	Ontex Resources Ltd.	7	1,208							7	1,208	384	417
2008	Ontex Resources Ltd.	16	5,638	9	3,823					25	9,461	928	833
2009	Ontex Resources Ltd.	50	23,291							50	23,291	2,575	3,098
2011	Premier	2	1,962							2	1,962	79	88
2012	Premier	3	1,937							3	1,937	431	629
2013	Premier	2	1,393							2	1,393	244	305
2016	GGM	14	6,377							14	6,377	5,152	4,956
Total		**377**	**117,928**	**121**	**26,993**	**81**	**21,521**	**3**	**934**	**582**	**167,376**	**25,424**	**26,061**

10.2.5 Kailey ("Little Long Lac")

The description of drilling method, surveying, and core logging procedures described in the previous sections are also applicable to the Kailey Deposit. A significant amount of historical information was available and digitised from hard copy records by GGM in 2019 and 2020, although none of this was verifiable and was not incorporated into the mineral resource estimate. On the more recent drilling undertaken by Premier is considered as verifiable.

10.2.5.1 Drilling Campaigns

In late 2007, Premier focused their drilling program in the area around the historic Little Long Lac mine. Eight holes were drilled approximately 200 m southeast of the old mine headframe. All the holes were orientated N334 and they had as target the undeveloped vein no.9. During the mine was in production, this mineralized structure was mined-out on levels 2nd, 4th and 16th. The vein is sub-vertical in about 50 m in strike length, and steeply dips to the west-southwest at around 60 degrees. Six of the eight drilled holes intercepted the structure returning anomalous gold values. The initial program also enabled to discover three additional parallel zones to south of the structure vein No.9 which were called veins No.10, No.11 and No.12.

Premier continued the drilling programs on the Little Long Lac Property area during 2008. The exploration was concentrated between two zones, the first target aimed to define mineralization on veins No.9, No.10 and No.11. the second was focused on the newly- discovered Kailey Zone. The extensions of the mineralized veins No.9 and No.10 were expanded down plunge and in-filled area was increased by the drilling.

The Kailey Zone is a low-grade bulk tonnage target proximal and parallel to the historic Little Long Lac Gold Mine workings. Drilling in the Kailey Zone has identified an area of mineralization characterized by a network of randomly oriented quartz-carbonate veins and stringers with traces of disseminated pyrite and arsenopyrite and visible gold, hosted in an altered arkose (sericite + lesser carbonate).The Kailey Zone has two parallel horizons (K1 - North and K2 - South) which converge in the central and eastern sections.

In 2011 exploration had a focus on in-fill and step-out drilling proximal to the main resource areas. The high-grade past producing Little Long Lac. The latest drilling programs has demonstrated that the Kailey mineralized zone is a shallow and low-grade potential open pit mineral resource.

Table 10.4: Summary of the Kailey Project Drilling Programs

Year(s)	Company	Drill Hole Type	Zone Kailey (Little Long Lac) No. of Holes	Metres	No. of Samples	Metres Assayed
2007	Premier Gold Mines	Surface DDH	8	2,625.8	2,525	2,350.1
2008	Premier Gold Mines	Surface DDH	68	25,452.4	23,840	23,579.9
2011	Premier Gold Mines	Surface DDH	6	6,520.4	1,153	1,613.7
Total Resource Drill Holes			**82**	**34,598.6**	**27,518**	**27,543.7**

10.2.6 Key Lake

The description of drilling method, surveying, and core logging procedures described in the previous sections are also applicable to the Key Lake Deposit.

10.2.6.1 Drilling Campaigns

All drilling campaigns at Key Lake prior to Premier in 2011 are described in Section 6 - History.

The exploration program by Premier in 2011 was designed to expand the footprint of the Key Lake Deposit trend along strike and aimed to test the down-plunge potential of some of the higher-grade gold values within the deposit. Premier drilled eight holes totalling 3,190 m and including 1,189 of assayed metres.

The 2011 drill program has successfully extended mineralization approximately 600 m along strike to the west of the core area, with the best results coming from the westernmost 200 m, where it remains wide open for expansion.

Table 10.5 summarizes the metres for the 1974 to 2011 drilling programs on the Key Lake Project.

Table 10.5: Summary of the Key Lake Project Drilling Programs

Year(s)	Company	Zone			
		Key Lake			
		No. of Holes	Metres	No. of Samples	Metres Assayed
1974	Jelex Mines Ltd.	2	251.2	-	-
Mid-80-1990	Dome Exploration	116	19,891.3	4,324	4,171.1
1995	Cyprus Canada	13	2,270.0	1,300	1,566.8
2010	Goldstone Resources	59	12,422.0	5,433	5,369.9
2011	Goldstone Resources	114	25,894.6	9,750	10,791.1
2011	Premier Gold Mines	8	3,190.0	896	1,188.8
Total		**312**	**63,919.1**	**21,703**	**23,087.7**

10.2.7 QP Opinion on Drilling – Brookbank, Kailey and Key Lake

During the Brookbank, Kailey and Key Lake site visits, GMS reviewed drilling procedures, sampling facilities and core storage facilities. Core recovery is excellent throughout the three deposits. There are no drilling, sampling or recovery factors that could materially impact the accuracy and reliability of the assay results. Overall, GMS considers the data obtained from the exploration and drilling programs carried out at the Brookbank, Kailey and Key Lake Properties to be reliable and meet industry standards

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11. SAMPLE PREPARATION, ANALYSES AND SECURITY

The following sections describe Greenstone Gold Mines ("GGM") sample preparation, analysis, and security procedures for the reverse circulation grade control ("RCGC") and diamond drill hole ("DDH") drilling programs performed at the Project since 2012. Data pertaining to sampling, analytical, security, and quality assurance-quality control ("QA/QC") protocols were supplied to G Mining Services Inc. ("GMS") from GGM's geology team and reviewed accordingly.

11.1 Hardrock

11.1.1 Laboratory Accreditation and Certification

The Geraldton facility belonging to Activation Laboratories Ltd ("Actlabs Geraldton") was used for the entire drilling and channelling programs. Actlabs Geraldton has received ISO 9001:2008 certification through Kiwa International Cert GmbH. Actlabs Geraldton is an independent commercial laboratory.

All re-assaying of batches (pulps) was undertaken at ALS-Chemex in Thunder Bay. ALS-Chemex laboratory is part of the ALS Global Group and has ISO 9001 certification and ISO/IEC 17025 accreditation through the Standards Council of Canada. ALS is an independent commercial laboratory.

11.1.2 GGM Sampling and Security

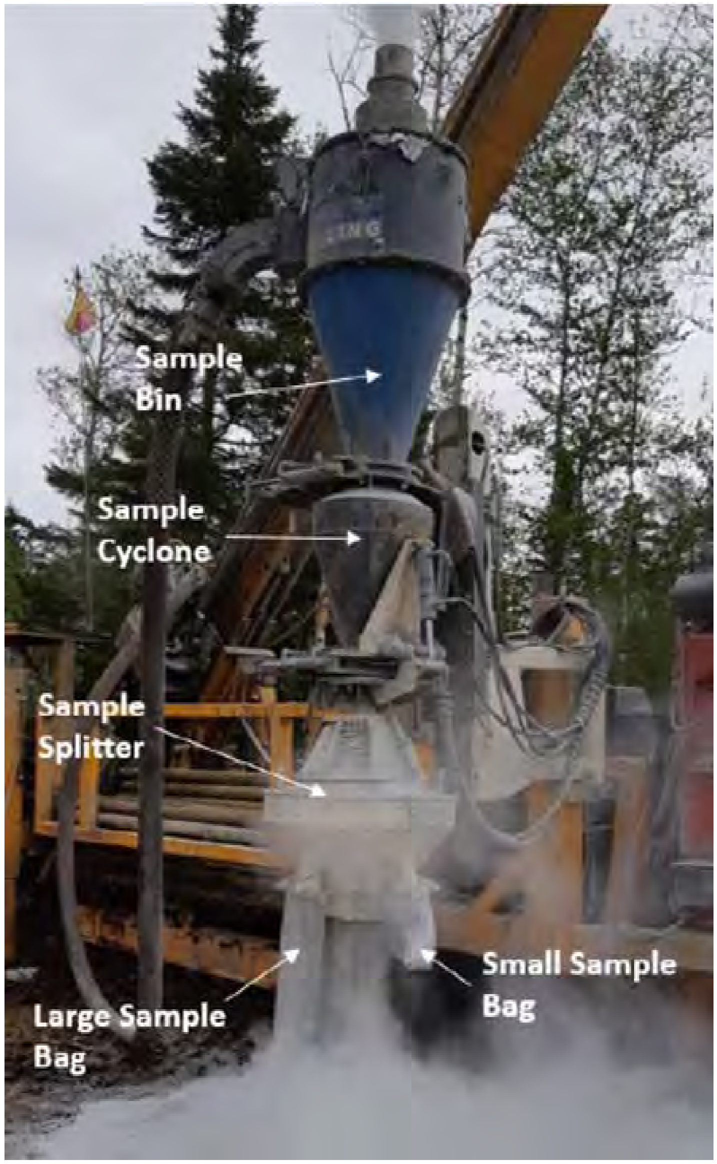
11.1.2.1 RCGC Sampling

GGM samplers assisted NPLH drillers in sampling the reverse circulation ("RC") drill material. The drill rig (Figure 11.2) drilled through overburden and recorded the depth at which rock was intersected. The drill was equipped with an on-board cone splitter that provided two simultaneous samples; a 4 kg sample for analysis, and a 10 kg metallurgical sample. The sampling interval was 2 m. The geotechnicians attached a large and small sample bag onto the sample splitter as shown in Figure 11.2. The sample meterage was recorded, and a sample ticket was placed into each of the sample bags.

Figure 11.1: Principal Sample (small) and Metallurgical Sample (large) from a 2 m Interval - RCGC Sample

A quality control sample was inserted into the sampling stream approximately every 10 samples and alternated between a standard, blank and duplicate. Once the samples were collected, the 4 kg principal sample was sent to Activation Laboratories, based in Geraldton, Ontario and AGAT Laboratories in Thunder Bay, Ontario. Samples were tested for 50 g fire assay (excluding QA/QC and rerun by gravimetric analytical method if the sample ran over 5 g/t). Approximately 10-15% of these samples were chosen for additional ICP-MS analyses for other elements at Activation Laboratories in Thunder Bay, Ontario.

Figure 11.2: Hardab 7000 Maxidrill RC Drill Rig and Splitter in Operation



11.1.2.2 DDH Sampling

Core sample intervals are typically 1.5 m in length but may be as small as 0.5 m if warranted by the occurrence of veining, intense sulfide mineralization or the presence of visible gold. Certified Reference Materials ("CRMs" or standards) and blanks are placed into the sample stream every tenth sample. These QC samples consist of crushed garden stone as a blank material and pre-weighed and packaged CRMs representing low, medium and high grades. The individual cut samples were placed into polyethylene bags, along with the sample tag and sealed. Samples were then placed into rice bags (approximately 8 to 10 samples per bag) and taken to Actlabs Geraldton. Drill core, RC samples, assay pulps and rejects are stored at the Magnet Mine Site.

Figure 11.3: Drill Core – Sawing Shack

11.1.2.3 Quality Control ("QC") Sample Preparation by GGM

All QA/QC ("QC") samples are prepared and bagged in advance by GGM personnel. The GGM employee in the core cutting facilities places one half of the ticket into a bag with the sample and staples the other half in the box. One half of each quality control sample ticket is placed in the appropriate type of control sample bag, which was prepared beforehand. A list of quality control samples and their numbers/locations is posted on the wall in the core logging facility ("core shack") and regularly updated by GGM personnel. Five to seven samples are placed in a rice bag and the contents identified on the outside of the bag. Each bag and its contents are recorded on a notepad and placed in a plastic holder once complete. These slips are picked up each morning by a GGM employee and recorded in an Excel spreadsheet. Once the batches are complete, GGM personnel deliver the bags to Actlabs Geraldton and no third party is involved in transportation.

Samples selected for analysis are sent in batches of 34 samples. Each purchase order covers one batch of 34 samples consisting of:

- 30 regular samples;
- 1 field duplicate sample;
- 1 field blank;
- 1 certified reference material (“CRM” or standard) with a low gold value;
- 1 certified reference material (“CRM” or standard) with a high gold value.

As a quality control check, Actlabs Geraldton adds a 35th sample to every field batch received in the form of a coarse duplicate of the last regular sample (the 30th sample), constituting a second pulp prepared from the reject. The quality of the reject is monitored to ensure that proper preparation procedures are used during crushing. For the fusion process, Actlabs Geraldton adds seven additional quality control samples (two analytical blanks, two certified reference materials and three pulp duplicates), bringing the fusible batch to a total of 42. The pulp duplicates are necessary to ensure that proper preparation procedures are used during pulverization.

At Actlabs Geraldton, the maximum furnace charge of 42 samples ensures that GGM samples are not mixed with others.

11.1.3 Assay Procedure - Sample Preparation and Analysis

11.1.3.1 Fire Assay Sample Preparation (Actlabs Geraldton)

Samples are received at Actlabs Geraldton, sorted and bar-coded. They are then placed in the sample drying room and dried at 60°C. Any samples that are damaged upon receipt (i.e. punctured sample bag, loose core) are documented and the client is informed with pictures.

Samples are crushed to 90% passing 10 mesh and split with a Jones riffle, and a 250 g split is pulverized to 95% passing 150 mesh. Sieve tests are performed on the crusher at the beginning of each day. Sieve tests are performed on the pulps on the first and fiftieth sample of each work order. If there is a failure, the samples are re-milled to ensure that they pass. There is a pulp duplicate made every 30th sample in sample prep and a coarse reject duplicate every 50th. Samples are then sent for fire assay.

11.1.3.2 Metallic Sieve Sample Preparation (Actlabs Geraldton)

All samples containing visible gold are prepared with metallic sieve sample preparation procedures.

A representative 2,000 g split (Code 1A4-2000) is sieved at 100 mesh (149 microns) with fire assays performed on the entire +100 mesh and two splits on the -100 mesh fraction. The total amount of sample and the +100 mesh and -100 mesh fractions are weighed for assay reconciliation. Measured amounts of cleaner sand are used between samples and saved to test for possible plating out of gold on the mill. Alternative sieving mesh sizes are available, however, the finer the grind the more likelihood of gold loss by plating out on the mill.

11.1.3.3 Fire Assay Procedures (Actlabs Geraldton)

The following description for the fire assay procedures was supplied by Actlabs Geraldton. Samples (50 g each) are sent to the fire assay area numbered and in order (usually 1 to 34+1). A rack of 42 crucibles is then labelled with an assigned letter code and numbered one to 42. The mixture is placed in a fire clay crucible. The mixture is then preheated to 850°C, intermediate at 950°C and finished at 1,060°C, with the entire fusion process lasting sixty minutes. The crucibles are then removed from the assay furnace and the molten slag (lighter material) is carefully poured from the crucible into a mould, leaving a lead button at the base of the mould. The lead button is then placed in a preheated cupel which absorbs the lead when cupelled at 950°C to recover the Ag (doré bead) + Au. The entire Ag doré bead is dissolved in aqua regia and the gold content is determined by AA (“Atomic Absorption”) finish (1A2-50 code).

On each tray of 42 samples there are two blanks, three sample duplicates and two certified reference materials, one high and one low (QC = 7 out of 42 samples).

All samples assaying grades over 5.0 g Au/t with AA were re-run with gravimetric finish to ensure accurate values. After the fire assay procedures, Au is separated from the Ag in the doré bead by parting with nitric acid. The resulting gold flake is annealed using a torch. The gold flake remaining is weighed gravimetrically on a microbalance.

11.1.3.4 Fire Assay Procedures with Gravimetric or Atomic Absorption Finish (ALS-Chemex Thunder Bay)

The fire assay technique uses high temperature and flux to “melt” the rock and allows the gold to be collected. Lead formed from the reduction of litharge is traditionally used as the collecting medium for silver and gold. The test sample is intimately mixed with a suitable flux that will fuse at high temperature with the gangue minerals present in the sample to produce a slag that is liquid at the fusion temperature. The liberated precious metals are scavenged by the molten lead and gravitate to the bottom of the fusion crucible.

Upon cooling, the lead button is separated from the slag and processed in a separate furnace for a high temperature oxidation (cupellation) where the lead is removed, leaving the precious metals behind as a metallic bead called a prill. Traditionally, this prill was then partially dissolved in nitric acid (parted) to remove silver and the remaining gold determined by weighing (gravimetry). Alternatively, the prill can be dissolved in a mixture of hydrochloric and nitric acid (aqua regia) and the concentration determined by spectroscopic methods (AAS, ICPAES or ICPMS).such as atomic absorption spectroscopy ("AAS"), inductively coupled plasma atomic emission spectroscopy ("ICPAES") or inductively coupled plasma mass spectroscopy ("ICPMS"). The concentration is normally expressed as parts per million ("ppm"), which is equivalent to grams per tonne ("g/t").

For the AA finish method, a pulp sample is fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents as required, inquarted with 6 mg of gold-free silver and then cupelled to yield a precious metal bead. The bead is digested in 0.5 ml dilute nitric acid in the microwave oven. The 0.5 ml concentrated hydrochloric acid is then added, and the bead is further digested in the microwave at a lower power setting. The digested solution is cooled, diluted to a total volume of 4 ml with de-mineralized water, and analyzed by AAS against matrix-matched standards.

For the gravimetric finish method, a pulp sample is fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents in order to produce a lead button. The lead button containing the precious metals is cupelled to remove the lead. The remaining gold and silver bead is parted in dilute nitric acid, annealed and weighed as gold. Silver, if requested, is then determined by the difference in weights.

At the ALS-Chemex laboratory, the batch size for all fire assay method is 84 including six internal QC. Therefore 78 client samples can be done per batch.

The maximum furnace charge of 78 client samples ensures that GGM samples are not mixed with others.

11.1.4 Quality Control Results – 2012 to 2016

Information in this section is sourced from the NI 43-101 Technical Report prepared by GMS on December 22, 2016.

Table 11.1, Table 11.2 and Table 11.3 were extracted from previous technical reports on the Project prior to the 2018 and 2019 drilling programs and summarize the CRM results from 2013 to 2015. GMS did not identify any flaws in the QA/QC results.

Table 11.1: Results for Standards used by Premier during the 2012-2013 Drilling Program on the Hardrock Deposit

Standard (CRM)	Standard Supplier	Certified gold value (g/t)	Laboratory	Analytical method	Amount of results	Lower process limit (±10%)	Upper process limit (±10%)	Outliers	(%) passing quality control
CDN-GS-5F	CDN Resource Laboratories LTD	5.300	Actlabs Geraldton	FA/AA	228	4.770	5.830	11	95.2%
CDN-GS-5K	CDN Resource Laboratories LTD	3.840	Actlabs Geraldton	FA/AA	376	3.456	4.224	27	92.8%
CDN-GS-7A	CDN Resource Laboratories LTD	7.200	Actlabs Geraldton	FA/AA	2	6.480	7.920	2	0.0%
CDN-GS-7B	CDN Resource Laboratories LTD	6.420	Actlabs Geraldton	FA/AA	583	5.778	7.062	40	93.1%
CDN-GS-8A	CDN Resource Laboratories LTD	8.250	Actlabs Geraldton	FA/AA	201	7.425	9.075	16	92.0%
SF67	Rocklabs Ltd	0.835	Actlabs Geraldton	FA/AA	227	0.752	0.919	18	92.1%
SG40	Rocklabs Ltd	0.976	Actlabs Geraldton	FA/AA	227	0.878	1.074	5	97.8%
SJ53	Rocklabs Ltd	2.637	Actlabs Geraldton	FA/AA	131	2.373	2.901	5	96.2%
SN60	Rocklabs Ltd	8.595	Actlabs Geraldton	FA/AA	204	7.736	9.455	15	92.6%
TOTAL					**2179**			**139**	**93.6%**

Source: Innovexplo, 2013

Table 11.2: Results for Standards used by Premier during the Drilling Program on the Hardrock Deposit from August 12, 2013 to December 31, 2013

Hardrock Project (From August 12, 2013 to December 31, 2013)								
Standard (CRM)	Standard Supplier	Certified gold value (g/t)	Analytical method	Amount of results	Lower process limit (-2SD)	Upper process limit (+2SD)	Outliers	(%) passing quality control
CDN_GS_5K	CDN Resources Laboratories LTD	3.85	FA/GRAV	1191	3.33	4.37	11	99.08%
CDN_GS_6C	CDN Resources Laboratories LTD	5.95	FA/GRAV	477	4.99	6.91	12	97.48%
CDN_GS_7B	CDN Resources Laboratories LTD	6.37	FA/GRAV	555	5.43	7.31	22	96.04%
CDN_GS_8A	CDN Resources Laboratories LTD	8.25	FA/GRAV	3	7.05	9.45	0	100.00%
SF67	Rocklabs Ltd	0.835	FA/GRAV	256	0.793	0.877	85	66.80%
SN60	Rocklabs Ltd	8.318	FA/GRAV	249	7.694	8.942	16	93.57%
TOTAL				2731			146	94.65%

Source: Innovexplo, 2015

Table 11.3: Results for Standards used by Premier during the Drilling Program on the Hardrock Deposit from January 2, 2014 to May 26, 2014

Hardrock Project (From January 2, 2014 to May 26, 2014)								
Standard (CRM)	Standard Supplier	Certified gold value (g/t)	Analytical method	Amount of results	Lower process limit (-2SD)	Upper process limit (+2SD)	Outliers	(%) passing quality control
CDN_GS_5K	CDN Resources Laboratories LTD	3.85	FA/GRAV	207	3.33	4.37	3	98.55%
CDN_GS_5K	CDN Resources Laboratories LTD	3.85	FA/AA	160	3.33	4.37	2	98.75%
CDN_GS_6C	CDN Resources Laboratories LTD	5.95	FA/GRAV	114	4.99	6.91	4	96.49%
CDN_GS_6C	CDN Resources Laboratories LTD	3.85	FA/AA	26	4.99	6.91	0	100.00%
CDN_GS_7B	CDN Resources Laboratories LTD	6.37	FA/GRAV	111	5.43	7.31	6	94.59%
CDN_GS_7B	CDN Resources Laboratories LTD	6.37	FA/AA	53	5.43	7.31	0	100.00%
SF67	Rocklabs Ltd	0.835	FA/GRAV	20	0.793	0.877	1	95.00%
SN60	Rocklabs Ltd	8.318	FA/AA	66	7.694	8.942	9	86.36%
TOTAL				757			25	96.70%

Source: Innovexplo, 2015

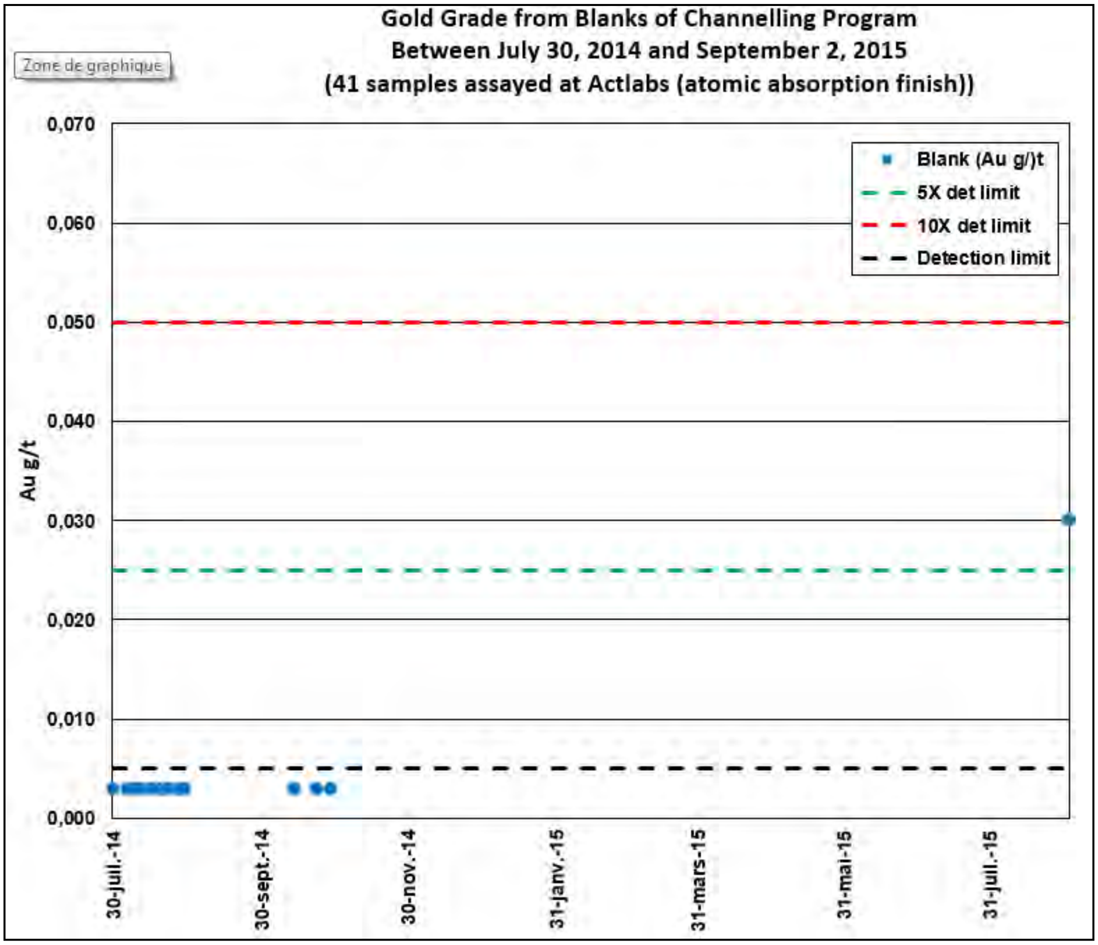
11.1.4.1 <u>Blanks</u>

The field blank used in the drilling program is from a barren sample of crushed white marble. One field blank is inserted for every 34 samples.

According to GGM's QA/QC protocol, if any blank yields a gold value above 0.05 g Au/t (10x detection limit for AA finish), the batch containing the blank should be re-assayed.

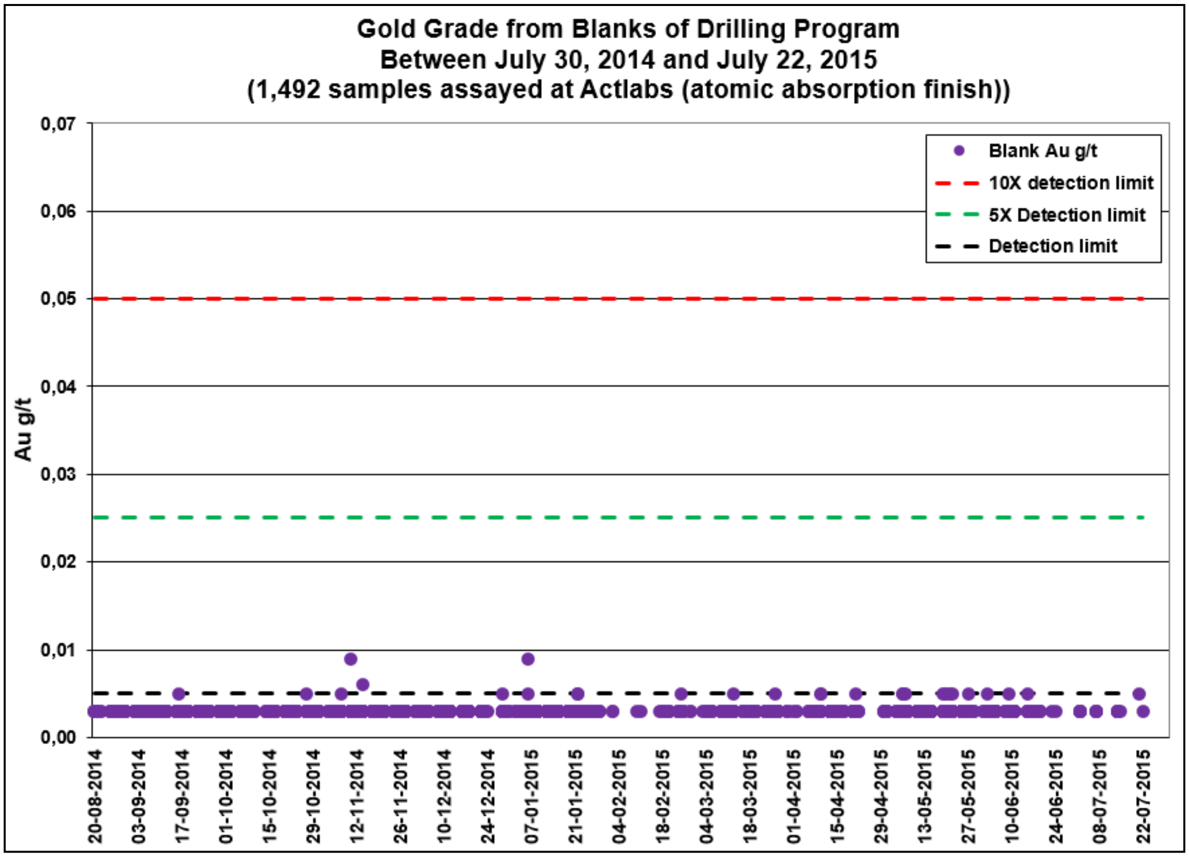
For the channelling program that ran from July 30, 2014 to September 2, 2015 on the Hardrock Deposit, none of the 41 blank results (10x detection limit for AA finish) yielded a gold value above 0.05 g Au/t (Figure 11.4).

Figure 11.4: Results of Blank Samples used for Quality Control during Channelling Program Hardrock Deposit between July 30, 2014 and September 2, 2015. Detection Limit = 0.005 g Au/t for AA Finish



For the drilling program that ran from July 30, 2014 to July 22, 2015 on the Hardrock Deposit, none of the 1,492 blank results (10x detection limit for AA finish) yielded a gold value above 0.05 g Au/t (Figure 11.5).

**Figure 11.5: Results of Blank Samples used for Quality Control during Drilling Program on the Hardrock Deposit between July 30, 2014 and July 22, 2015.
Detection Limit = 0.005 g Au/t for AA Finish**



11.1.4.2 <u>Certified Reference Material (Standards)</u>

Two certified reference materials (“CRMs” or standards) were inserted for every 34 samples during the channelling and drilling programs. Nine standards were used, with gold grades ranging from 0.417 to 8.595 g Au/t as follows:

- CDN-GS-P4B with a theoretical value of 0.417 ± 0.023 g Au/t;
- CDN-GS-P7J with a theoretical value of 0.722 ± 0.036 g Au/t;
- CDN-GS-1L with a theoretical value of 1.160 ± 0.050 g Au/t;
- CDN-GS-2P with a theoretical value of 1.990 ± 0.075 g Au/t;
- CDN-GS-5K with a theoretical value of 3.840 ± 0.140 g Au/t;
- CDN-GS-6C with a theoretical value of 6.030 ± 0.280 g Au/t;
- CDN-GS-7B with a theoretical value of 6.420 ± 0.230 g Au/t;

- SF67 with a theoretical value of 0.835 ± 0.021 g Au/t;
- SN60 with a theoretical value of 8.595 ± 0.223 g Au/t.

GGM quality control protocol stipulates that if any analyzed standard yields a gold value above or below three standard deviations ("3SD") of the certified grade for that standard, then the Project Manager is informed and must decide whether the batch containing that standard should be re-analyzed. All re-analyzed batches (pulps) were sent to ALS-Chemex in Thunder Bay.

The results of all standards used in the Hardrock channelling program carried out from July 30, 2014 to September 2, 2015 are summarized in Table 11.4, and those used in the drilling program from July 30, 2014 to July 22, 2015 are summarized in Table 11.5.

Overall, more than 97.50% of the available results for standards passed the quality control criteria for the channelling program, while more than 97.55% passed for the drilling program.

GMS is of the opinion that all results of the standards are reliable and valid.

Table 11.4: Results for Standards used by GGM during Channelling Program on Hardrock Deposit July 20, 2014 – September 2, 2015

Hardrock Project (From July 30, 2014 to September 2, 2015)								
Standard (CRM)	Standard Supplier	Certified gold value (g/t)	Analytical method	Amount of results	Lower process limit (-3SD)	Upper process limit (+3SD)	Outliers	(%) passing quality control
CDN_GS_2P	CDN Resources Laboratories LTD	1.99	FA/AA	2	1.765	2.22	0	100.00%
CDN_GS_5K	CDN Resources Laboratories LTD	3.84	FA/AA	39	3.46	4.24	2	94.87%
CDN_GS_6C	CDN Resources Laboratories LTD	6.03	FA/AA	40	5.31	6.75	0	100.00%
CDN_GS_7B	CDN Resources Laboratories LTD	6.42	FA/AA	1	5.73	7.11	0	100.00%
TOTAL				**80**			**2**	**97.50**

Table 11.5: Results for Standards used by Premier during the Drilling Program on Hardrock Deposit from July 30, 2014 to July 22, 2015

Hardrock Project (From July 30, 2014 to July 22, 2015)								
Standard (CRM)	Standard Supplier	Certified gold value (g/t)	Analytical method	Amount of results	Lower process limit (-3SD)	Upper process limit (+3SD)	Outliers	(%) passing quality control
CDN_GS_P4B	CDN Resources Laboratories LTD	0.417	FA/AA	474	0.348	0.486	21	95.57%
CDN_GS_P7J	CDN Resources Laboratories LTD	0.722	FA/AA	70	1.01	1.31	2	97.14%
CDN_GS_1L	CDN Resources Laboratories LTD	1.16	FA/AA	71	1.01	1.31	1	98.59%
CDN_GS_2P	CDN Resources Laboratories LTD	1.99	FA/AA	114	1.77	2.22	3	97.37%
CDN_GS_5K	CDN Resources Laboratories LTD	3.84	FA/AA	804	3.46	4.24	18	97.76%
CDN_GS_6C	CDN Resources Laboratories LTD	6.03	FA/AA	589	5.47	6.59	12	97.96%
CDN_GS_7B	CDN Resources Laboratories LTD	6.42	FA/AA	531	5.72	7.12	8	98.48%
SF67	Rocklabs Ltd	0.835	FA/AA	177	0.772	0.898	1	99.44%
SN60	Rocklabs Ltd	8.595	FA/AA	145	7.926	9.264	7	95.17%
TOTAL				2975			73	97.55%

11.1.4.3 Coarse Reject Duplicates

The quality control protocol between 2012 and 2016 required that a coarse duplicate be prepared for the 30th sample in each batch. The duplicate was prepared by taking half of the crushed material derived from the original sample. By measuring the precision of the coarse duplicates, the incremental loss of precision can be determined for the coarse-crush stage of the process, thus indicating whether two sub-samples taken after primary crushing is adequate for the given crushed particle size to ensure a representative sub-split.

Duplicates are used to check the representativeness of results obtained for a given population. To determine reproducibility, precision (as a percentage) is calculated according to the following formula:

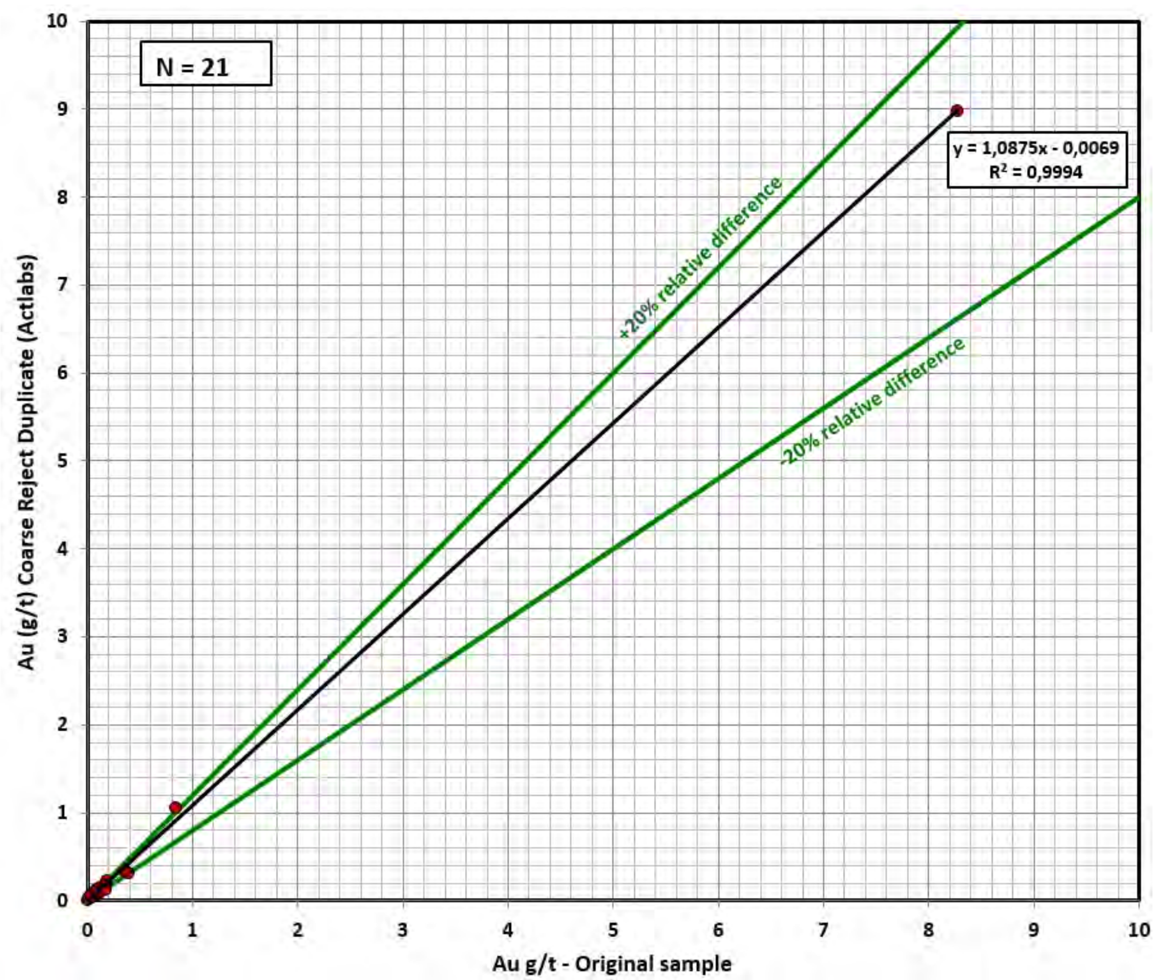
$$\text{Precision (\%)} = \frac{\text{(Duplicate Sample Gold Grade – Original Sample Gold Grade)}}{\text{Average Between Duplicate Sample Gold Grade and Original Sample Gold Grade}} \times 100$$

Precision ranges from 0 to 200% with the best being 0%, meaning that both the original and the duplicate sample returned the same grade.

A total of 21 original-coarse crush duplicate pairs (channelling) were identified in the database corresponding to the period between July 30, 2014 and September 2, 2015. Figure 11.6 shows a linear regression slope of 1.0875 and a correlation coefficient of 99.9%.

The correlation coefficient (%) is given by the square root of R^2 and represents the degree scatter of data around the linear regression slope. The results obtained indicate an excellent reproducibility of gold values with a gravimetric finish at Actlabs Geraldton. For gold values greater than 1 g Au/t, no outlier is observed on the graph because no duplicate pair is outside the lines marking a ±20% relative difference.

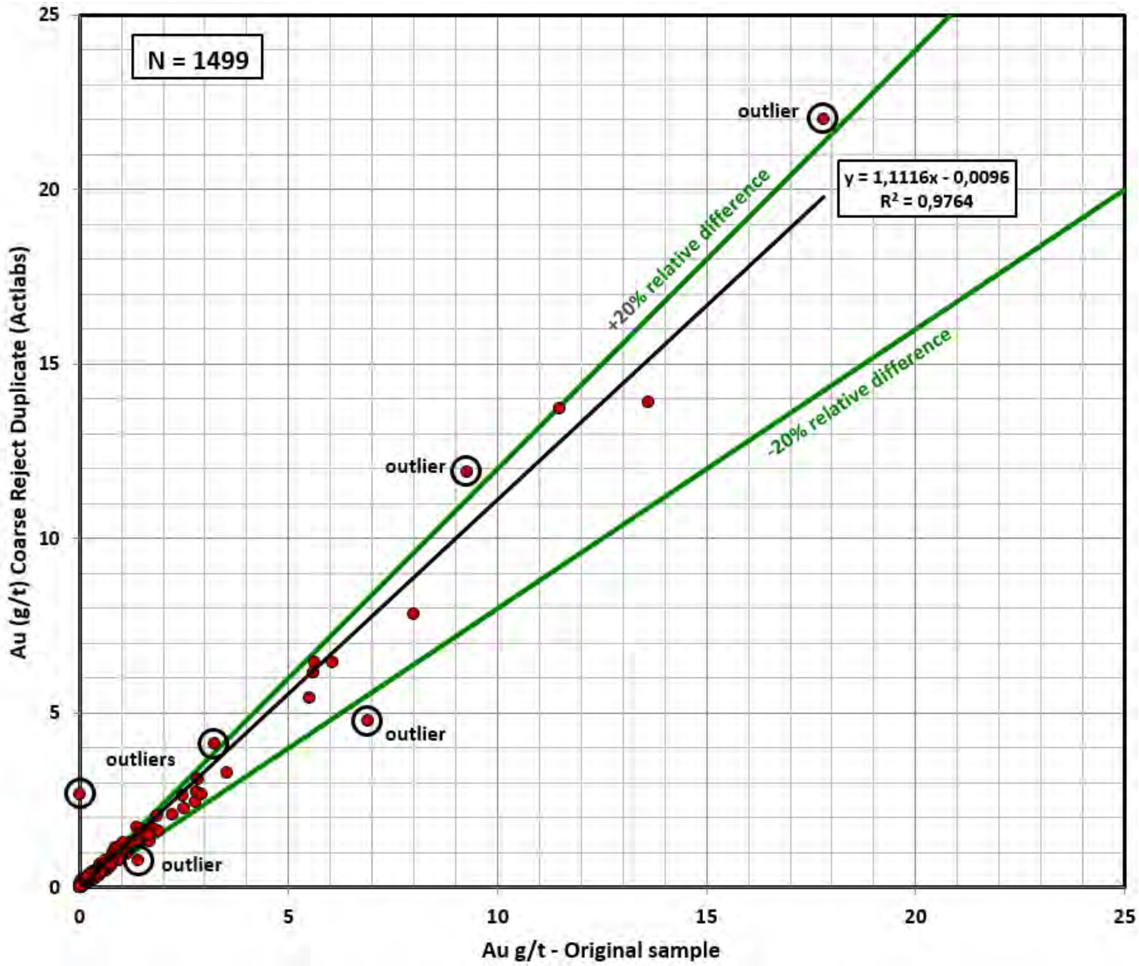
Figure 11.6: Linear Graph Comparing Original Samples and Crush Coarse Duplicate Samples (duplicate pairs) between July 30, 2014 and September 2, 2015 (channelling)



A total of 1,499 coarse duplicate pairs from drilling were identified in the database corresponding to the period between July 30, 2014 and July 22, 2015. Figure 11.7 shows a linear regression slope of 1.1116 and a correlation coefficient of 98.8%. The results obtained indicate an excellent reproducibility of gold values with AA finish at Actlabs Geraldton. For gold values greater than 1 g Au/t, only six outliers are observed on the graph because these coarse duplicate pairs are outside the lines marking a ±20% relative difference.

GMS is of the opinion that the results obtained for the Hardrock coarse duplicates between 2012 and 2016 are reliable and valid.

Figure 11.7: Linear Graph Comparing Original Samples and Crush Coarse Duplicate Samples (duplicate pairs) for the Period between July 30, 2014 and July 22, 2015 (drilling)



11.1.5 Quality Control Results – 2018 and 2019 Drilling Programs

The results of the 2018-2019 QA/QC program (May 2018 through May 2019) were provided by GGM staff and reviewed by GMS.

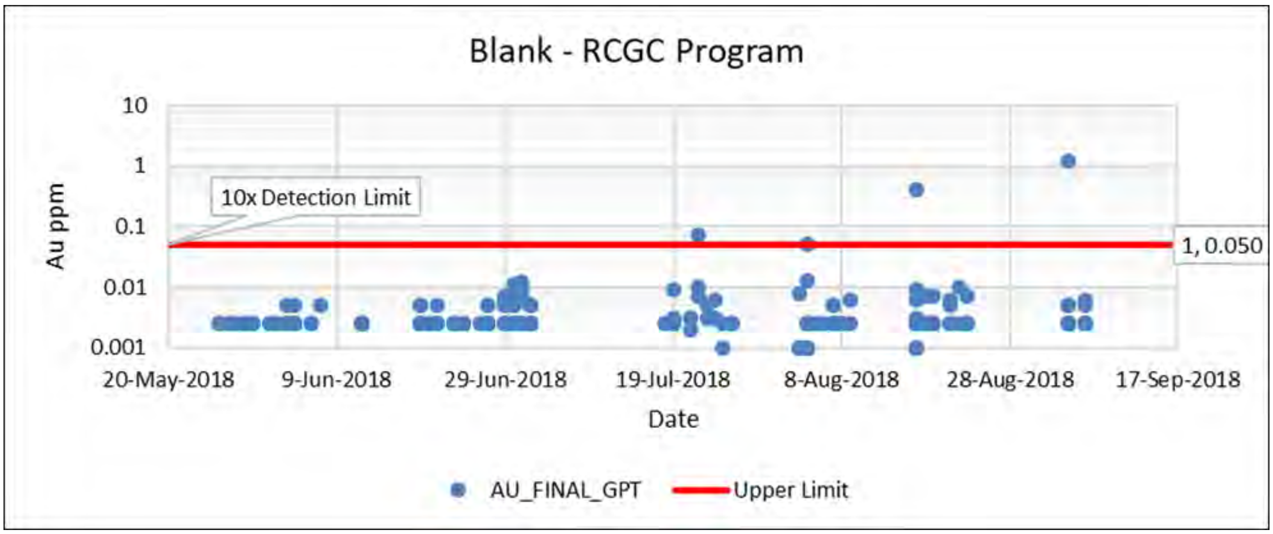
In addition to the Actlabs internal QC protocol, GGM implemented a rigorous QA/QC program for its drill core sampling completed in 2018 and 2019. As part of the QA/QC procedure, blanks, CRMs and various duplicates were inserted into the sample stream at a rate of one for every 10 samples.

11.1.5.1 Blanks

GGM's protocol is to insert one blank into the sampling stream every 50 samples. The field blank used in the RCGC and DDH drill programs was from a barren sample of crushed white gardening stone. GGM's QA/QC protocol stipulates that if any blank yields a gold value above 0.05 g Au/t (10X detection limit "DL" for gravimetric finish), the blank is rerun as well as 10 samples before and after the failed QC sample. All reruns are undertaken by Actlabs Geraldton, Ontario.

For the 2018 RCGC drilling program (May 1 to October 10, 2018) on the Hardrock Deposit, four of the 341 blank results exceeded this recommended 10X DL value, representing 0.9% of the total blank population (Figure 11.8). Samples were re-run and the results were acceptable.

Figure 11.8: QA/QC - 2018 Blank Results of RCGC Samples



Results for the 2019 RCGC and DDH drill programs show that there are zero blank QC failures (see Figure 11.9 and Figure 11.10). Peaks in the data are insignificant and may be attributed to improper cleaning of apparatus at Actlabs.

Figure 11.9: QA/QC - 2019 Blank Results of RCGC Assays

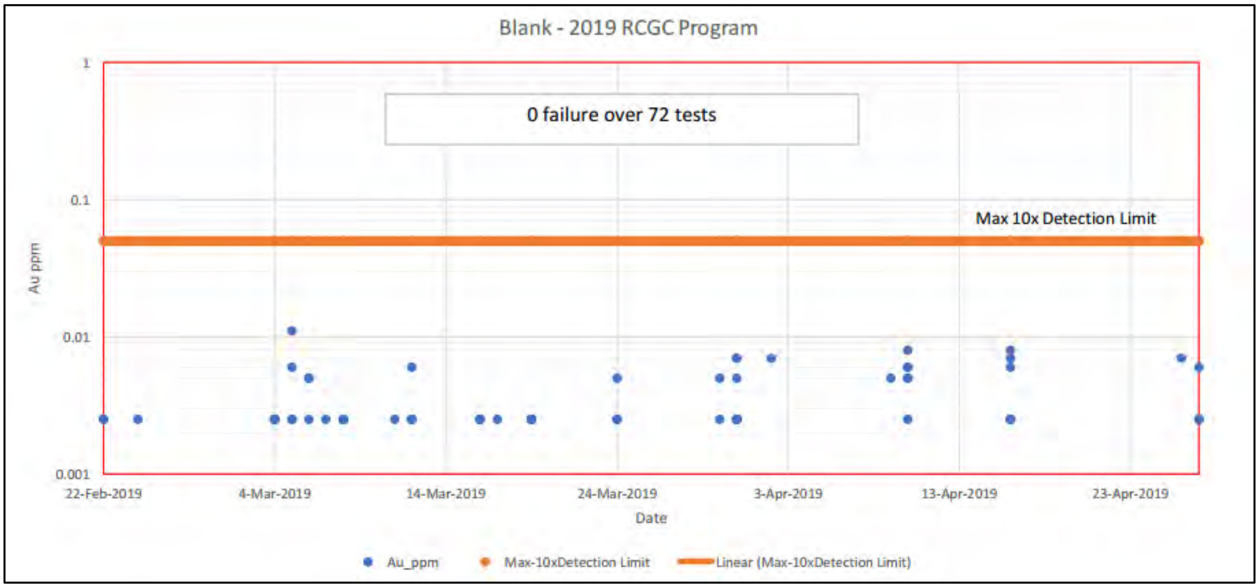
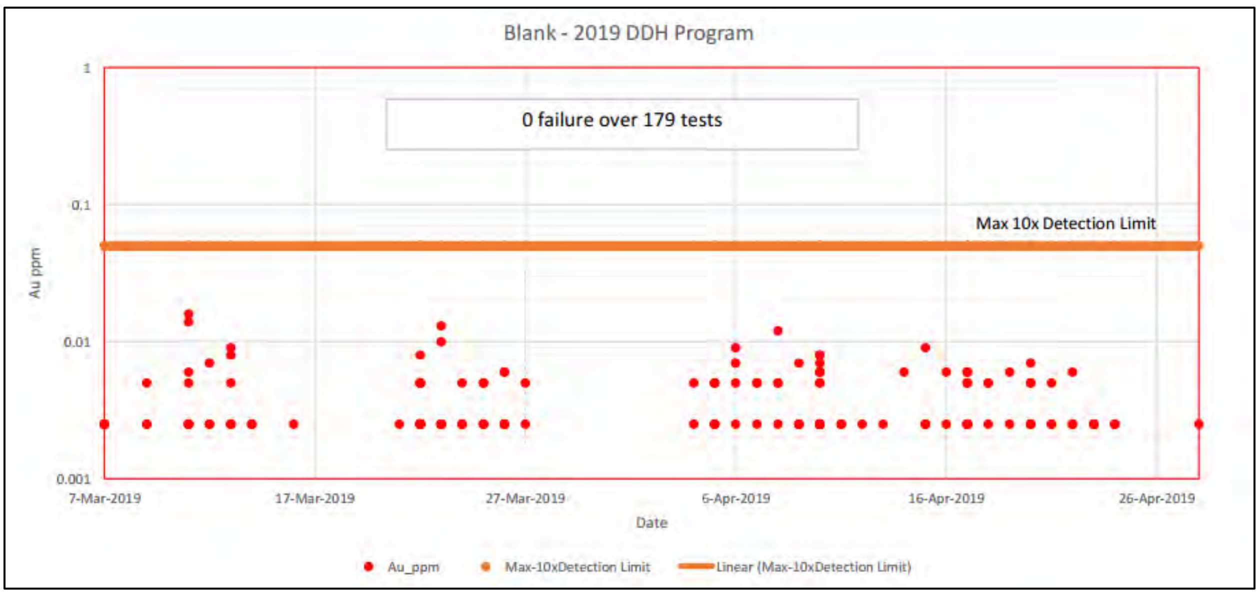


Figure 11.10: QA/QC - Blank Results of DDH Assays



11.1.5.2 Certified Reference Material ("CRMs" or Standards)

GGM's protocol is to insert three CRMs into the sampling stream every 50 samples. This alternates between a low-grade standard, middle (ore) grade standard, and a high-grade standard. In total, six CRMs were used to monitor the consistency and accuracy of a laboratory. Two of six CRMS were manufactured by Ore Research & Exploration Pty Ltd ("OREAS"), in Australia. The other four CRMs were produced by CDN Resource Laboratories Ltd. ("CDN Labs"), in Canada. Both OREAS and CDN standards are certified in accordance with International Standards Organization ("ISO") recommendations. The Performance Gates applied for the Hardrock Project are available on the ORE Research & Exploration Pty Ltd. and CDN Resource Laboratories Ltd. website respectively (https://www.ore.com.au/oreas-reports/ and http://www.cdnlabs.com/Cu-Au-standards.htm).

The standards were inserted by GGM, with gold grades ranging from 0.468 to 5.95 g Au/t as follows:

- CDN_GS_1P5R with a certified value of 1.81 ± 0.14 g Au/t;
- CDN_GS_5J with a certified value of 4.90 ± 0.45 g Au/t;
- CDN_GS_6C with a certified value of 5.95 ± 0.480 g Au/t;
- CDN_GS_P4G with a certified value of 0.468 ± 0.052 g Au/t;
- OREAS_2PD with a certified value of 0.885 ± 0.014 g Au/t;
- OREAS_6PC with a certified value of 1.52 ± 0.03 g Au/t.

Internal laboratory standards were also used by Activation Laboratories as below:

- OREAS 218 with a theoretical value of 0.531 ± 0.017 g Au/t;
- OREAS 221 with a theoretical value of 1.06 ± 0.036 g Au/t;
- OREAS 222 with a theoretical value of 1.22 ± 0.033 g Au/t;
- OREAS 224 with a theoretical value of 2.15 ± 0.053 g Au/t;
- OREAS 216 with a theoretical value of 6.66 ± 0.16 g Au/t.

GGM Quality Control protocol stipulates that if any analyzed standard yields a gold value above or below three standard deviations ("3SD") of the certified grade for that standard, then the CRM is rerun as well as 10 samples before and after. The rerun material is analyzed and compared to original sample value. If precision of the new value is less than 40%, the old value is accepted. If the new value is greater than 40%, further follow up is required. Figure 11.11 and Figure 11.12 illustrate one example of CRM CDN-GS-P4G results from the RCGC and DDH sampling program carried out in 2019.

Figure 11.11: Standard CDN-GS-P4G Results – RC Assays

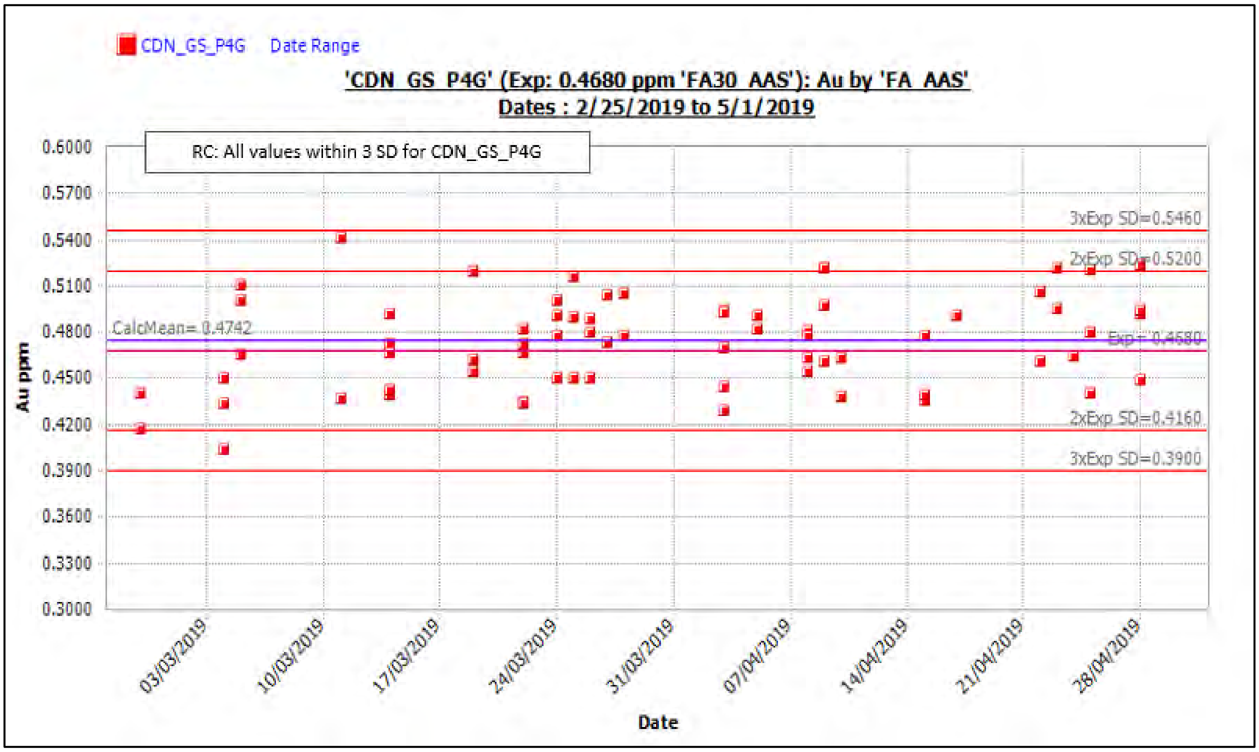
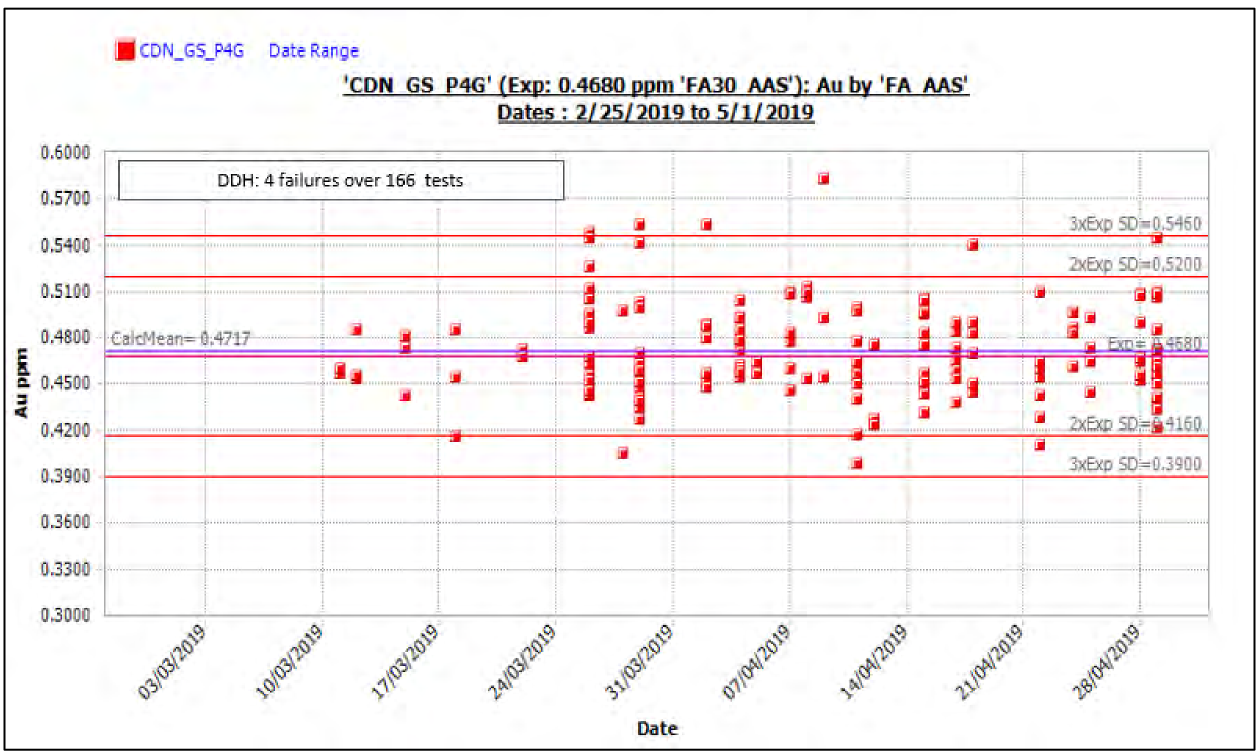


Figure 11.12: Standard CDN-GS-P4G Results – DDH Assays



The 2018 results of all three standards used in the Hardrock RCGC program are summarized in Table 11.6. More than 98.6% of the available results for standards passed the quality control criteria for the RCGC program.

Table 11.6: 2018 Standard Result Summary (excluding internal lab standards) – RCGC Samples

Au (g/t) Standard(s)			Limits		No. of Samples	No. of Failures	% Failure	Calculated Values			
Standard (CRM)	Analytical Method	Certified Value	-3SD	+3SD				Mean Au	SD	CV	(%) Passing QC
CDN_GS_P4B	FA_AAS	0.417	0.348	0.486	130	2	1.54%	0.427	0.055	0.13	98.5%
OREAS_2Pd	FA_AAS	0.89	0.795	0.975	182	1	0.55%	0.830	0.062	0.07	99.5%
CDN_GS_6C	FA_GRAV	5.95	5.23	6.67	123	3	2.44%	5.900	0.280	0.05	97.6%
Total					**435**	**6**					**98.6%**

The 2019 results of all standards used in the Hardrock RCGC program are summarized in Table 11.7 , and those used in the diamond drilling program are summarized in Table 11.8. More than 98.2% of the available results for standards passed the quality control criteria for the RCGC program, while only 96.3% passed for the diamond drilling program.

GMS is of the opinion that all results of the standards are reliable and valid.

Table 11.7: 2019 Standard Result Summary (excluding internal lab standards) – RCGC Samples

Au (g/t) Standard(s)			Limits		No. of Samples	No. of Failures	% Failure	Calculated Values			
Standard (CRM)	Analytical Method	Certified Value	-3SD	+3SD				Mean Au	SD	CV	(%) Passing QC
CDN_GS_P4G	FA_AAS	0.468	0.39	0.546	70	0	0.0%	0.473	0.030	0.06	100.0%
OREAS_2Pd	FA_AAS	0.885	0.795	0.975	1	0	0.0%	0.815	0.000	0.00	100.0%
OREAS_6Pc	FA_AAS	1.52	1.32	1.72	46	2	4.3%	1.460	0.081	0.06	95.7%
CDN_GS_1P5R	FA_AAS	1.81	1.6	2.02	40	1	2.5%	1.740	0.078	0.04	97.5%
CDN_GS_6C	FA_GRAV	5.95	5.23	6.67	10	0	0.0%	5.881	0.134	0.02	100.0%
Total					**167**	**3**					**98.20%**

Table 11.8: Standard Result Summary (excluding internal lab standards) – DDH Samples

Au (g/t) Standard(s)			Limits		No. of Samples	No. of Failures	% Failure	Calculated Values			
Standard (CRM)	Analytical Method	Certified Value	-3SD	+3SD				Mean Au	SD	CV	(%) Passing QC
CDN_GS_P4G	FA_AAS	0.468	0.39	0.546	166	4	2.41%	0.472	0.033	0.07	97.6%
OREAS_6Pc	FA_AAS	1.52	1.325	1.715	62	6	9.68%	1.481	0.087	0.06	90.3%
CDN_GS_1P5R	FA_AAS	1.81	1.6	2.02	109	5	4.59%	1.760	0.086	0.05	95.4%
CDN_GS_5J	FA_AAS	4.90	4.23	5.58	5	0	0.00%	4.860	0.104	0.02	100.0%
CDN_GS_6C	FA_GRAV	5.95	5.23	6.67	91	1	1.10%	5.881	0.134	0.02	98.9%
OREAS_6Pc	FA_GRAV	1.52	1.325	1.715	1	0	0.00%	1.481	0.087	0.06	100.0%
Total					**434**	**16**					**96.3%**

11.1.5.3 RC Field Duplicates and ¼ Core Duplicates

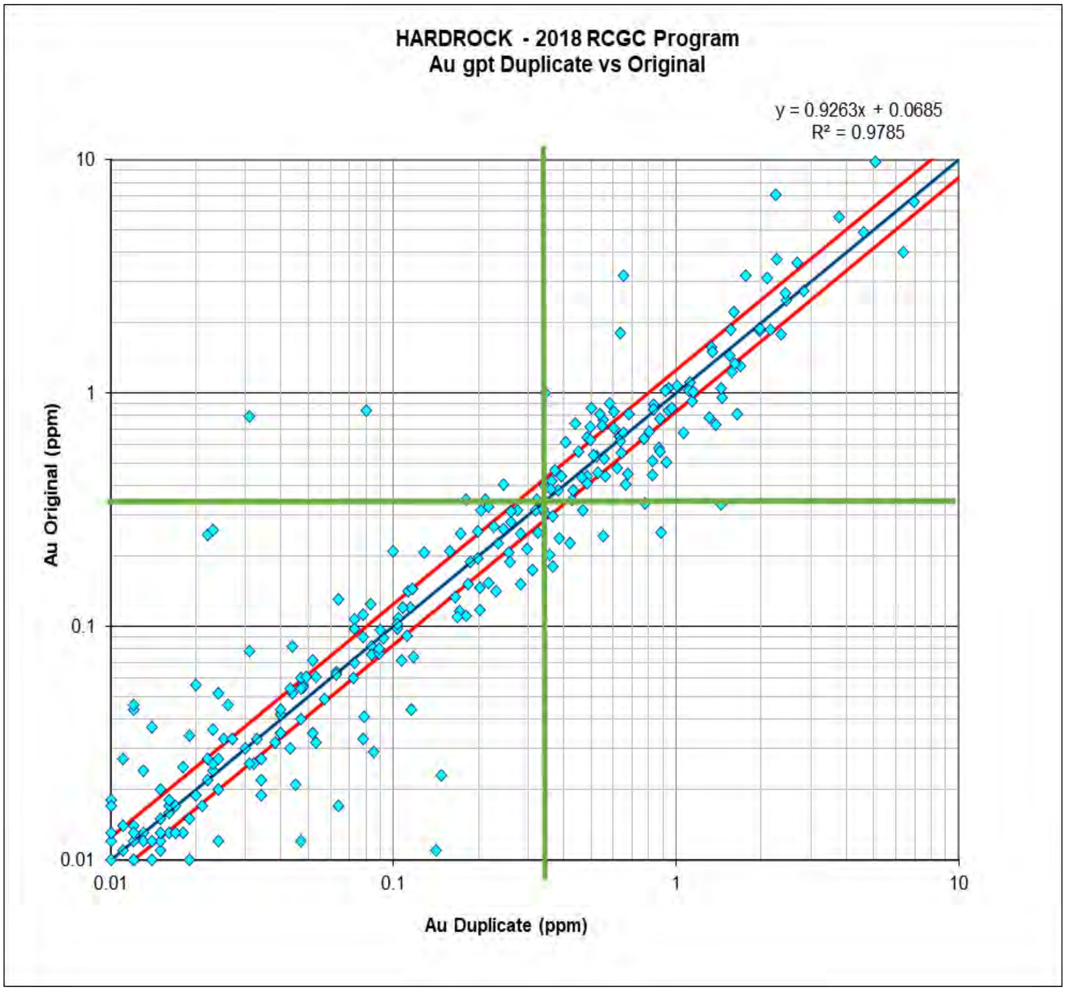
During the 2018 and 2019 RCGC drilling campaigns, field duplicates were taken at the drill rig using the onboard cyclone splitter. A third small sample bag was attached to the splitter, and the duplicate was collected at the same time as the principal sample and the metallurgical sample.

For the 2019 DDH campaign, ¼ core duplicates were collected (not coarse rejects).

The original assays versus duplicate assays for 2018 RCGC program are plotted in Figure 11.13. Duplicate sets are presented as log-scaled plots to provide details at lower concentrations. Results show considerably more scatter at lower (<0.3 g Au/t) gold values. This is not uncommon, because as the Au values approach the limit of detection, determinations become less accurate. For this reason, a precision limit of ±20% where the value was 10 times the detection limit (“10X DL”) was used to determine the precision of the duplicates. Alternatively, precision increases as grade increases. In general, there is good agreement between the original assay and duplicate results. Approximately 4% fall outside of the acceptable ranges.

A total of 290 field duplicates illustrate a linear regression slope of 0.9264 and a correlation coefficient of 97.8%, which means that the average grade is close to the average original grade, and there is a very good reproducibility.

Figure 11.13: 2018 Field Duplicates for Gold Values – RCGC Samples



The original samples and duplicate assays for both 2019 DDH and RCGC programs are plotted in Figure 11.14 and Figure 11.15 respectively. Duplicate sets are presented as log-scaled plots to provide details at lower concentrations.

Figure 11.14: 2019 ¼ Core Duplicates for Gold Values – DDH Samples

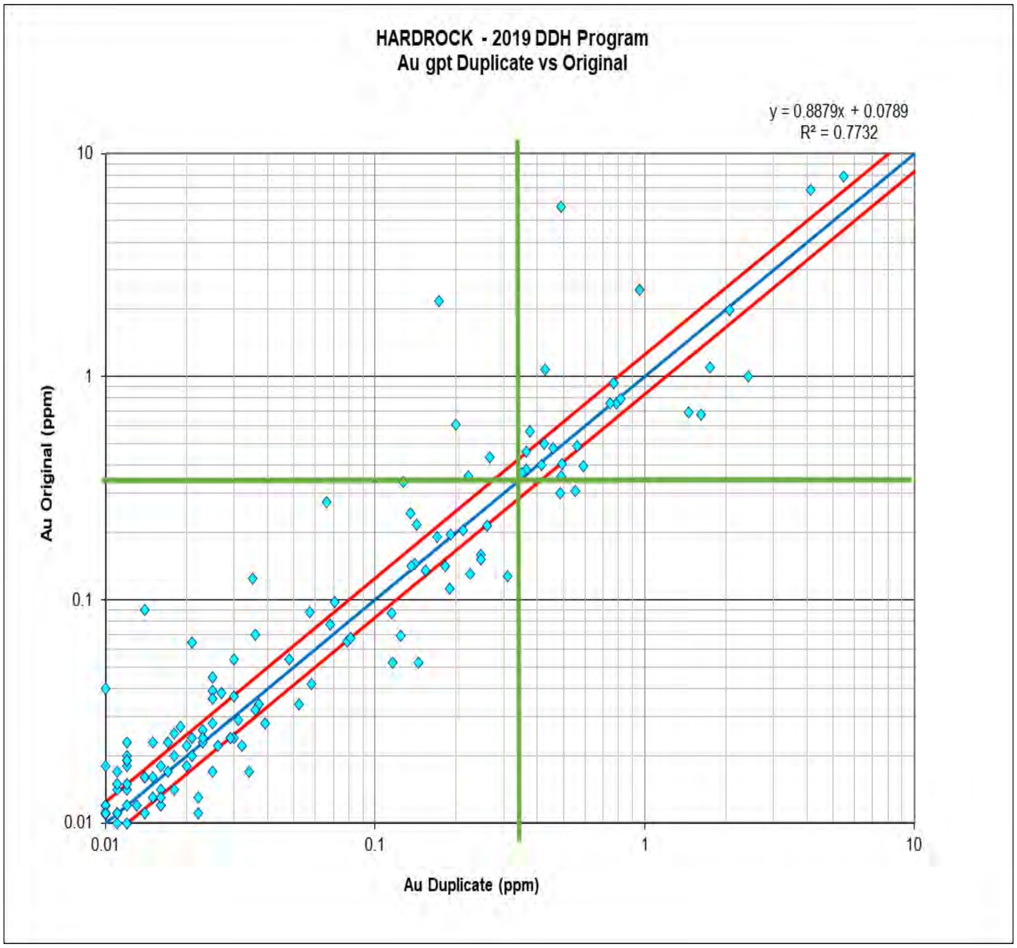
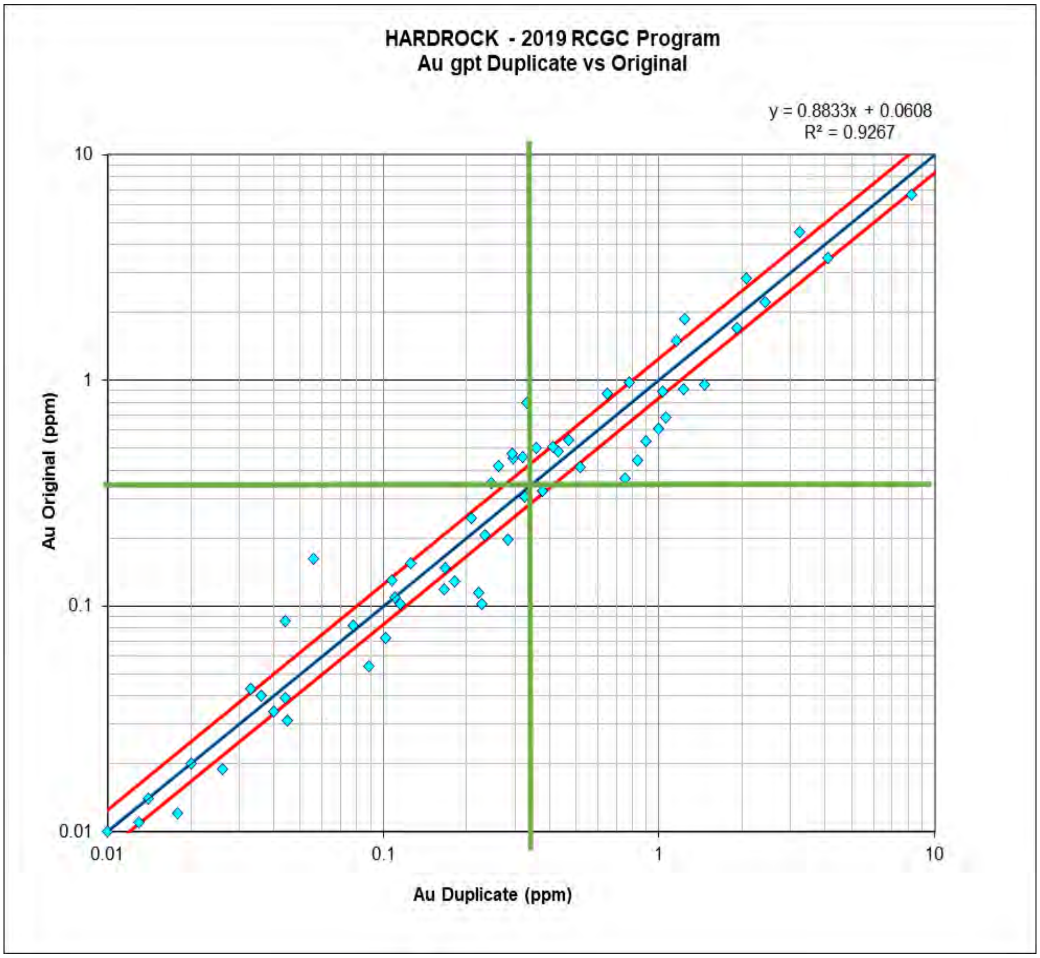


Figure 11.15: 2019 Field Duplicates for Gold Values – RCGC Samples



A total of 172 ¼ core duplicate pairs (DDH samples) were identified in the database corresponding to the period between February 25 and May 1, 2019. A linear regression slope of 0.8879 and a correlation coefficient of 77.3% is observed.

A total of 63 field duplicate pairs (RCGC samples) were identified in the database corresponding to the period between February 25 and May 1, 2019. A regression slope of 0.8833 and a correlation coefficient of 92.7% is observed.

GMS is of the opinion that the results obtained for the Hardrock field and ¼ core duplicates are reliable and valid.

11.1.6 QP Conclusions

A statistical analysis of the QA/QC data provided by GGM did not reveal any significant analytical issues. GMS is of the opinion that the sample preparation, analysis, QA/QC and security protocols used for the Hardrock Project follow generally accepted industry standards and that the data is of sufficient quality to be used for Mineral Resource estimation.

11.2 Brookbank, Kailey and Key Lake Deposits

The following information is based on data provided by GGM and an earlier technical reports prepared by Scott Wilson RPA in 2009 and GMS in 2016.

11.2.1 Historical Sampling Procedures and QA/QC (Pre-Premier Gold Involvement)

11.2.1.1 Brookbank

Prior to Ontex Resources Ltd. (“Ontex”) fully involvement on the property in October 1998, descriptions of sampling and gold assaying methodologies are not available. The majority of analyzes focused on gold, and less often silver, using a fire assay with a gravimetric finish as analytical method. The grade results were expressed in ounces per ton (until the mid-1990s) and in grams per metric tonne afterward.

During 1999 to 2009 drilling campaigns, Ontex mainly used either the Activation (“Actlabs”) Laboratories (Geraldton, Ontario) or Swastika Laboratories Ltd. (Swastika, Ontario) for analysis of drill core samples. Swastika Laboratories Ltd. (“Swastika”) was ISO 9001:2000 registered and accredited by the Standards Council of Canada.

Replicate samples were assayed at Accurassay Laboratories ("Accurassay") in Thunder Bay, Ontario. Accurassay is an accredited Laboratory by the Standards Council of Canada and conforms to requirements of ISO/IEC 17025.

The sample preparation and analysis procedures used by Swastika, which performed all the assaying for Ontex drilling programs, are summarized as below.

11.2.1.1.1 Sample Preparation and Analyses

11.2.1.1.1.1 Swastika Laboratories

Each sample shipment was checked for the count of sample, the condition of the packaging, the integrity of the sample's seals, and the customer's analysis instructions. Any damage, evidence of altering original batches, and/or missing sample containers are noted and immediately reported.

The following information was derived from the RPA 2009 technical report and describes the Ontex sample preparation prior to assaying at Swastika Laboratories:

- Depending on the moisture content of the customer sample, the entire sample was either air-dried or oven-dried in a clean metal pan prior to crushing.
- The entire dried sample was passed through a jaw crusher to arrive at a prepared sample, 80% or more of which is passing through a ten mesh (1,700 microns) screen. The crushed material was split successively in a riffle divider to arrive at a subsample of 300 g to 400 g. The subsample was placed in a labelled envelope for pulverizing.
- The subsample was pulverized in a ring and puck pulverizer to enabling 90% to 95% of the material to pass through a 100 mesh (150 microns) screen. Methyl hydrate was added to the sample prior to pulverizing to prevent clumping.
- The pulverized material from the bowl, ring and puck was carefully brushed onto a rubber mat from which it was poured back into the labelled envelope.

Gold assay procedures were described as follows:

- A one assay 1 kg sample was drawn of pulverized material, weighed and placed into 30 g crucible containing flux. Crucibles were marked with the customer name, sample number and certificate number.
- Depending on rock type, varying amounts of flour, silica and borax were added to ensure a proper fusion from the crucible.
- The crucible containing the sample, flux and other necessary ingredients were fully mixed in a tumbler prior to fusion in the furnace oven.
- The crucible was placed in the fusion oven and heated until a proper fusion was completed, after which it was removed, and the contents were transferred into a metal mould for cooling and solidification.

- The solidified material was hammered to remove the slag and the lead button was placed in a cupel.
- The cupel including the lead button was placed into a furnace until all the lead had been absorbed into the cupel.
- The gold bead was removed from the cupel and placed in a porcelain cup containing parting acid (7:1 concentration of nitric acid and distilled water). The contents were heated in a hot water bath and the solution was thereafter decanted.
- The bead was dried in a hot water bath and a visual assessment was made to proceed with either a gravimetric or an atomic absorption spectrometry analytical method.
- Precious metal beads from the cupel furnace were assayed for gold content using AA spectrometry or gravimetric ("GRAV") techniques.
- In the AA technique, the gold bead was dissolved in five mm of aqua regia in a porcelain cup and then cooled at room temperature. The solution was then analyzed by an AA spectrometer to determine the gold grade results.
- In the gravimetric technique, the gold bead is carefully removed from the porcelain cup and weighed using a microbalance. The gold calculation is based on a sample amount of 29.166 g. Gravimetric method ere normally used when the assay result was over 2 g Au/t.
- All grades obtained are reported in either parts per million or grams per tonne.

The internal Quality Control at Swastika Laboratory was carried-out by using in-house or Canmet certified standards, blanks and by re-assaying at least 10% of all samples. All data are evaluated by supervisor and additional checks were run on presence of anomalous values.

11.2.1.1.1.2 Accurassay

The description below, excerpted from RPA 2009 technical report, discuss the sample preparation and gold analyses performed on drill core samples at Accurassay Laboratories.

All rock samples were entered into Accurassay's Information Management System ("LIMS"). The samples were dried, crushed to approximately eight mesh (2,360 microns) and then a 250 g to 500 g subsample was taken. The subsample was pulverized to 90% passing 150 mesh (106 microns) and then matted to ensure homogeneity. To prevent cross contamination, silica sand was used after each sample was pulverized. The homogeneous sample was then sent for gold analysis with the analytical method required.

Accurassay gold analysis procedures were described as follows:

- For the analysis of precious metals, the sample is mixed with a lead-based flux and fused for one hour and fifteen minutes.
- Each sample has a silver solution added to it prior to fusion, producing a precious metal bead after cupellation.
- The button was placed in a cupelling furnace.
- All the lead was absorbed by the cupel, and a silver bead, which contained any gold was left in the cupel.
- The cupel was removed from the furnace and cooled, the silver bead is placed in an appropriately labelled test tube and digested using aqua regia.
- The samples were diluted with one millilitre of distilled deionized water and one millilitre of 1% digested lanthanum solution. The samples were cooled and mixed to ensure homogeneity of the solution.
- The samples were analyzed for gold using AA spectroscopy.
- The results for the AA technique were checked by the technician and then forwarded to data entry, by means of electronic transfer, and a certificate is produced.
- The Laboratory Manager validated the data and the certificates. The results were sent in the client requested format.
- Accurassay had an internal threshold that automatically sent back samples greater than 30 g Au/t to be verified by re-assay (assayed in triplicate) to ensure reproducibility.
- Ontex samples grading greater than 30 g Au/t would have been verified by Accurassay internally but not reported. Gravimetric analysis is offered only by request.
- Accurassay employed an internal quality control system that tracks certified reference materials and in-house quality assurance standards.
- Accurassay used a combination of reference materials, including reference materials purchased from CANMET, standards created in-house by Accurassay and tested by round-robin with laboratories across Canada, and ISO certified calibration standards purchased from suppliers.

11.2.1.1.1.3 Actlabs

The details of sample preparation and analysis performed for gold at Actlabs in 2009, are similar to the description discussed in Subsections 11.1.3.1, 11.1.3.2 and 11.1.3.3.

11.2.1.1.2 Ontex QA/QC Program

11.2.1.1.2.1 Blanks

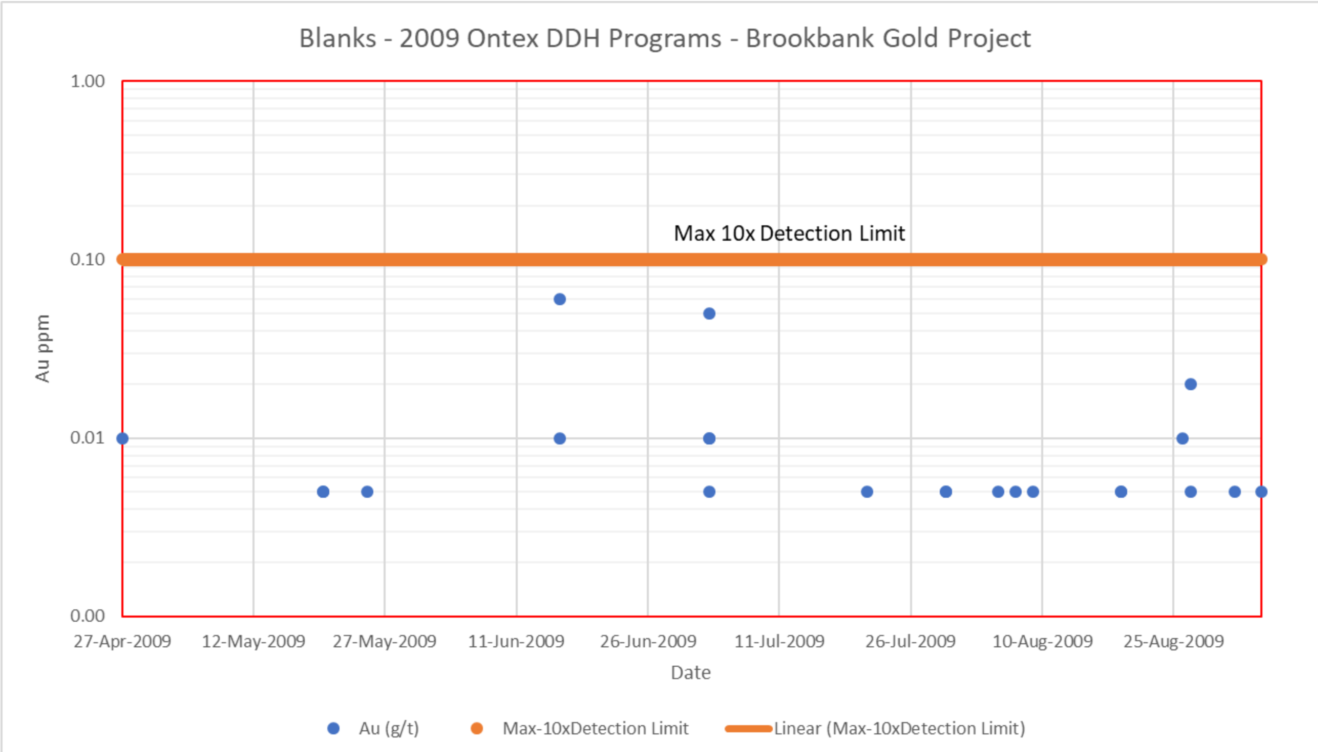
Ontex procedures stipulated that field blanks were inserted immediately after a potential high-grade intercept. The field blanks were sourced from four different locations:

- An aggregate consisting of landscape limestone purchased from a grocery store in Geraldton, Ontario;
- Split core consisting of gabbro from a previous Brookbank drill hole that contains no visible sulphide mineralization;
- Fragments of granite collected from a nearby road cut;
- Certified blanks.

GMS verified the barren material results with the original certificates obtained from GGM exploration team. Not all the original certificates from Swastika are available for validation. Figure 11.16 illustrates some of the blanks results analyzed at Swastika during the Ontex drilling program in 2009.

In total, 23 of 41 blanks were available to be corroborated and verified with the original laboratory certificate and the results show that blanks are below the control limit of 10 times the detection limit. No evidence of contamination was observed between April to August 2009.

Figure 11.16: Blanks at Swastika Laboratories Ltd. Ontex 2009 Drilling Campaign



11.2.1.1.2.2 Certified Standards

Ontex inserted two reference materials into the sample stream at a rate of one for each 20 sample batch submitted at Swastika Laboratory. The certified standard was supplied by ROCKLABS® of Auckland, New Zealand. The assigned value and 95% confidence limits established by the laboratory are presented in Table 11.9.

Table 11.9: ROCKLABS Certified Material used by Ontex between April to August 2009

Au (g/t) Standard(s)			95% Confidence Interval	No. of Samples	No. of Failures	% Failure
CRM Code	CRM Supplier	Certified Value				
SN38	ROCKLABS	8.753	±0.061	16	0	0.00%
HiSilP1	ROCKLABS	12.050	±0.13	16	1	6.25%
Total				**32**	**1**	

Since not all the original certificates were available for validation, only a total of 32 of 53 certified standards were validated by GMS. Both ROCKLABS certified standards (SN8 and HiSilP1) control charts are illustrated in Figure 11.17and Figure 11.18.

The results of the data validated by GMS show overall a good accuracy and precision within the control limits of ± 3 standard deviation ("SD").

Figure 11.17: Control Chart of Certified Standard (SN38) – 2009 QA/QC Program

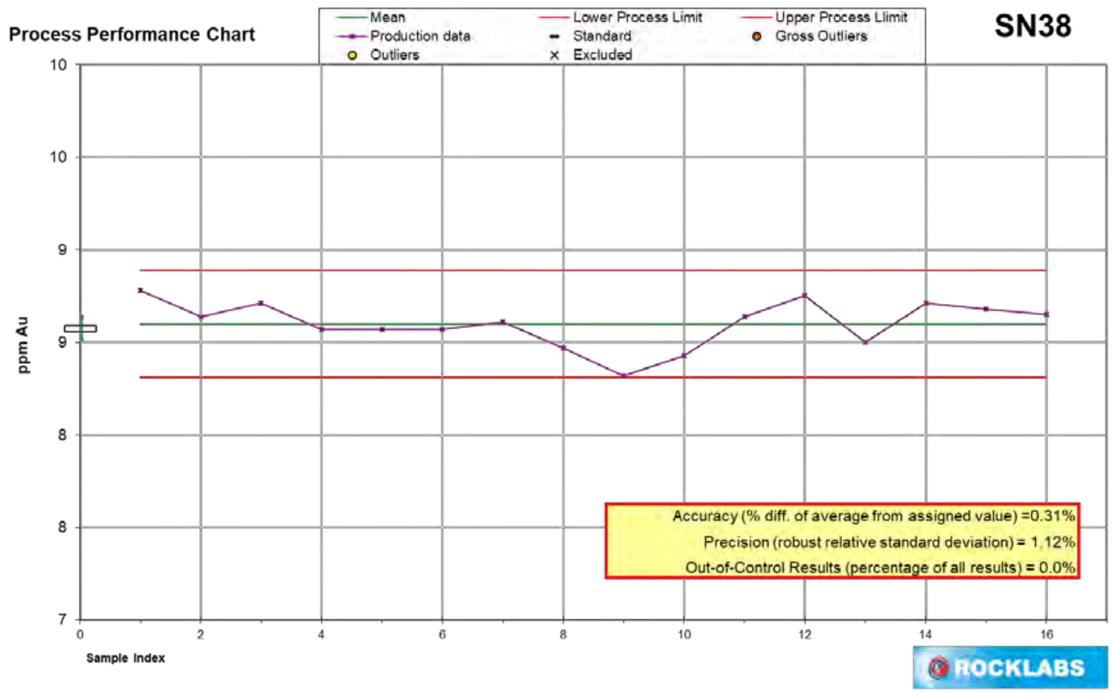
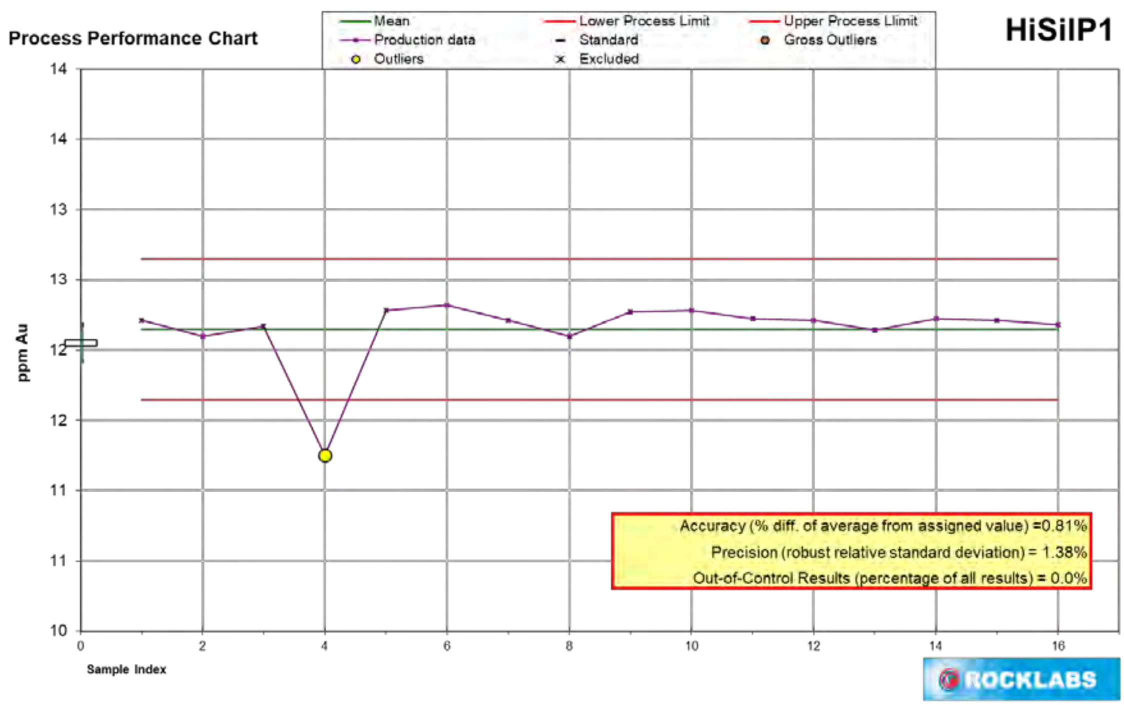


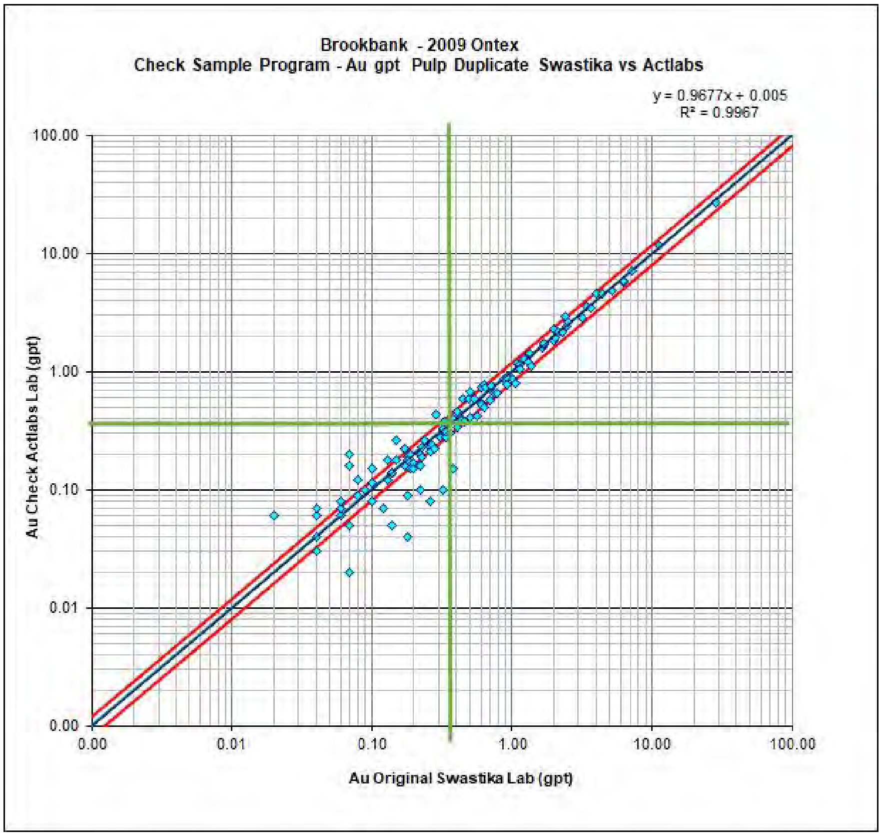
Figure 11.18: Control Chart of Certified Standard (HiSilP1) – 2009 QA/QC Program



11.2.1.1.2.3 Duplicates

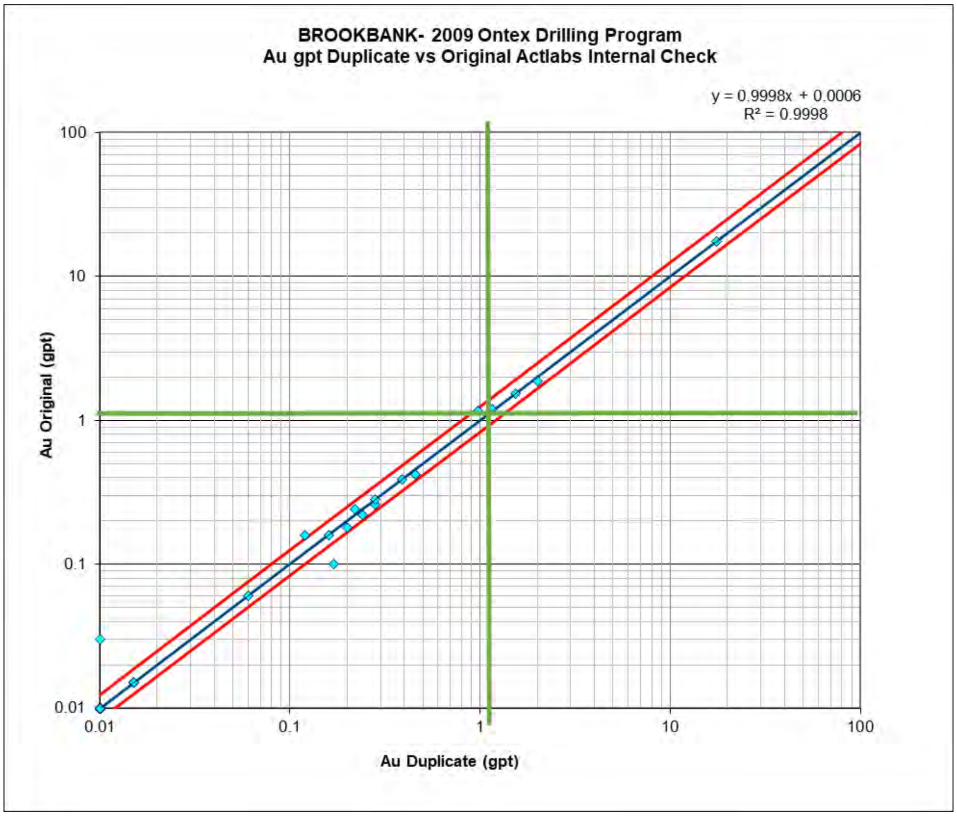
Ontex protocol was to quarter-cut the Brookbank drill core in the zones of significant mineralization. Samples were sent for analyses to both Swastika and Accurassay to check their reproducibility. In addition, 144 pulp duplicates were sent to both laboratories for comparison. Pulp duplicates comparing Swastika and Accurassay are shown in Figure 11.19.

Figure 11.19: Pulp Duplicate Results Comparing Swastika and Actlabs Laboratories - 2009



In addition, 44 pulp duplicates were assayed by Actlabs in 2009 as an additional quality control and is shown in Figure 11.20. The comparison shows excellent reproducibility of the original assay results.

Figure 11.20: Actlabs Internal Quality Control Chart – Brookbank (2009)



11.2.1.2 Kailey

Descriptions of sample preparation, analysis or security are not available for exploration work carried-out on the Kailey Project before previous companies to Premier Gold Mines Limited ("Premier"). All historical data integrated by GGM exploration team was obtained from original paper plans and sections. No assay certificates or QA/QC data of historical drilling programs are available for validation.

11.2.1.3 Key Lake

The information included in this section is based on data provided by GGM exploration team and based on a technical report completed by Geodatrix Consulting in March 21, 2011.

QA/QC data is available from 2010 and 2011, where a significant infill drilling program was conducted by Goldstone to increase confidence in the main resource area.

11.2.1.3.1 Sample Preparation and Analyses

All samples were sent to Actlabs in Geraldton or Thunder Bay, Ontario for sample preparation, with analysis carried-out at Actlabs in Thunder Bay, Ontario.

Samples were analysed by lead collection fire assay method with atomic absorption finish (30 g charge). All samples with results >3 g Au/t were subject to a gravimetric finish.

11.2.1.3.2 Goldstone QA/QC Program (2010 - 2011)

This section will discuss the QA/QC program performed at the Key Lake Project during 2010 and 2011.

Goldstone implemented a thorough QA/QC ("QC") program for the drill program at the Key Lake Project with the insertion of two certified reference material samples (standards), one blank (a coarse synthetic silica sand), as well as a field (1/4 core) duplicate, a crushed duplicate and a pulp duplicate in each batch of 38 samples.

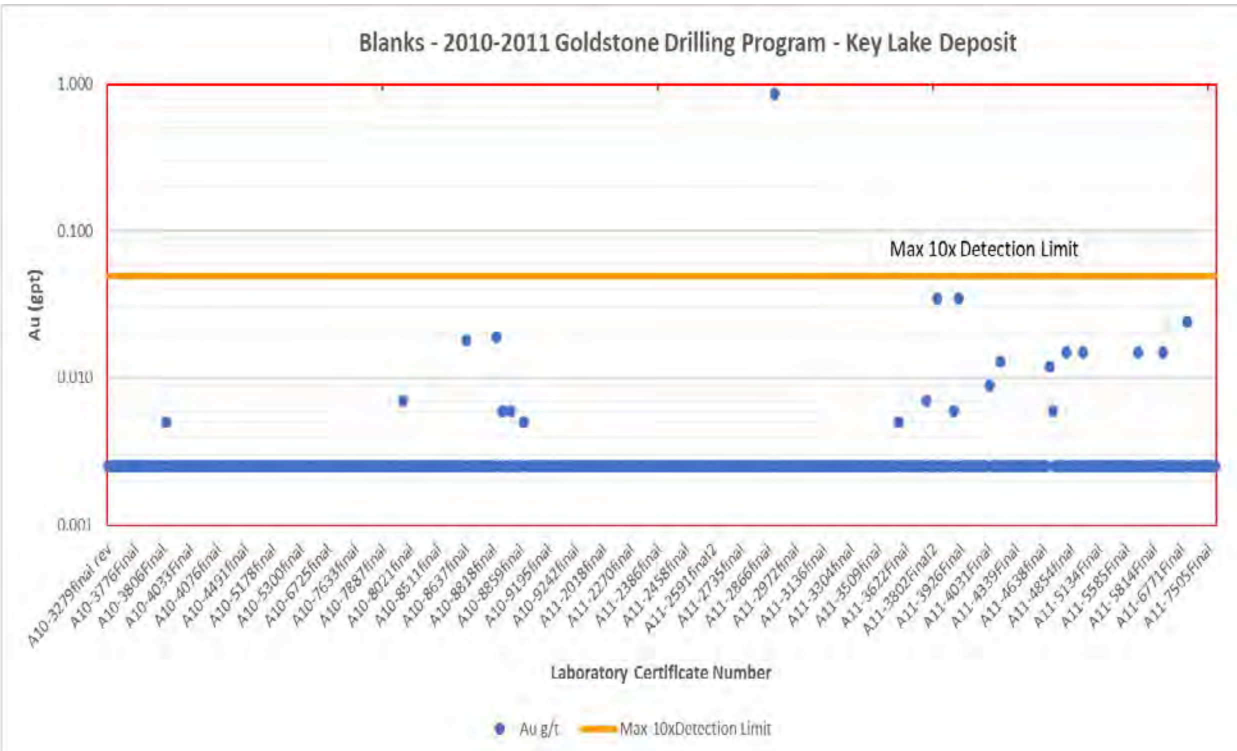
A total of 6,535 samples over 172 batches were sent to Actlabs for analysis. This number includes the QC samples described above inserted in each batch. Samples were assembled into batches ranging in size from 36 to 40 samples and all batches included two standards, one blank plus duplicate samples.

11.2.1.3.2.1 Blanks

All blank material data for Au were plotted, using an upper tolerance limit of ten times the detection limit ("10 x DL") of 0.005 ppb.

Figure 11.21 show that the blanks performed well with only one result falling outside the 10 x DL. The failure value was identified by the lab's internal QC, and has no impact on the resource database.

Figure 11.21: Performance of Blank for Au



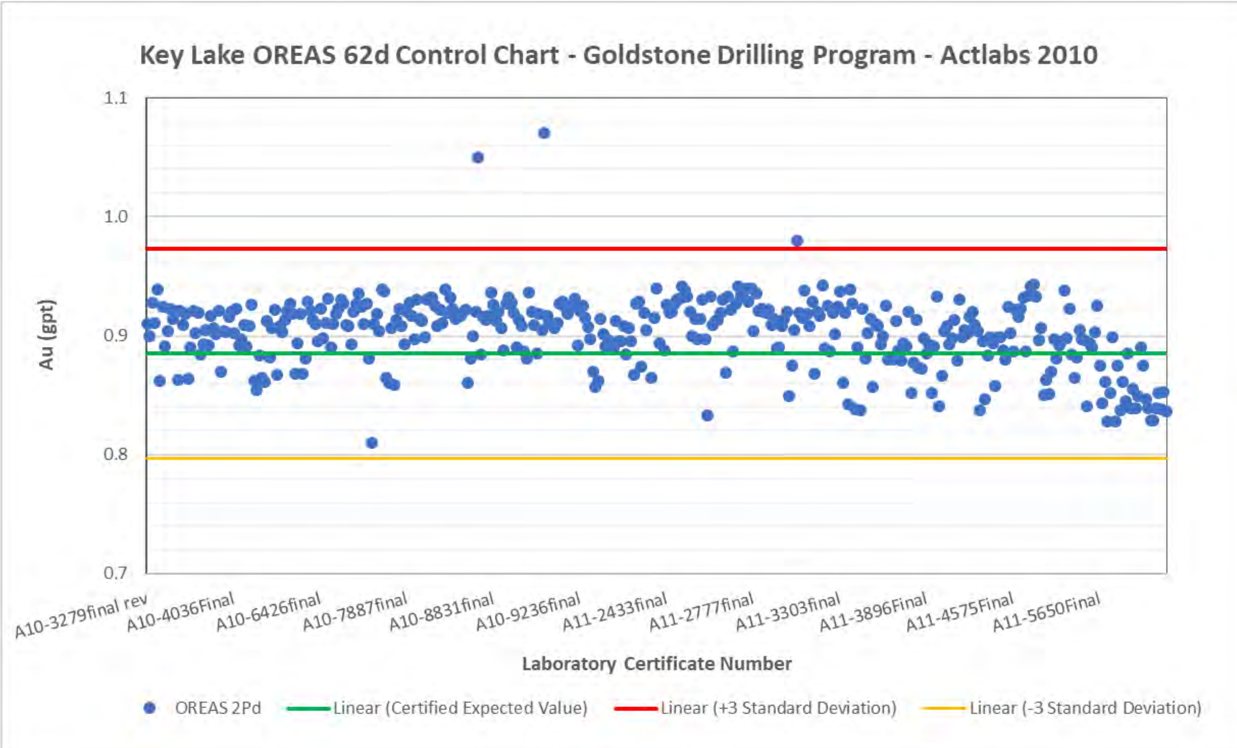
11.2.1.3.2.2 Certified Standards

A total of four different certified reference materials or standards were used throughout the 2010 and 2011 Key Lake drill programs. The OREAS certified reference materials (OREAS 2Pd, 6Pc, 54Pa and 62d) were purchased from Analytical Solutions Ltd. ("ASL") in Toronto, Ontario and the standard supplier was Ore Research & Exploration Pty Ltd. in Australia.

Standards OREAS 2Pd and OREAS 6Pc are plotted and shown in Figure 11.22 and Figure 11.23. All results outside ±3 standard deviations control limit from the certified mean value of the standard are considered as fails and a further investigation was needed to confirm and approve the result or rerun the batch of samples.

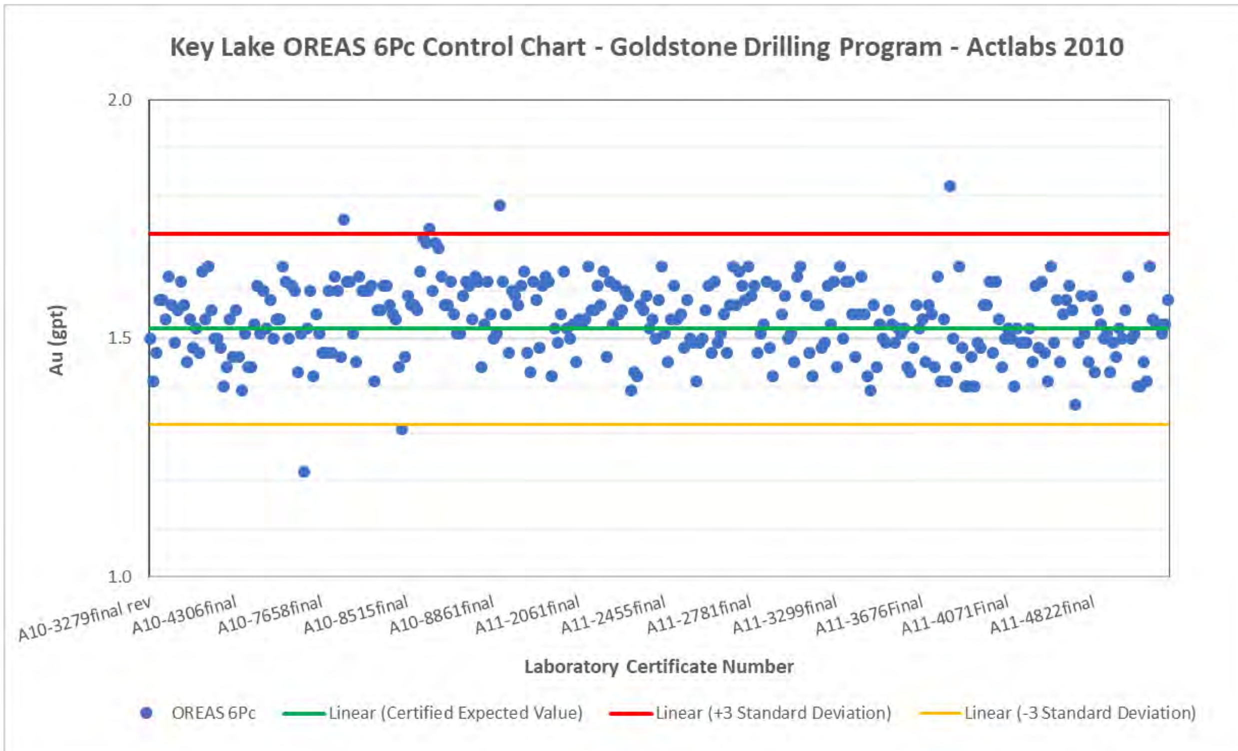
OREAS 2Pd performed well with only five failures (1.2%) recorded for gold (Figure 11.22). All the failures were re-run, and no major impact was detected from these results and no further action was judged necessary.

Figure 11.22: Performance of OREAS 2Pd Reference Material for Au



OREAS 6Pc performed well with eight failures for gold values analyzed at the laboratory. Five of these failures were validated and resolved, due to the other standard in the same batch passing the QC protocol, as well as conformance of the lab's internal QC.

Figure 11.23: Performance of OREAS 6Pc Reference Material for Au



OREAS 54Pa performed poorly during its insertion on sample batches and a low bias was noted for the recorded results. Recommendations were made to discontinue the use of this standard midway through the drilling program and the OREAS 54Pa standard was no longer used after certificate A10-5295.

11.2.1.3.2.3 Duplicates

The three different duplicate types (field, coarse reject and pulp) were analyzed throughout the 2010 to 2011 drilling program. There were 393 field, 397 coarse reject and 398 pulp duplicate pairs in the database and GMS compiled statistics to understand the precision at the various stages of fraction size and homogeneity of the samples.

The field (1/4 core) duplicates are expected to have the least precision, followed by the coarse reject duplicates, with the pulp duplicates having the best precision (due to fineness of grain size and homogenization).

Figure 11.24 show the results of field duplicates. The comparison has poor reproducibility with a coefficient R^2 value of 0.5566 which is mostly affected by the results of 2011 field duplicates.

Figure 11.24: Field Duplicates Control Chart – Key Lake (2010 to 2011)

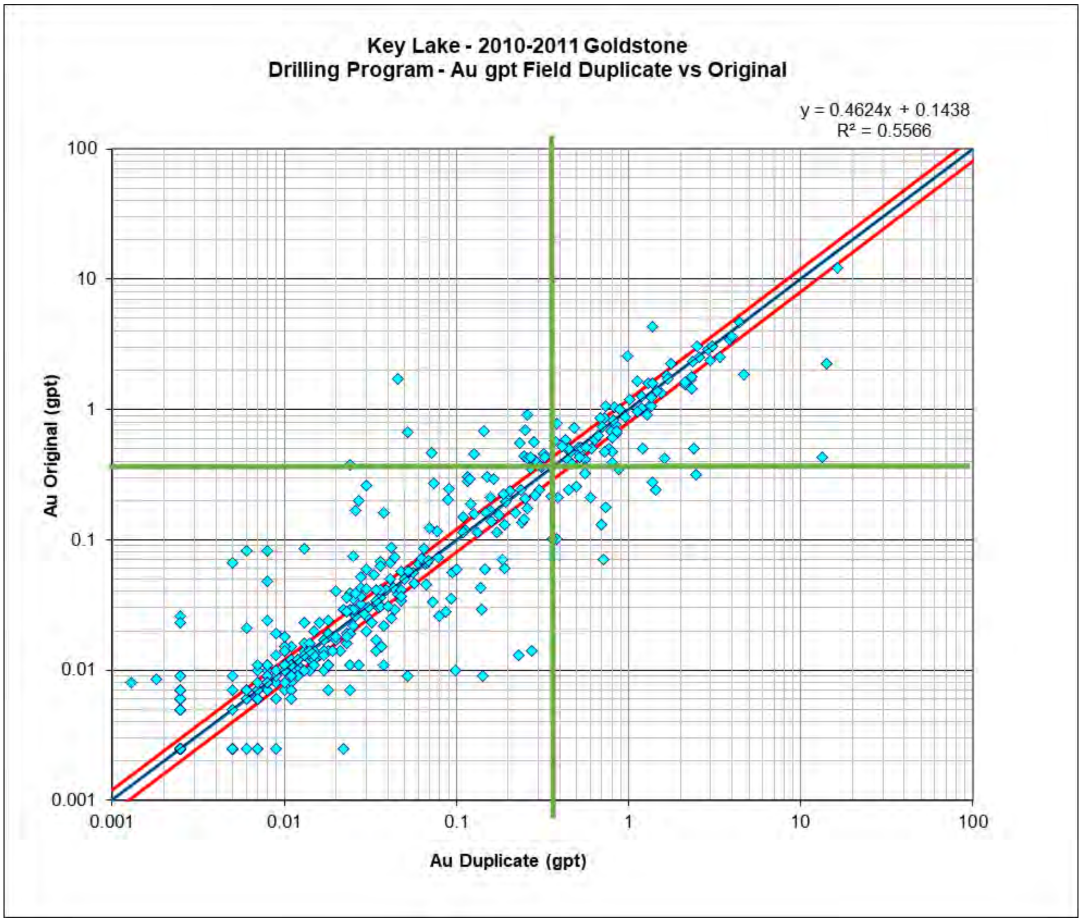


Figure 11.25 and Figure 11.26 illustrate a good reproducibility with a coefficient R^2 close to 0.9918 and 0.9836 respectively.

Figure 11.25: Coarse Reject Duplicates Control Chart – Key Lake (2010 to 2011)

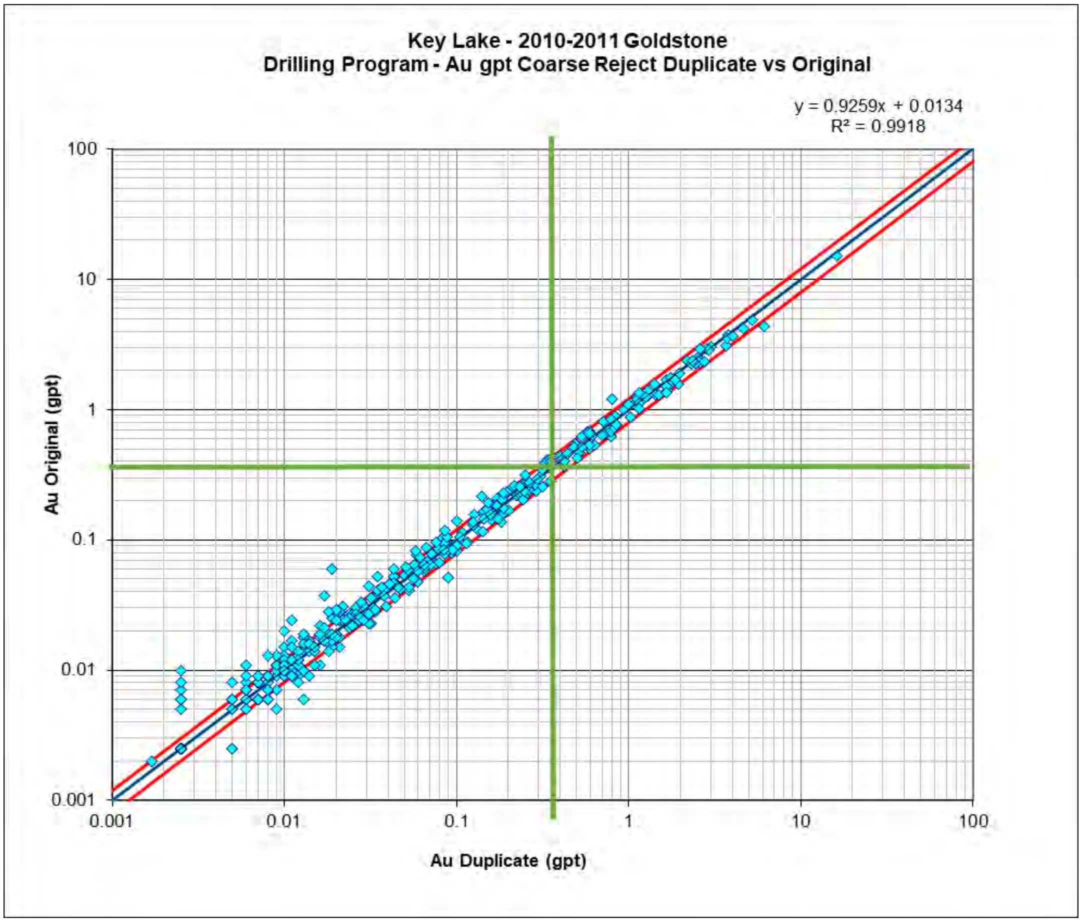
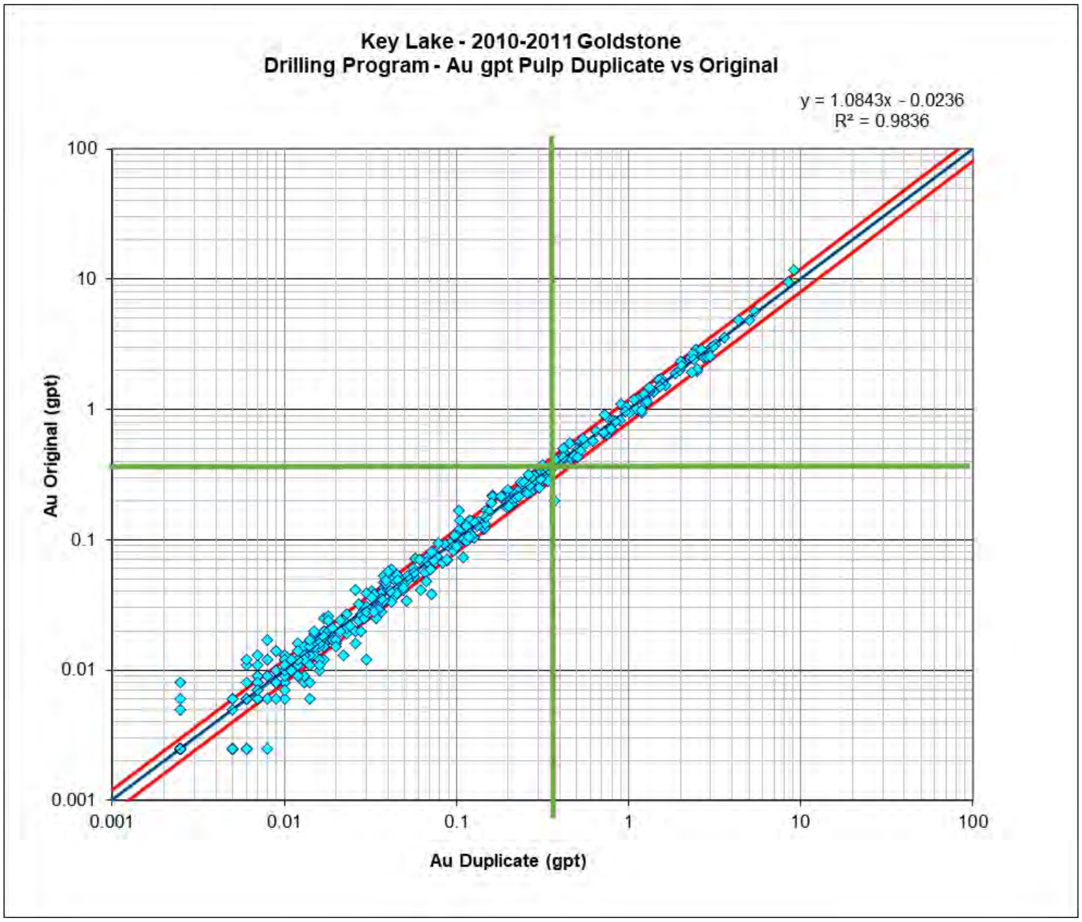


Figure 11.26: Pulp Duplicates Control Chart – Key Lake (2010 to 2011)



11.2.2 Premier Sampling Procedures and QA/QC

This section will discuss the most recent sample preparation, analysis, and security protocols which have been performed by Premier since the various historical drilling campaigns at the three deposits.

11.2.2.1 Protocols Before Sample Dispatch

Drill core sampling protocols are described in Subsection 11.1.2.2. Sample batches including the core drill and quality control samples were placed into rice bags, sealed and transported to Actlabs sample preparation facilities in Geraldton in trucks by Premier staff. Sample pulps were shipped to the Actlabs in Thunder Bay for analytical work. Actlabs is independent of Premier and provides analytical services to the mining and mineral exploration industry worldwide. It is ISO 17025 accredited. Other than the sampling and insertion of control samples, there was no other action taken at site.

11.2.2.2 Sample Preparation and Analysis

The description of sample preparation and analysis procedures used for the Brookbank, Kailey and Key Lake Projects are the same as described for the Hardrock Project in Subsection 11.1.3.

11.2.2.3 Quality Assurance / Quality Control

The QA/QC protocols implemented on all the projects at the Hardrock Property have been validated by GMS. The same method and approach as that adopted by Premier since 2009 was continued through 2010 to 2016 exploration programs.

During the drilling programs executed by Premier, a quality control procedure was implemented for quality monitoring purposes on each sample shipment. The procedure included the insertion of one certified standard, one blank material and one core duplicate for every batch of 34 samples sent to the assay laboratory.

11.2.2.3.1 Analytical Standards

From 2007 to 2016, sixteen (16) different certified reference materials (“CRMs” or “standards”) were used on all three deposits, and a total of 830 standards were inserted with the drill core samples. All these standards were purchased from CDN Resource Laboratories Ltd., ROCKLABS Ltd. and Accurassay Laboratories.

Ten (10) of the reference materials were purchased from CDN Resource Laboratories Ltd. (CDN-GS-1D, CDN-GS-2P, CDN-GS-4A, CDN-GS-5F, CDN-GS-5K, CDN-GS-C, CDN-GS-7A, CDNGS-8A, CDN-GS-P4B and CDN-P7J), four (4) of the reference materials were purchased from ROCKLABS (SH35, SL34, SG40 and SJ53) and two (2) of the materials was a standard prepared at Accurassay, (AUQ2-1 and HGS1-3).

The expected values, which each accredited Laboratory states on the certificates as the “recommended concentration”, and all the results of standards used in the Brookbank, Key Lake and Kailey Deposits are listed in Table 11.10, Table 11.11 and Table 11.12.

Table 11.10: Results for Standards Used by Premier During the Drilling Program on the Brookbank Deposit from 2016 to 2017

Au (g/t) Standard (s)			Limits		No. of Samples	No. of Failures	% Failure	Calculated Values			
CRM Code	Analytical Method	Certified Value	-3SD	+3SD				Mean Au	SD	CV	(%) Passing QC
CDN_GS_P4B	FA_AAS	0.417	0.348	0.486	217	21	9.7%	0.424	0.052	0.12	90.32%
CDN_GS_P7J	FA_AAS	0.722	0.614	0.830	158	14	0.0%	0.716	0.067	0.09	91.14%
CDN_GS_2P	FA_AAS	1.99	1.77	2.22	24	5	0.0%	1.955	0.097	0.05	79.17%
CDN_GS_5K	FA_AAS	3.850	3.460	4.240	48	12	0.0%	3.657	0.214	0.06	75.00%
CDN_GS_6C	FA_GRAV	5.95	5.23	6.67	2	0	0.0%	5.630	0.127	0.02	100.00%
Total					**449**	**52**					**88.42%**

Table 11.11: Results for Standards Used by Premier During the Drilling Program on the Kailey ("Little Long Lac") Deposit from 2007 to 2011

Au (g/t) Standard (s)			Limits		No. of Samples	No. of Failures	% Failure	Calculated Values			
CRM Code	Analytical Method	Certified Value	-3SD	+3SD				Mean Au	SD	CV	(%) Passing QC
CDN-GS-1D	FA_GRAV	1.05	0.9	1.2	33	1	3.03%	1.085	0.050	0.05	96.97%
CDN-GS-4A	FA_GRAV	4.42	3.73	5.11	1	0	0.00%				100.00%
CDN-GS-5F	FA_GRAV	5.27	4.76	5.78	37	2	5.41%	5.239	0.324	0.06	94.59%
CDN-GS-7A	FA_GRAV	7.20	6.3	8.1	1	0	0.00%				100.00%
CDN-GS-8A	FA_GRAV	8.25	7.35	9.15	1	0	0.00%				100.00%
AUQ2-1	FA_AAS	1.431	1.149	1.713	37	13	35.14%	1.261	0.278	0.22	64.86%
HGS1-3	FA_AAS	2.78	2.109	3.459	24	0	0.00%	2.657	0.168	0.01	100.00%
SG40	FA_GRAV	0.976	0.91	1.042	4	2	50.00%	1.063	0.033	0.03	50.00%
SH35	FA_AAS	1.32	1.191	1.455	90	14	15.56%	1.274	0.188	0.15	84.44%
SL34	FA_AAS	5.893	5.473	6.313	93	33	35.48%	5.449	1.033	0.19	64.52%
Total					**321**	**65**					**79.75%**

Table 11.12: Results for Standards Used by Premier During the Drilling Program on the Key Lake Deposit from 2011

Au Standard (s)			Limits		No. of Samples	No. of Failures	% Failure	Calculated Values			
CRM Code	Analytical Method	Certified Value	-3SD	+3SD				Mean Au	SD	CV	(%) Passing QC
CDN-GS-8A	FA_AAS	8.25	7.35	9.15	30	3	10.00%	8.180	0.528	0.06	90.00%
SJ35	FA_AAS	2.64	2.493	2.781	30	14	46.67%	2.518	0.070	0.03	53.33%
Total					**60**	**17**					**71.67%**

The analytical results were graphed to illustrate the performance of the quality control samples by using the convention of ± three times the standard deviations (± 3SD) control limit in which the standard values should be within. In case of failures, all re-analyzed batches (pulps) were sent ALS Minerals in Vancouver.

Figure 11.27 and Figure 11.28 present examples of standards control chart for Kailey and Key Lake Deposits.

Figure 11.27: Control Chart - Standard CDN-GS-5F : Outliers Included - Kailey Drilling Program (2011)

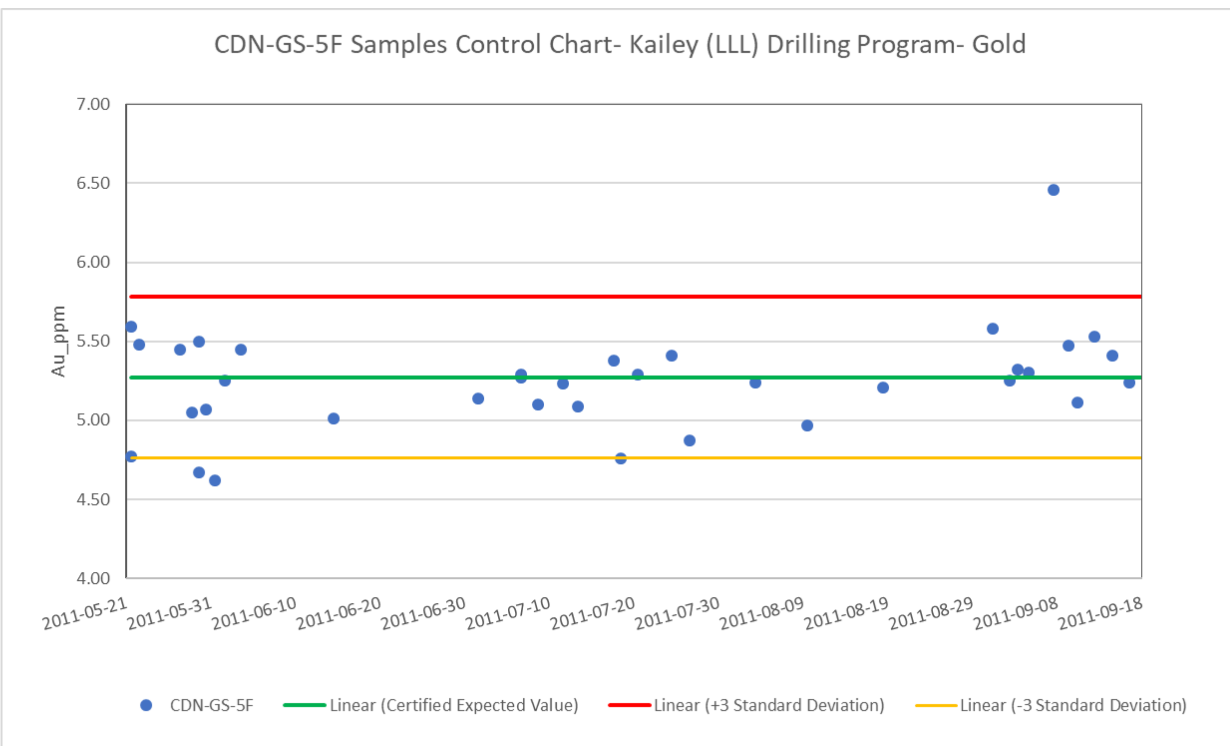
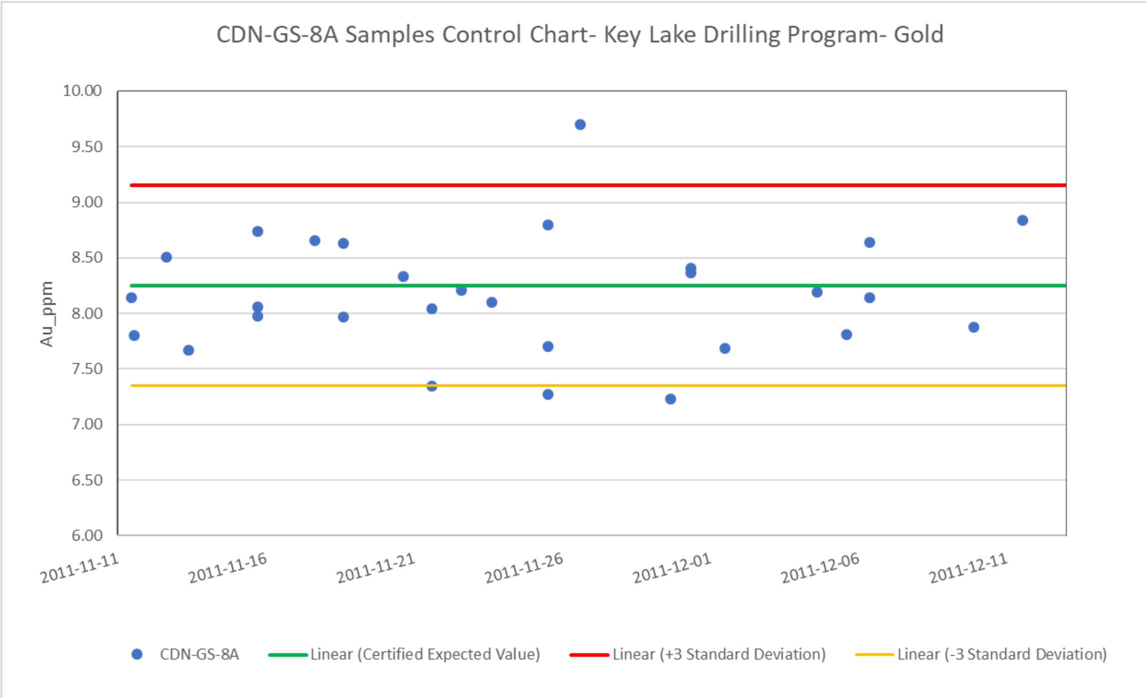
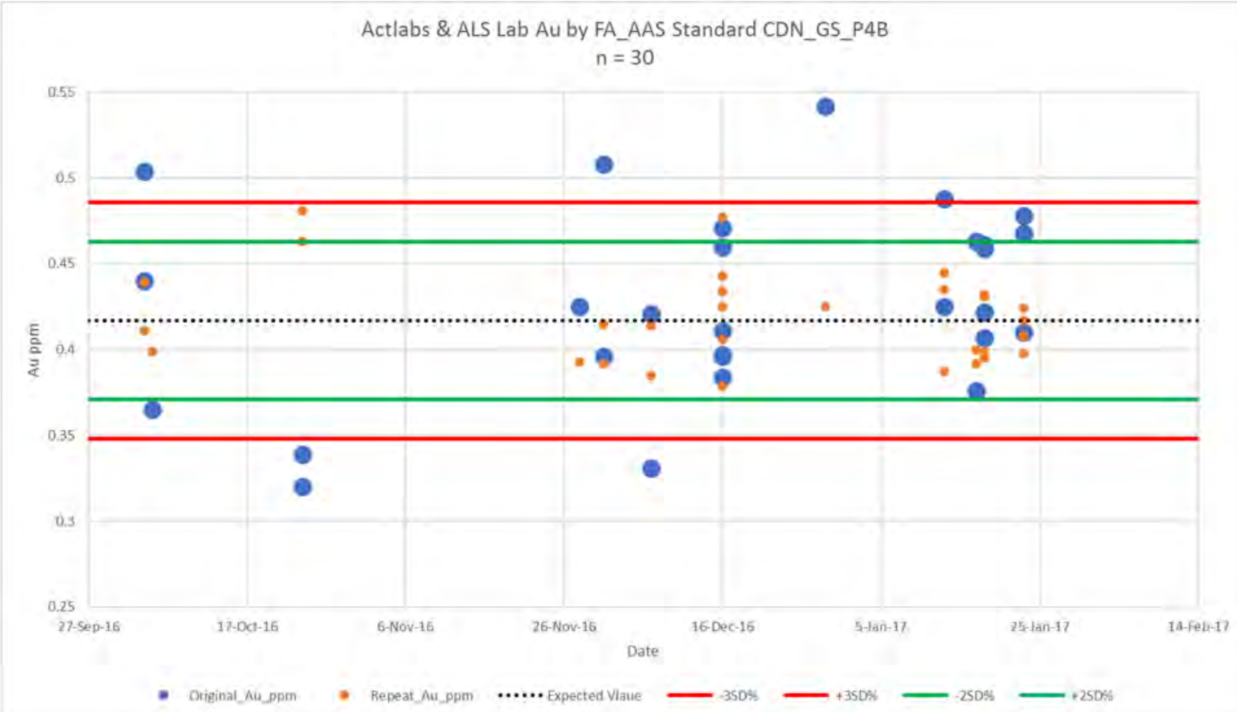


Figure 11.28: Control Chart - Standard CDN-GS-8A : Outliers Included – Key Lake Drilling Program (2011)



A representative number (15%) of Brookbank assay batches with standards that exceeded ±3SD limits were selected and sent for re-assay at GGM's Umpire laboratory (ALS Minerals). The certified reference material and the blank pulps used in the original batches were replaced with new material as the amount of pulp remaining is not enough for a 50 g Fire Assay ("FA") by AA finish analysis and to confirm the precision of the original results. All results returned from ALS labs, were within the recommended limits of ±3SD of the expected value for all standards submitted.

Figure 11.29: Control Chart of Certified CDN-GS-P4B – ALS vs Actlabs Laboratories Check (Source: GGM QA/QC Report,2016)



11.2.2.3.2 Analytical Blanks

The blank materials used by Premier were mostly diabase rocks from the Nipigon area that were tested for gold prior to using as blank samples. From December 2012 Premier used landscape rock as a blank. A total of 874 blank samples were inserted during the 2007 to 2016 drilling programs.

In 2016 at Brookbank 2016, the blank used was from a barren sample of crushed white gardening stone. QA/QC protocol requires that if any blank yields a gold value above 0.05 g Au/t (10 times the detection limit for analytical method), the batch containing the blank should be re-assayed. All batches to be re-assayed (pulps) were sent to ALS Minerals in Vancouver.

Of the 874 blanks only nine (9) blank sample analyzed for gold, produced a result over 10 times the detection limit (“10 x DL”) and no other assay result returned high-grade values as a product of a possible contamination. For all of the three projects within the Hardrock Property, the blanks consistently provided a good quality control sample as a barren material.

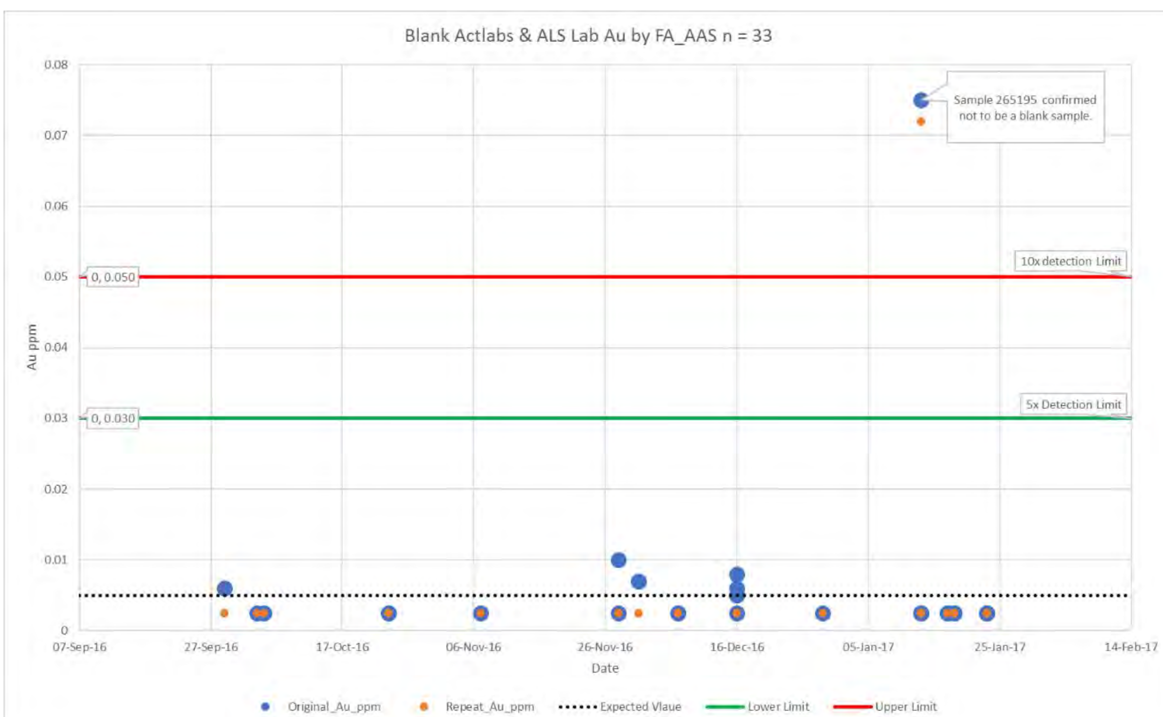
Table 11.13: Standard Blank : Outliers Included – All Projects (2011 - 2016)

Au (g/t) Blanks				No. of Samples	No. of Failures	% Failure	Calculated Values			
Prospect	Analytical Method	Certified Value	10x Detection Limit				Mean Au	SD	CV	Mean Bias
Brookbank	FA_AAS	0.005	0.05	250	1	0.40%	0.0034	0.005	1.47	-32%
Kailey				594	8	1.35%	0.009	0.06	7.03	-80%
Key Lake				30	0	0.00%	0.01	0.000	3.5E-16	100%
Total				**874**	**9**					

For all the Brookbank historical resampling, channel sampling and development/exploration drilling programs analyzed from August to December 2016, only one blank inserted into the sampling stream exceeded the recommended upper limit with a result of 0.075 g Au/t. According to the sample tag and database, sample 265195 was supposed to be a blank inserted at 371 m. The sample pulp was examined and consistent with a core sample pulp. It has been determined there was a sampling error and that the sample inserted was not a blank. The batch was sent for umpire assay at ALS Vancouver. The result was confirmed and all repeats and standards in the batch passed QA/QC.

Figure 11.30 shows rerun results returned from the ALS Mineral repeats within the expected range, well above the upper limit of 0.05 g Au/t except for sample 265195.

Figure 11.30: Umpire Repeat Assays – Actlabs vs ALS – Brookbank Project



11.2.2.3.3 Analytical Duplicates

Premier used split core for its duplicate samples until October 3, 2012. After this period, the duplicate samples consisted of two samples taken from the same piece of core which was crushed and riffle split at the assay laboratory. A combined total of 5,906 check assays or 6.1% were conducted by Premier for 2011 and 2012.

From 2012 to 2016, GGM quality control protocol required a coarse reject duplicate to be analyzed for the 30th sample of each batch. The duplicate is prepared by taking half of the crushed material derived from the original sample. By measuring the precision of the coarse duplicates, the incremental loss of precision can be determined for the coarse-crush stage of the process, thus indicating whether two (2) sub-samples taken after primary crushing is adequate for the given crushed particle size to ensure a representative sub-split.

As mentioned in Subsection 11.1.4.3, duplicates are inserted in a sampled batch to check the representativeness of results obtained for a given population. The reproducibility and precision (as a percentage) between two sample is calculated using the following formula:

$$\text{Precision (\%)} = \frac{\text{(Duplicate Sample Gold Grade – Original Sample Gold Grade)}}{\text{Average Between Duplicate Sample Gold Grade and Original Sample Gold Grade}} \times 100$$

Figure 11.31 represents the results of coarse rejects duplicates and show a linear regression slope of 1.0052 and a correlation coefficient of 99.9%. The duplicate results obtained indicate a good reproducibility of gold values with an AA finish performed by Actlabs.

Of the 231 coarse reject duplicates, a total of 8 duplicates had a relative difference >20% for gold values analyzed by FA AA (Table 11.14). Overall, all results with >20% relative difference are in low concentrations with less than 0.5 gpt Au. For gold values >0.5 gpt there are no outliers and show good reproducibility.

Figure 11.31: Scatter Plot – Drill Hole (Sample Name) : Original vs. Coarse Reject Duplicate for Au (g/t) FA AAS Analytical Method (Source: GGM QA/QC Report, 2016)

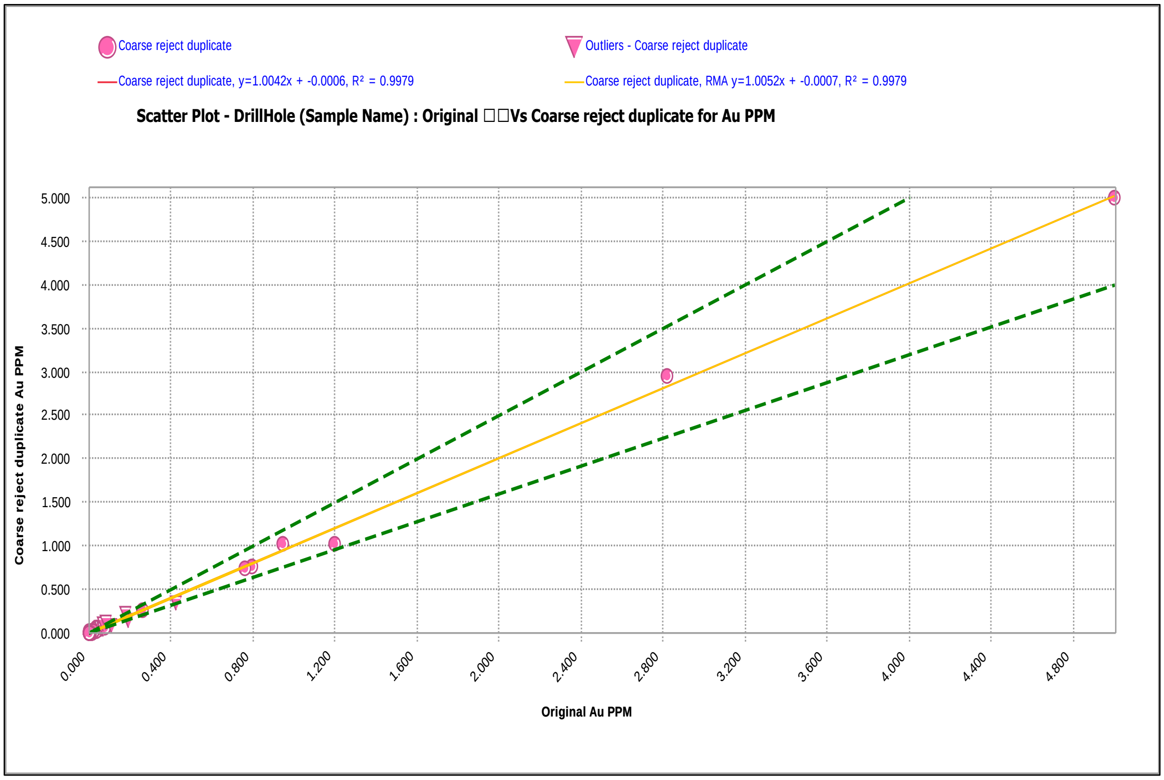
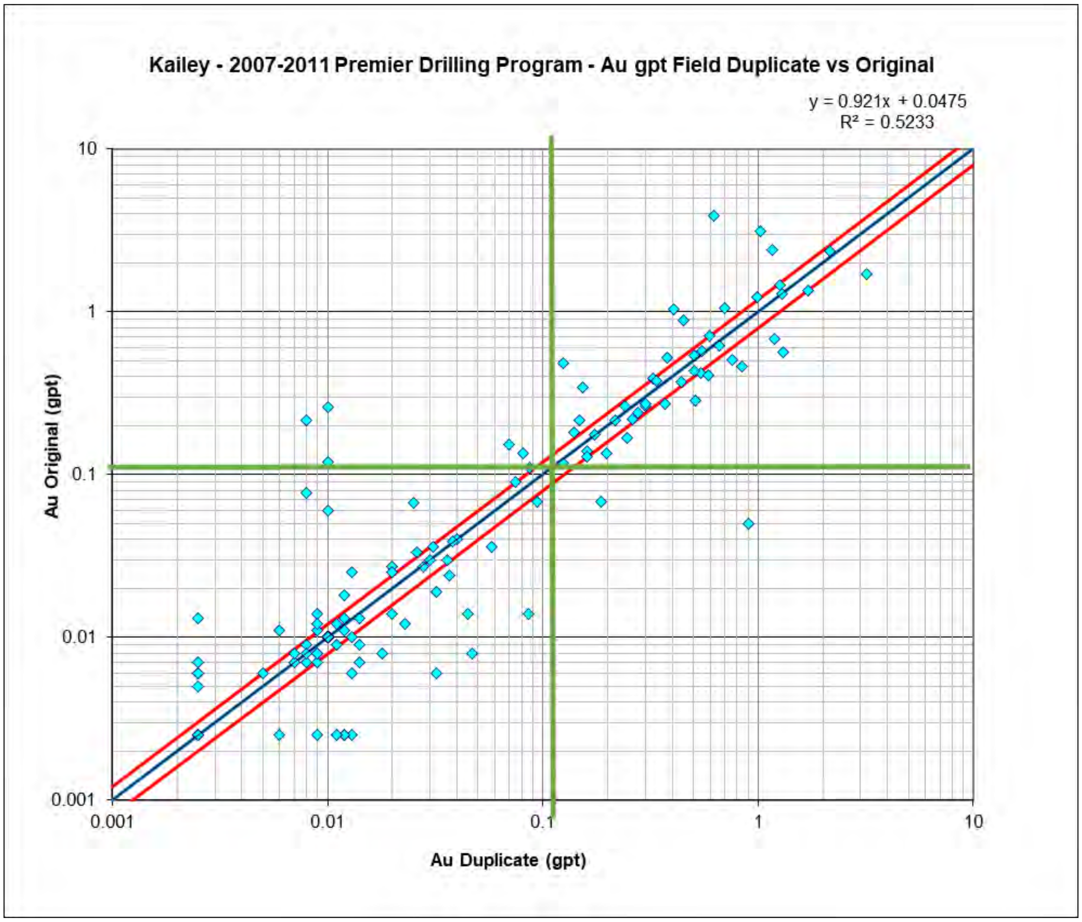


Table 11.14: Duplicate Gold Results with a Precision >20% (Control Limit) – Brookbank QA/QC Program

Sampled Batch No.	Area	Sample Number	Duplicate Sample Number	Duplicate Type	Analytical Method	Original Au Value (g/t)	Duplicate Au Value (g/t)	Precision (%)
A16-08448	Jellicoe	177849	177850	Coarse Reject	FA_AAS	0.108	0.075	-36.1
A16-08496	Brookbank	275411	275412	Coarse Reject	FA_AAS	0.053	0.067	23.3
A16-09102	Jellicoe	178359	178360	Coarse Reject	FA_AAS	0.18	0.23	24.4
A16-10709	Brookbank	178802	178803	Coarse Reject	FA_AAS	0.194	0.139	-33.0
A16-10718	Brookbank	178870	178871	Coarse Reject	FA_AAS	0.07	0.041	-52.3
A16-12074	Brookbank	246253	246254	Coarse Reject	FA_AAS	0.07	0.095	30.3
A16-12366	Brookbank	263610	263611	Coarse Reject	FA_AAS	0.425	0.338	-22.8
A17-00252	Brookbank	229447	229448	Coarse Reject	FA_AAS	0.079	0.12	41.2

Duplicate sample program performed at Kailey from 2007 to 2011 are illustrated in Figure 11.32. In total, 72 of 148 field duplicates returned values outside the ±20% control limit. Most of the failures are in the low concentration of the graph. The impact of these failures is relatively low and will not affect the total resource calculation in this area.

Figure 11.32: Field Duplicates Control Chart – Kailey (2007 to 2011)



The duplicate drill core sampling program performed by Premier in 2011 consisted of 30 samples. The total amount of data can not be used to confirm the precision of the duplicated sample.

11.2.2.4 Security

The Premier Project Manager, a P. Geo, supervised all aspects related to sampling, recording, packaging and transportation of samples to the laboratory. James Purchase, P.Geo of GMS inspected sampling facilities and core storage areas, which is discussed further in Section 12 – Data Verification.

All Premier's drill core samples are kept within Premier's core logging or sampling facility until shipment to the laboratory. Drill core samples were sawn (in half length-wise) using a diamond saw at its core logging/cutting facility in Geraldton. Samples of halved drill core were sealed in labelled plastic samples bags and securely packed for shipping. Bags of samples were then shipped by road transport to the Actlabs preparation facilities in Geraldton since June 2009. The samples were delivered to the preparation facility by Premier staff.

11.2.3 QP Conclusions

GMS has reviewed all the information regarding sample preparation, security, and analytical procedures (QA/QC) used to ensure the accuracy of assays at the Brookbank, Kailey and Key Lake Projects, and considers that the drilling and assay database are suitable for use in the calculation of a mineral resource.

GMS recommends the following for any future exploration programs:

- Ensure that the insertion into the sample stream rate of CRM (5%), blank material (5%), field (5%), coarse and pulp (5%) duplicates is achieved.
- Continue to document warnings, failures, and most importantly any remedial action taken;
- If a CRM shows consistent bias at multiple laboratories, this issue needs to be understood and resolved in a reasonable period of time and if needed replace it with a new CRM.
- Add always the Hole ID number to the QA/QC sample database as a cross check to ensure QA/QC samples relate to the dataset and the time period in question.
- Increase umpire sample submission rate to around 5% of all samples.

12. DATA VERIFICATION

The following section has sourced information from the NI 43-101 Technical Report for the Hardrock, Brookbank and Key Lake Projects prepared by G Mining Services Inc. ("GMS", December 22, 2016). As the 2016 MRE database formed the basis of the 2019 MRE, this information is still valid and has been included for the sake of completeness. New data verification activities were undertaken for the Brookbank, Kailey and Key Lake Deposits since the 2016 Technical Report, which are described in Section 12.2.

12.1 Hardrock Data Verification

12.1.1 Data Verification for the 2016 MRE

The diamond drill hole database used for the 2016 Mineral Resource estimate presented herein was provided by Greenstone Gold Mines ("GGM") and is referred to as the "GGM database" in this section. A drilling program in the Hardrock Deposit resource area ended on July 20, 2015, and the database close-out date for the resource estimate update was established as November 18, 2015. The last hole included in the database is MM754B. A significant re-sampling program was also completed in 2015 by GGM, including 6,411 new samples from 79 historical diamond drill holes. These were added to the GGM database for the resource estimate update herein. The 2014-2015 stripping program was also included in the update.

GMS' data verification included visits to the Hardrock field sites (outcrops and drill collars), as well as to the logging facilities. It also included an independent re-sampling of selected core intervals and a review of drill hole collar locations, assays, the QA/QC program, downhole surveys, the information on mined-out areas and the descriptions of lithologies, alterations and structures. The site visit was completed by Réjean Sirois, a GMS employee and QP, between August 1st and 4th, 2016.

12.1.1.1 Historical Work

The historical information used in this Report has been taken mainly from reports produced before the implementation of NI 43-101 Canadian Standards of Disclosure for Mineral Projects. In some cases, these reports provide little information on sample preparation, analyses or security procedures.

12.1.1.2 GGM Database

GMS was granted access to the certificates of assays for all holes in the latest drilling programs that took place between May 2014 and July 2015. Assays were verified for 2% of the drill holes from these programs.

Minor errors of the type normally encountered in a project database were identified and corrected. The final database is considered to be of good overall quality. GMS considers the GGM database for the Hardrock Deposit to be valid and reliable.

12.1.1.3 Greenstone Gold Mines Diamond Drilling

The historical surface drill holes collars on the Hardrock Deposit were either professionally surveyed or surveyed using a Trimble GPS unit without post-processing. However, the 2015 drill hole collars were surveyed using an RTK system with millimetre precision in all directions, including elevation.

Underground drill holes were compiled by Greenstone Economic Development Corporation ("GEDC"). However, these holes were excluded from the current resource estimate since the location data is considered unreliable, and the assay results could not be verified.

Downhole surveys were conducted on the majority of the surface holes. The Gyro and/or Reflex survey information was verified for 5% of the drill holes from the latest drilling programs. Minor errors were observed in the downhole surveys and corrections were made to the database. For the 2015 drilling program, final collar azimuths and dip measurements were collected directly on the casing using an APS system. Gyro, RTK and APS survey methods were reviewed during the site visit. Figure 12.1 and Figure 12.2 show the different survey tools and some examples of drill sites that were reviewed during the site visit.

During the GMS site visit, a total of seven drill hole collars were checked for X-Y accuracy. A handheld Garmin GPS was used to collect ground survey data, as summarized in Table 12.1. Given the accuracy of handheld GPS, the results are judged satisfactory by GMS. Figure 12.1 shows some examples of drill hole collars surveyed during the site visit.

Table 12.1: Drill Hole Collar Checks - 2016 Site Visit

Hole-ID	Check		Database		Difference	
	Easting	Northing	Easting	Northing	Easting	Northing
88-17A	504,781	5,502,825	504,781	5,502,827	0.1	2.0
EP100	504,451	5,502,970	504,450	5,502,969	-1.5	-0.8
EP120	504,400	5,502,999	504,402	5,502,998	1.6	-0.7
EP161	504,900	5,502,929	504,900	5,502,930	0.4	0.8
MM267	504,798	5,502,801	504,800	5,502,800	2.3	-1.1
MM534	504,503	5,502,963	504,501	5,502,965	-2.4	1.7
MM598	504,247	5,502,968	504,250	5,502,964	3.2	-4.0

Figure 12.1: Drill Hole Collars Surveyed during GMS 2016 Site Visit

12.1.1.4 GGM Logging, Sampling and Assaying Procedures

GMS reviewed several sections of mineralized core while visiting the on-site core logging and core storage facilities. All core boxes were labelled and properly stored outside. Sample tags were still present in the boxes and it was possible to validate sample numbers and confirm the presence of mineralization in witness half-core samples from mineralized zones.

Drilling was not underway in the resource area during the GMS site visit. GGM personnel explained the entire path of the drill core, from the drill rig to the logging and sampling facility and finally to the laboratory (Figure 12.2). GMS is of the opinion that the protocols in place are adequate.

Figure 12.2: Core Logging Procedures Reviewed during Site Visit

12.1.1.5 <u>Independent Re-sampling</u>

GMS re-sampled a series of intervals from the latest drilling program. During the site visit, quarter-splits of selected core intervals were cut by GGM personnel. The author collected several samples representing different types of host rocks and a wide range of gold grades were re-analyzed at Actlabs Geraldton. Samples were collected in a random order inside relevant mineralized intercepts. For each zone and drill hole, one sample was collected at around 20 m interval, when possible. Only samples grading more than 1.0 g Au/t were selected and 50 cm of quarter core splits were collected randomly in the sample interval.

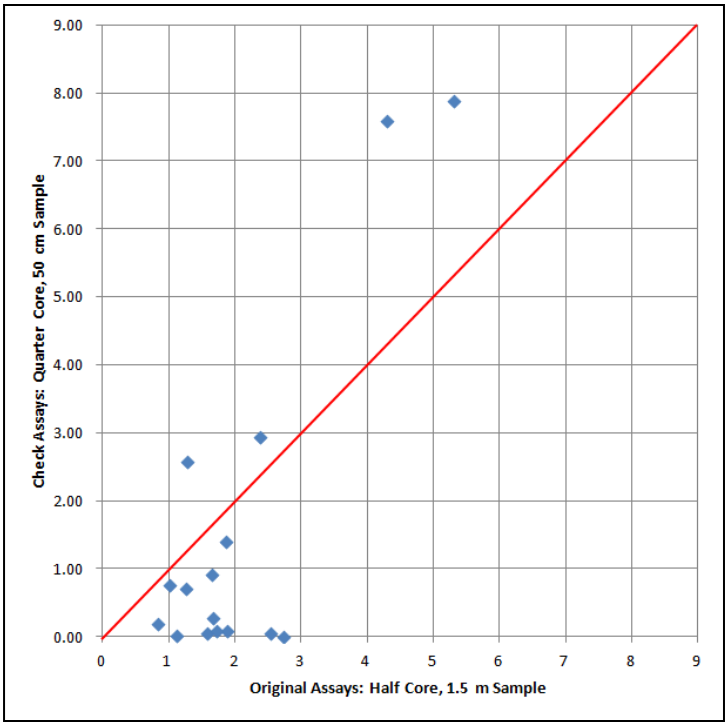
A total of 16 samples were assayed for gold using fire assay with AA finish. Samples assaying more than 5 g Au/t with AA were rerun with gravimetric finish. Table 12.2 presents the results of the field duplicate compared to the original samples.

Table 12.2: Original and Re-sampling Gold Analysis Results

DDH	Zone	From (m)	To (m)	Length (m)	Original Grade (g Au/t)	Lab Check (g Au/t)	Check Sample #
MM444	3300	481.5	482	0.5	4.29	7.59	262701
MM444	11140	514.1	514.6	0.5	2.54	0.05	262702
MM444	11140	532.9	533.4	0.5	0.82	0.21	262703
MM534	3600	314	314.5	0.5	5.31	7.90	262704
MM700	3205	333.4	333.9	0.5	1.00	0.78	262705
MM700	3205	355	355.5	0.5	1.85	1.41	262706
MM700	3205	368.6	369.1	0.5	1.28	2.58	262707
MM752	3500	333.8	334.25	0.45	1.67	0.29	262708
MM752	3105	458.2	458.7	0.5	1.58	0.07	262709
MM752	3105	479.8	480.25	0.45	1.11	0.03	262710
MM494	3105	341.3	341.8	0.5	1.26	0.72	262711
MM494	3105	361.6	362.1	0.5	2.72	0.01	262712
MM494	3105	383.2	383.7	0.5	2.38	2.95	262713
MM503	3205	481.8	482.3	0.5	1.87	0.09	262714
MM503	3205	499.3	499.8	0.5	1.72	0.10	262715
MM503	3205	528.5	529	0.5	1.64	0.93	262716

Figure 12.3 presents a linear graph comparing original samples and the field duplicate samples for all 16 samples. This graph displays that six out of 16 samples were reproduced within a 50% confidence level. Two more samples yielded a higher result compared to the original assay (+77% and +102%). The remaining samples (8) all show a significant decrease in gold grades, ranging from 75% to near 100%. Since one sixth 1/6) of core samples were randomly selected in the original sample interval (0.50 m quarter core interval versus 1.5 m half core), GMS is satisfied with the results given the mineralization style of gold and the inherent nugget effect.

Figure 12.3: Original Assays Compared to Check Assays



12.1.1.6 <u>Mined-out Voids</u>

Considerable effort has been made to improve the accuracy of the stope and drift 3D objects to provide a more accurate representation of the mined-out volumes in the historical workings. In 2015, a thorough archival search was undertaken by GGM and yielded additional historical plan views, cross sections and longitudinal views. An exhaustive compilation of breakthrough drilling was also completed by GGM. This additional information allowed the 3D model to be adjusted and corrected, and also provided additional missing stopes and drifts.

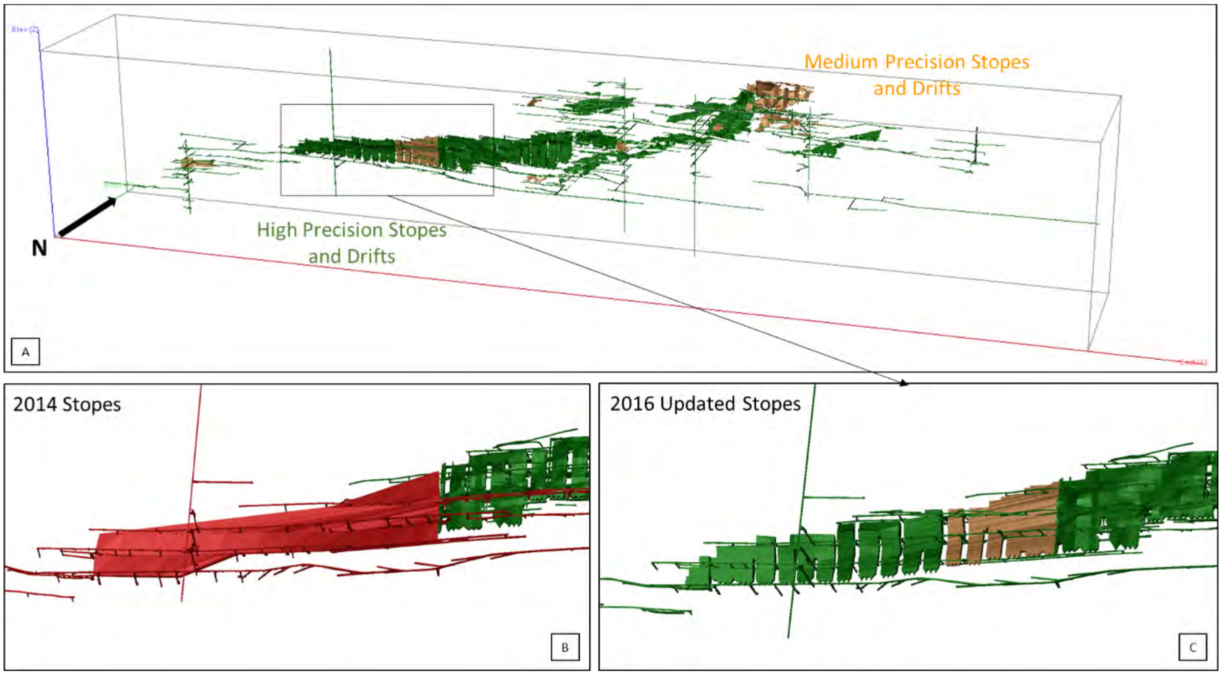
Based on the type of data used to model each void, the voids were classified as medium- or high-precision.

- *Medium-precision voids*: modelled using only digitized longitudinal views combined with breakthrough drilling information;
- *High-precision voids*: modelled using digitized plan views and/or cross sections with accurate location information for drift and stope positions.

In the end, the new information allowed all the low-precision stopes of the 2014 model to be upgraded to medium-precision in the 2016 model.

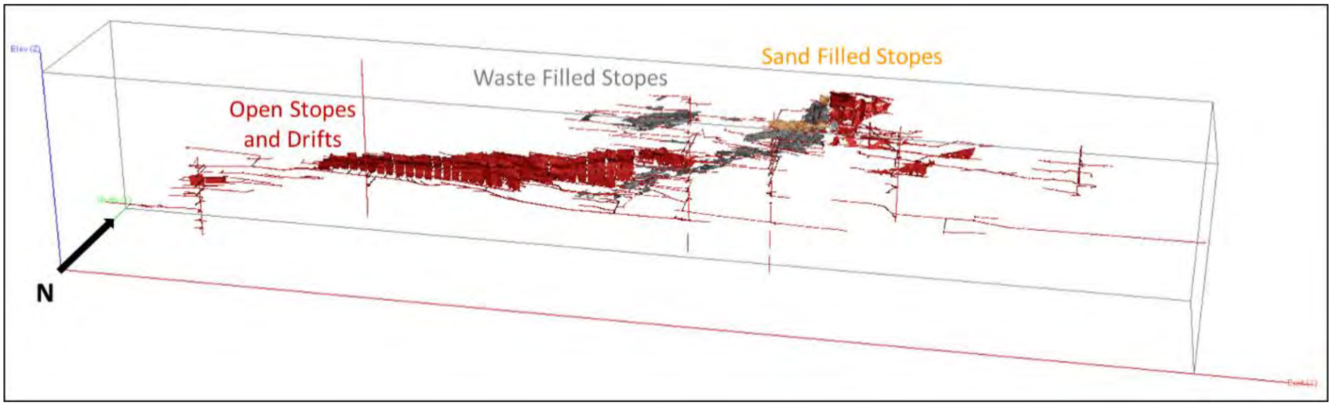
Figure 12.4 shows a compilation of the underground voids based on their level of precision as a result of the 2016 update.

Figure 12.4: Isometric View looking NNW showing a Compilation of the Mined-out Underground Voids: A) Overall View of Stopes and Drifts by Level of Precision; B) Close-up View of the Stopes Modelled in 2014; C) Close-up View of the Stopes Updated in 2016



Information on the type of backfill in the stopes was updated from the 2014 compilation and integrated into the database. The result is a classification of stopes according to three types of backfill: open (filled with water); waste (corresponding to a mix of waste and “clinker”, a reject from the process plant); and sand (corresponding to a mix of wet sand and gravel). Figure 12.5 shows a compilation of the underground voids based on backfill type. The specific gravities for each type of backfill were provided by GGM.

Figure 12.5: Isometric View looking NNW showing a Compilation of the Mined-out Underground Voids Based on their Backfill Type



For the 2016 update, the total stopes model corresponds to 89% of the total historical milled tonnes at an average density of 2.84 g/cm^3 for the Hardrock Deposit, including stopes in the Hard Rock, MacLeod-Cockshutt, Mosher Long Lac and Macleod-Mosher mines.

GMS considers the refinement of the voids triangulation to be of good quality and reliable.

12.1.1.7 Conclusion

Overall, GMS is of the opinion that the data verification process demonstrated the validity of the data and protocols for the Hardrock Project. GMS considers the GGM database to be valid and of sufficient quality to be used for the 2016 MRE.

12.1.2 Data Verification for the 2019 MRE

Drilling activities at the Hardrock Deposit Resource area ended on April 25, 2019, and the database close-out date for the resource estimate update was established as **May 14, 2019.** GMS appended the new drilling data acquired in 2018 and 2019 to the 2016 MRE database described in the previous section.

In 2018, a significant RCGC drilling campaign was completed by GGM comprising of 20,015 m of drilling from 405 RC drill holes, as calculated from the data provided in September 2018. This new information was appended to the drilling database in preparation for the MRE update. In addition, 76 RCGC holes (5,946 m) and 53 diamond drill holes (12,009 m) were drilled in the Project during 2019. A single failed diamond drill hole was excluded from the database. All data collected from these drill holes were incorporated into the drilling database for the current MRE update.

GMS' data verification for the Hardrock Project consisted of numerous site visits to monitor drilling activities, reviewing new drill hole data merged into the 2016 MRE database, reviewing new voids, and the review of new lithology, alteration and structural data. Finally, the verification also included a comparison of the RCGC assay grades versus the diamond drilling assay grades on section. The dates of site visits completed by Réjean Sirois of GMS in 2018 and 2019 are shown below:

- 24-25 May 2018;
- 03-04 Oct 2018;
- 25-26 Feb 2019;
- 10-11 April 2019;
- 08-09 May 2019;
- 29-30 July 2019.

12.1.2.1 2019 MRE Drilling Database

The 2019 MRE database contains a total of 481 drill holes from RCGC and 1,682 drill holes from surface totalling 2,163 drill holes in the Mineral Resource area (Table 12.3). From this, 534 new drill holes were completed by GGM and included in the present mineral resource since the 2016 MRE.

The database includes drilling totalling 536,850 m assayed for gold taken from 722,086 m of RC and DDH programs (see Table 12.3). The 26 channel samples totaling 1,323 m that were used to estimate the Project Mineral Resources in 2016 were excluded from the current MRE.

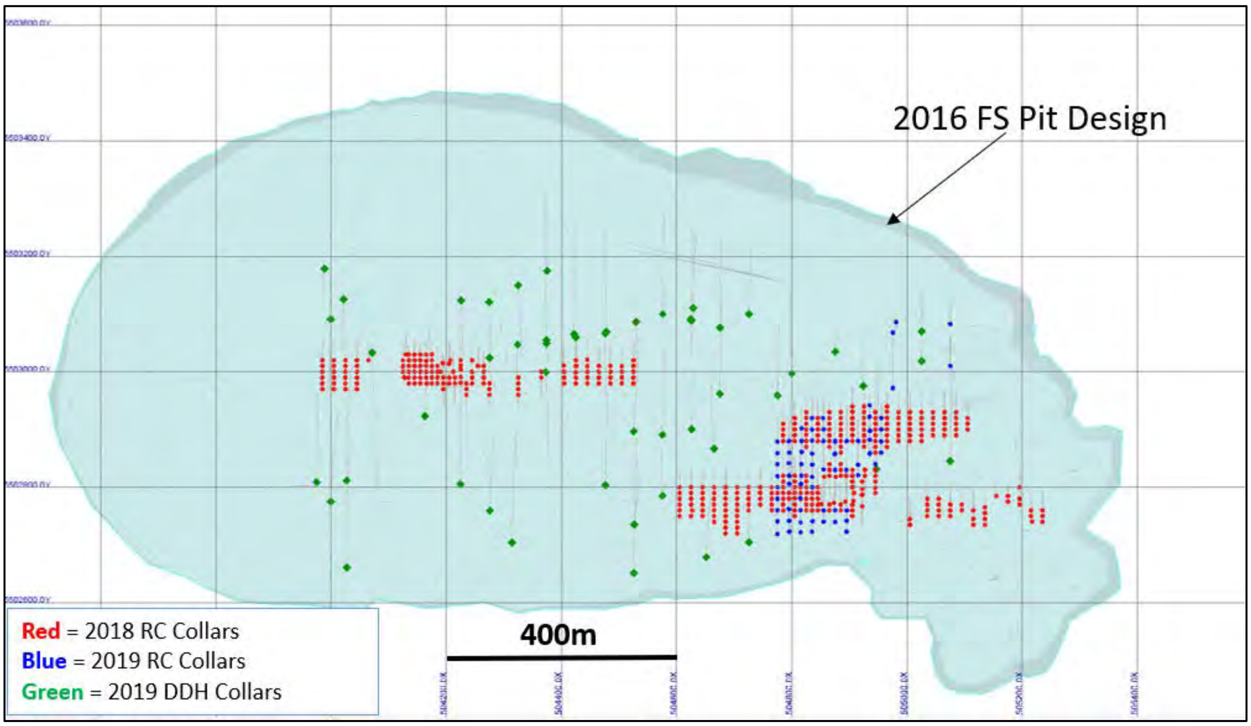
Table 12.3: Hardrock Gold Deposit - Resource Database Summary

Type of Drill Hole	No. of Drill Holes	Metres of Drill Holes	Metres of Assayed Samples
2016 MRE DDH	1,629	684,116	502,776
2019 DDH	53	12,009	10,469
Total DDH	**1,682**	**696,125**	**513,245**
2018 RCGC	405	20,015	18,050
2019 RCGC	76	5,946	5,555
Total RCGC	**481**	**25,961**	**23,605**
Grand Total	**2,163**	**722,086**	**536,850**

GMS reviewed the updated GGM database, and only minor errors were detected. Subsequent to the completion of the 2019 MRE, GMS became aware that GGM staff had renamed the 2018 RCGC drill holes with new hole ID's and resurveyed several drill collars. As these events were subsequent to the completion of the 2019 MRE, GMS retained the originally provided drill hole ID's and collar surveys for the 2018 RCGC drilling. Of the 52 collars that were revisited, only 6 of the 405 RCGC drill holes from 2018 are affected by a deviation in the collar survey by greater than 5 m. GMS does not believe that these deviations have a material effect on the 2019 MRE, but recommend that they are corrected for future mineral resource estimates. Drilling data from the 2019 drilling campaign is unaffected and up to date in the drilling database.

Figure 12.6 shows the drilling campaigns performed by GGM on the Property between 2018 and 2019.

Figure 12.6: 2018-2019 Drilling Programs - Hardrock Project



GMS was provided access to the original assay certificates for all 2018-2019 holes drilled in the deposit area. In total, 7% of 2018 RCGC sampling and 25% of the 2019 RCGC and DDH samples assayed for gold were verified by GMS by comparing them to the original gold values indicated by the laboratory certificates.

There were no significant errors or issues identified with the GGM database. The drilling database is considered to be of good overall quality. GMS is of the opinion that the GGM database for the Hardrock Deposit is of sufficient quality to be used for the current Resource estimate.

12.1.2.2 Mined-Out Voids Model Update

Prior to 2016, the wireframes of mined-out areas and existing development were built using historical plan views, cross sections, and/or longitudinal views. Each wireframe was classified as medium or high precision based on the source of information used to build the wireframe. Low-precision wireframes were based solely on drift plans.

12.1.2.2.1 GGM Review of the Mined-out Void Model

GGM undertook a study in May 2019 comparing the voids intersected in drilling against the void wireframes included in the 2019 MRE. The following information has been sourced for this Study:

Drill holes from the 2018 and 2019 drilling campaigns were used to adjust and assess accuracy of the historical underground ("UG") working wireframes. The wireframes representing mined-out areas were last updated during the 2016 MRE. Since then, no additional work has been made to provide a more accurate representation of the mined-out volumes in the historical UG workings.

The highlights observed between the last updated mined-out void model and the historical UG working data collected by the infill drilling programs performed in 2018-2019 are as follow:

- A total of 75 voids were encountered during the 2018 and 2019 RC and 2019 DH campaigns;
- 43% of the drill holes encountered new voids that were not modelled in the current void model;
- 57% were reported as expected or possibly expanded existing structures, and 43% were new voids not modelled in the most recent void model (Figure 12.7);
    - There are various possibilities: Post structural failures of the roof or hanging wall of the underground excavation due to lack of ground support reinforcement or adequate backfill;
    - Inaccurate historical mapping used in creating the void model, may have understated the size of the underground openings;
    - Following the closure of the historic mines, there are reports of miners venturing into the mines to create new developments to mine out resources that were left behind;
- The void model is accurately constructed as the expected measured (M_Expected) has an average difference of 1.8 m to the Void Depth as shown in Figure 12.8 below. Figure 12.9 below shows the difference regarding the new voids with an average difference of 34 m as you would expect.

GMS notes that the majority of new voids were intersected in the near-surface 2018 and 2019 RCGC drilling campaigns, and therefore discovered new voids that could not be defined by the wider-spaced DDH drilling. The deeper 2019 diamond drilling campaign confirmed the robustness of the existing void model.

Figure 12.7: Mined-out Voids Status as Intercepted by RC and DDH Drilling Programs

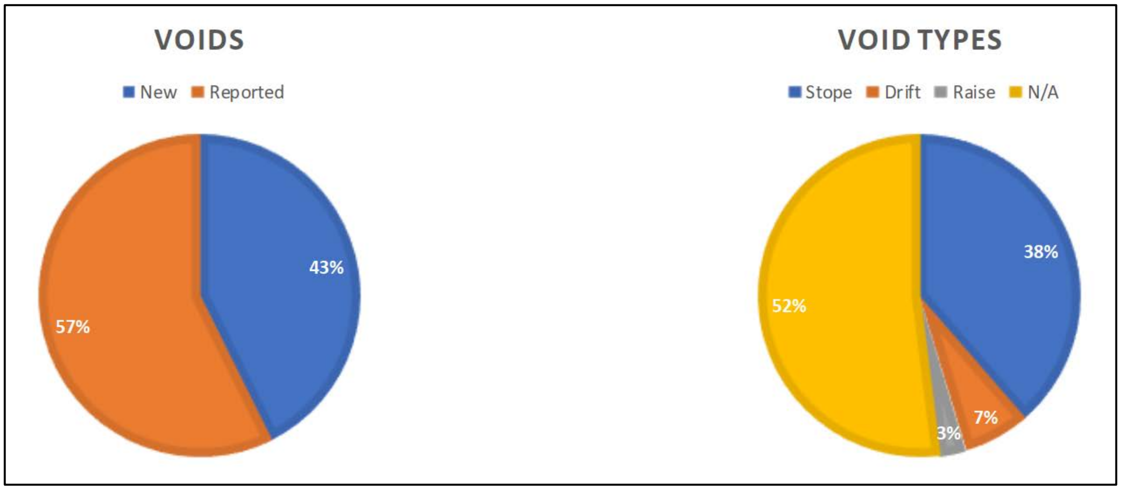


Figure 12.8: Reported Voids: Expected Length vs. Actual Length

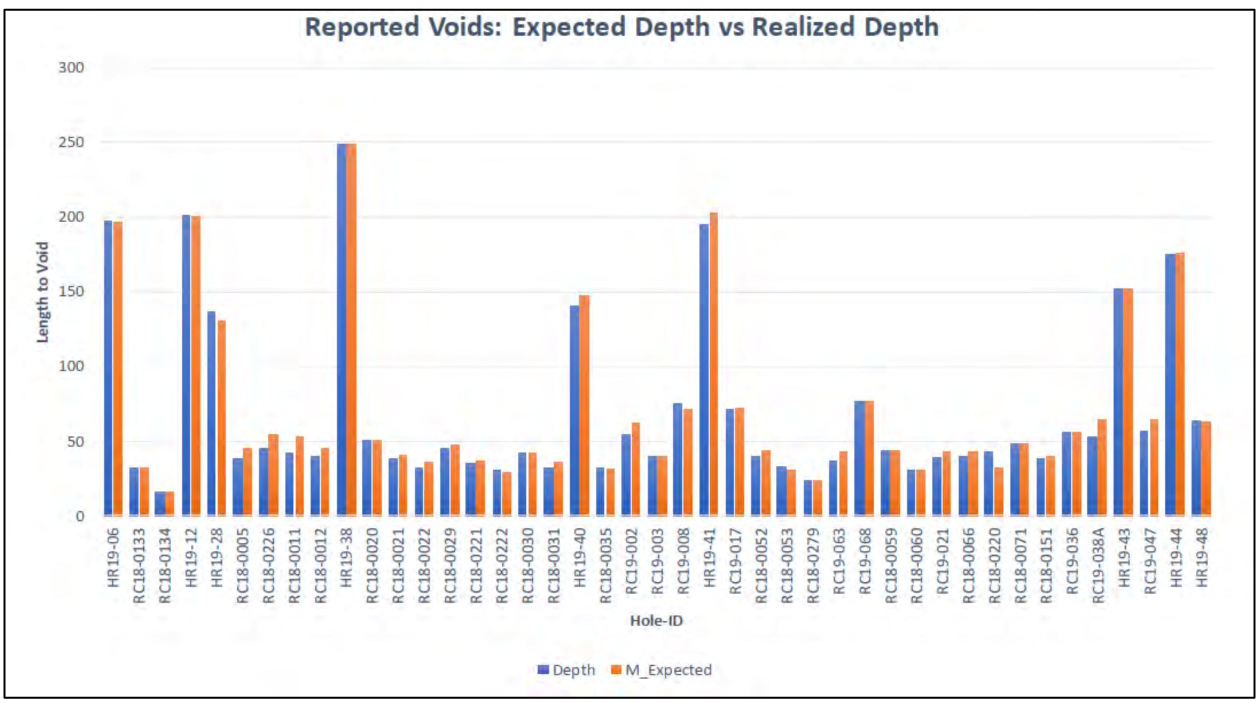
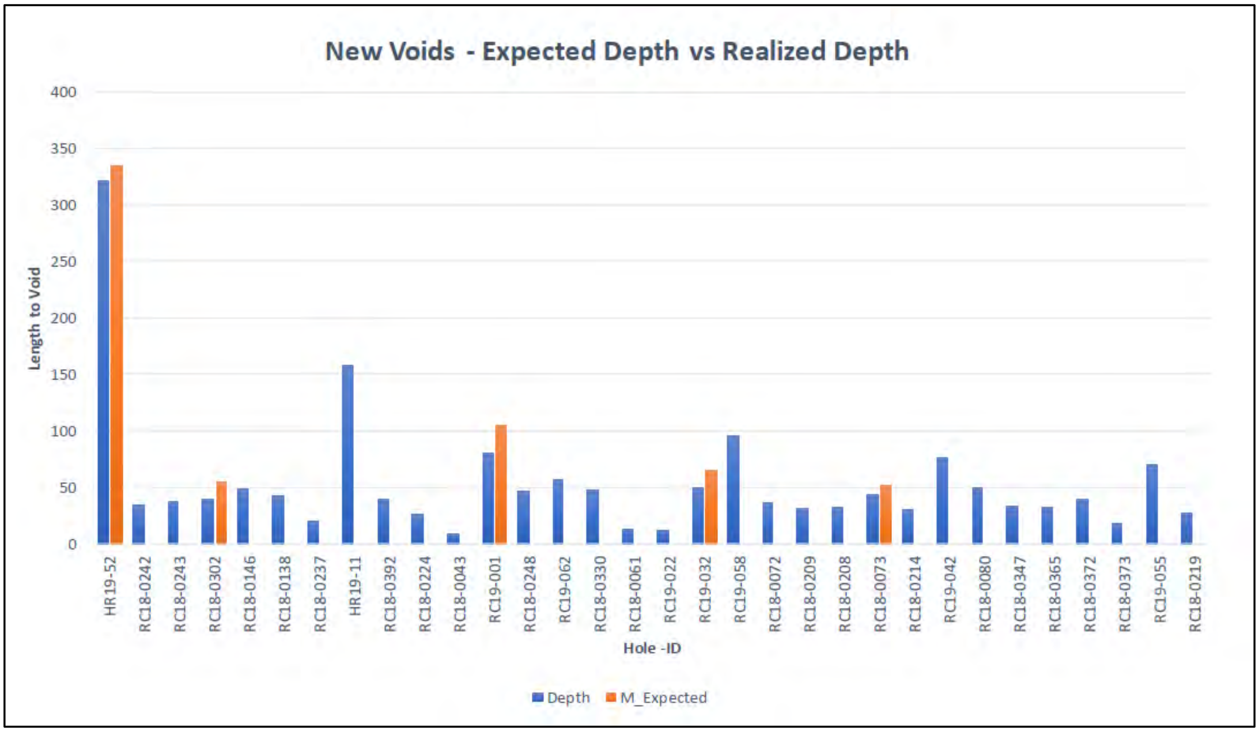


Figure 12.9: New Voids: Expected Length vs. Actual Length



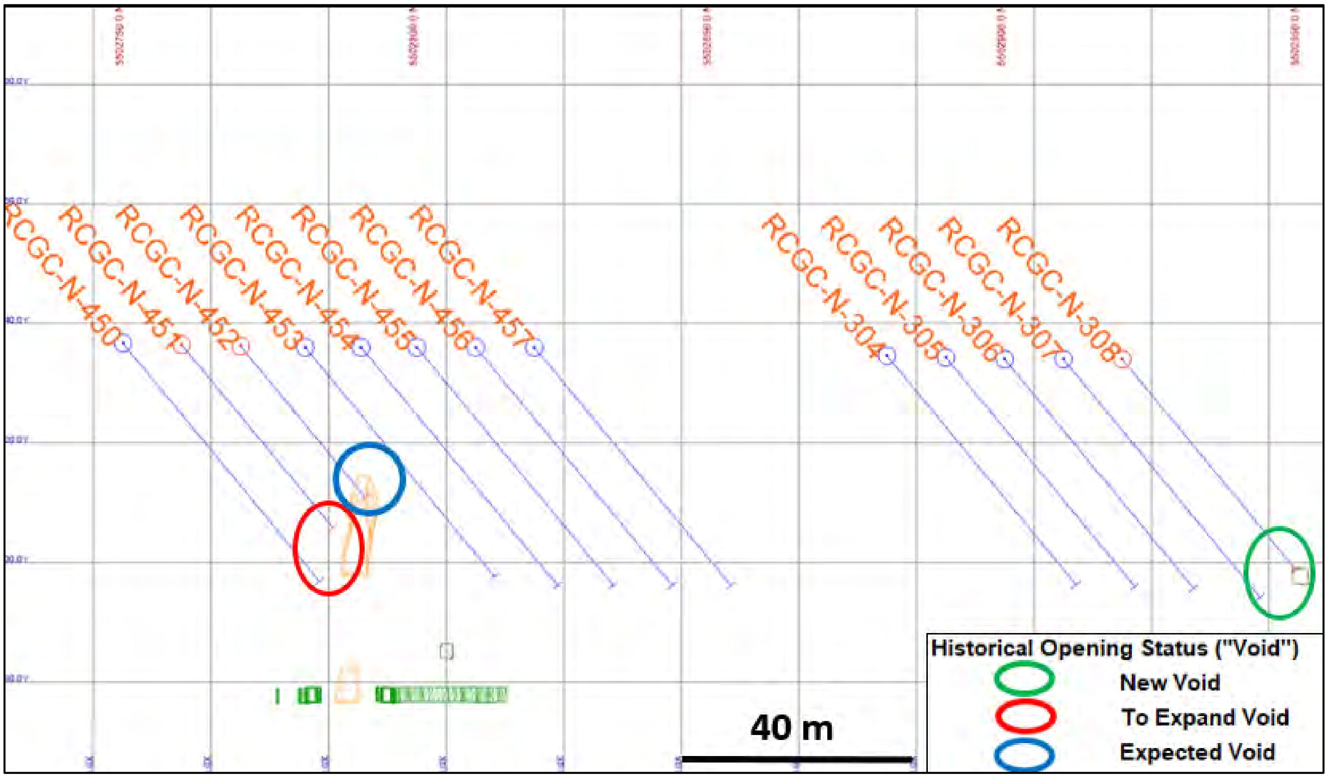
12.1.2.2.2 <u>2019 Voids Model Update in Hardrock Gems Project</u>

On May 6, 2019, GMS received the void intercepts from the 2018 and 2019 drilling programs.

GMS' observations regarding the drill hole intercepts vs. the historical UG workings are as follows:

- For 2018 RC grade control drilling, a total of 47 RC holes intersected historical openings (or Mined-out "voids"). Some 26 RC holes of 47 are classified as *New voids* since the intercepted interval was not expected (Figure 12.10);
- For 2019 RC drilling, a total of 22 holes have crossed over historical openings. 12 RC holes of 22 are now classified as *New voids* and 6 RC holes out of 22 are classified as *To expand* since the intercepted interval had a different depth compare to the expected depth in GGM database;
- 2019 diamond drilling has a total of 12 DDH that have intersected historical workings. Only 1 DDH has been classified as *To expand* because this opening has not intersected the historical void as expected. All the other DDH have intercepted a nearby historical opening.

Figure 12.10: 2018 RC Holes vs. Historical Openings – Section 4805E (Looking West)



Further iterations should be undertaken to improve the overall accuracy of the historical UG workings. Several drill holes encountered voids in proximity of a known stope or drift, indicating that adjustments could be made to the existing interpretation, but they are not material to the overall volume of the voids at the Project.

Regarding updating the void model with the recent drilling, GMS found that the voids were generally intersected as expected in the 2019 diamond drilling campaign (±5 m), so no adjustment was applied to the existing wireframes. New voids intersected and recorded in the RCGC drilling were modelled as single 10 m-long cuboids, to ensure that a representative volume of the UG voids was removed from the 2019 MRE.

12.1.3 Data Validation Conclusions and Recommendations – Hardrock Deposit

Overall, GMS is of the opinion that GGM's protocols for drilling, sampling, analysis, security, and database management meet industry standard practices. The 2019 data verification process demonstrated the validity of the data and protocols for the Hardrock Project. GMS considers the GGM database to be valid and of sufficient quality to be used for the Mineral Resource estimation.

Considering the current stage of the project, GMS believes that optimizing and updating the void model should be undertaken before mining commences to reduce operational H&S risks associated with historical stopes and other UG workings.

12.2 Brookbank, Kailey and Key Lake Data Verification

12.2.1 Recent Site Visit

For the 2020 update of the Brookbank, Kailey and Key Lake mineral resources, James Purchase, P. Geo., of GMS conducted a site visit to the three properties from July 27th to 30th, 2020. The following activities were undertaken:

- Review of core storage and sampling facilities;
- Verification of drill hole collars for each deposit;
- Examination of drill core/visual verification of mineralized intercepts;
- Comparison between analytical results by comparing assays with drill core intercepts;
- Review of geological models in Leapfrog GEO™ software;
- Review of drilling databases, assay certificates and QA/QC protocols.

GMS were able to locate collars for the drill holes identified in Table 12.4 The collars were found to be within +-5 m accuracy of the database coordinates when using a handheld GPS. See Figure 12.11 for an image of the Key Lake Property and a drill collar found on the Kailey Property.

Table 12.4: List of Drill Hole Collars Identified in the Field at the Brookbank, Kailey and Key Lake Deposits

List of Drill Holes					
11-85	11-86	11-87	PLL 08-32	PLL 08-30	PLL 08–13
16-BB-001	B99-04	B99-03			

Figure 12.11: Left: Mine Workings and the Capped Jellicoe Shaft, Key Lake Property. Right: Drill Hole Collar PLL 08-32 at the Kailey Property.

During this process, a single drill hole at Kailey (PLL-08-013) was identified as having conflicting survey data. This hole was subsequently ignored from the MRE as GMS could not confirm the true downhole dip and azimuth of the downhole surveys.

GMS visited the core storage facilities at the Brookbank Property and the Magnet Core Farm, and we conclude that the core storage and sampling facilities on-site are adequate for the processing of drill core. The QAQC protocol in place is closely adhered to and meets industry standards.

GMS found that core recovery was generally excellent, and that half-core sampling had been used for all drilling intervals inspected.

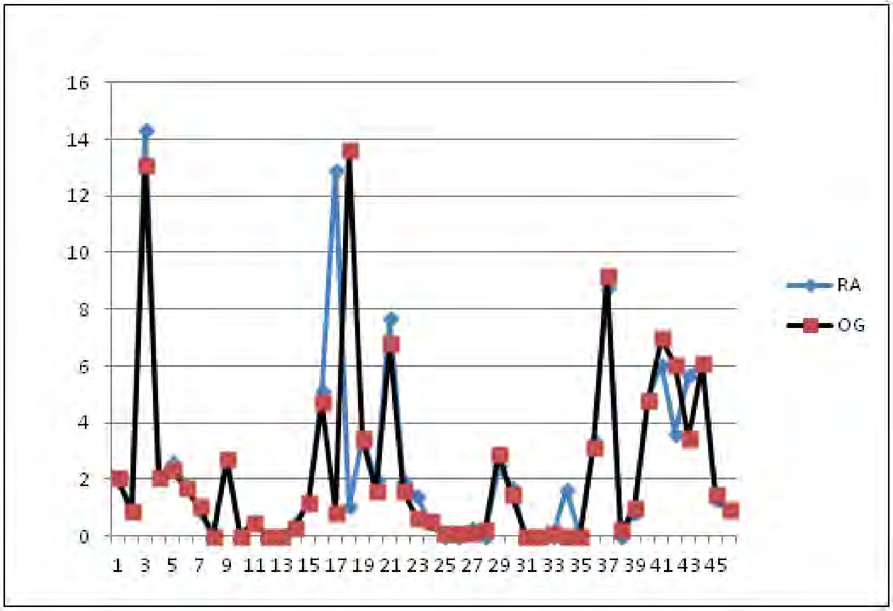
12.2.2 Independent Repeat Analyses

This section has been derived from the 2016 NI 43-101 Technical Report for the Hardrock Property, and outlines independent sampling undertaken by Micon International in 2012.

Micon selected 46 sample pulps encompassing a wide range of assay values (from low through medium to high) and re-numbered them in a different sequence before submitting them to Actlabs Geraldton for repeat analyses using the same method previously used.

Comparisons between original ("OG") and repeat assays ("RA") Figure 12.12 confirm the laboratory's high degree of accuracy (lack of bias) and precision with the exception of one mismatch. This mismatch is attributed to mistaken sample switch.

Figure 12.12: Comparison of Original (OG) and Repeat Analyses (RA)



12.2.3 <u>Database Validation</u>

Since the 2012 MRE, there has been a significant effort by GGM staff at increasing the confidence of the database for the three properties. Historical collars have been resurveyed, and assay certificates have been checked, organized and incorporated into the database. Most of the focus has been on the Brookbank Property, where a number of errors have been corrected in relation to erroneous surveys, missing sampling intervals and minor inconsistencies between assay certificates and database assays. An extensive resampling campaign was conducted between 2016 and 2017 (6,923 samples) to complete previously unsampled intervals and to confirm existing values in the database.

The resource database validation conducted by GMS involved the following steps:

- Checking for any non-conforming assay information such as duplicate samples and missing sample numbers;
- Verifying collar elevations against topography;
- Verifying the dip and azimuth against survey information for each hole;
- Comparing the database assays and intervals against the original assay certificates and drill logs.

The Key Lake Deposit (and to a certain extent the Brookbank Deposit) suffers from selective, incomplete sampling of drill core which has likely resulted in intervals of low-grade mineralisation remaining unsampled. In all such zones, GMS has assigned a detection limit assay value of 0.001 g Au/t. This will likely result in an understatement of the ounces, but is industry standard approach when faced with under-sampled drill core.

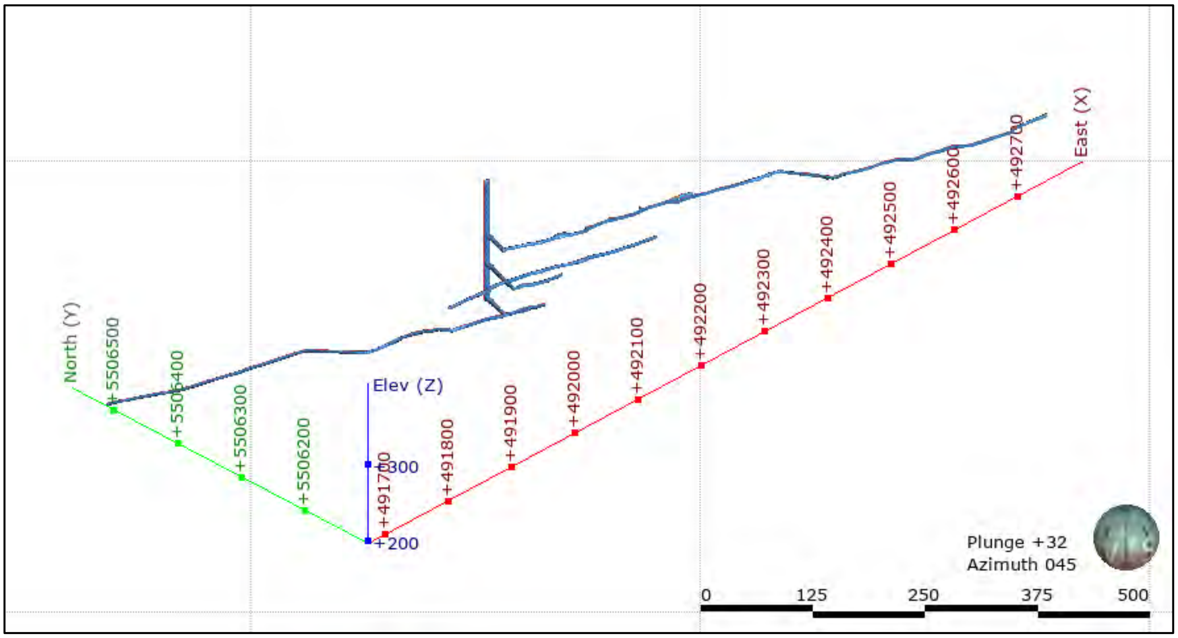
In addition, GMS has ignored drilling intervals where underground workings were intersected (flagged as “breakthroughs” in the lithology logging), rather than applied a 0.001 g/t grade.

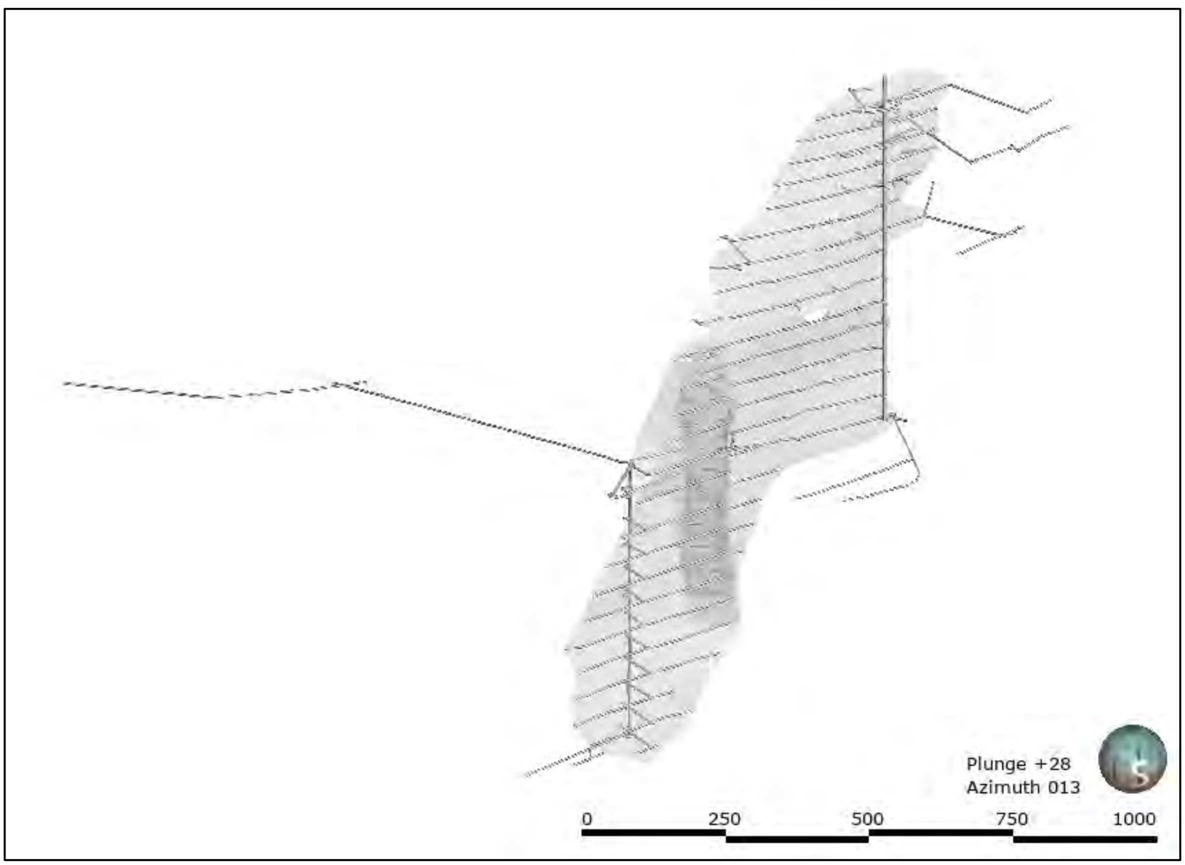
12.2.4 Underground Void Models

Historical plans and sections of underground workings have been recently digitized for the Kailey and Key Lake Deposits, which have resulted in a 3D void model that can be incorporated into the 2020 MRE update. For Key Lake, the Jellicoe shaft collar was surveyed and all plans and sections were digitised and pinned to the collar coordinate. At Kailey, underground drifts were digitised from level plans, and the mined vein was also modelled from a long-section assuming a constant width of 1 m. Unfortunately, no stope information was available to incorporate into the block model, however GMS did subtract the mined vein from the MRE. The void models are shown in Figure 12.13.

No underground voids are present at the Brookbank Deposit as no past production has taken place.

Figure 12.13: Top: Jellicoe Shaft and Underground Drives at Key Lake, Bottom: Little Long Lac (Kailey) Shaft and Underground Drifts, with the Mined Vein Wireframe





12.2.5 Data Verification Conclusions and Recommendations – Brookbank, Kailey and Key Lake

Considering the data verification that has been undertaken on the three properties by GMS, and previously by Micon and Scott Wilson, GMS believes that database generated by GGM is suitable for use in Mineral Resource estimation. A significant effort has been undertaken since 2012 to increase the confidence in the drilling database, and the current DataShed database is comprehensive in tracking drilling with validated collar, surveys and assay certificates.

The insertion of a detection limit value of 0.001 g Au/t for a missing assay may likely lead to an understatement of the resource grade at Key Lake, but nonetheless it ensures that all intercepts are used in the estimate and no over-extrapolation of grades occur into unmineralized areas.

12.3 Data Verification for Tailings Management Facility

The site visit revealed the necessity for field investigations to characterize the sub-surface conditions along the Tailings Management Facility (“TMF”) dam footprint. Geotechnical investigations were undertaken during 2014, 2015, 2016, 2018 and 2019 which included test pitting, borehole drilling, cone penetration test and field vane shear tests. Disturbed and undisturbed samples were extracted and tested for various index, strength parameters in an approved laboratory. The stratigraphic and strength information of the subsurface units were reviewed, interpreted and utilized in the design of the TMF dams.

Extensive geotechnical investigations in the TMF dam footprint have facilitated the assessment of design parameters required for the TMF dam design.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

This section summarizes all the relevant test work performed on the Hardrock Deposit. It is divided into three sub-sections: test work completed prior to the Feasibility Study ("FS"), test work performed during the FS and test work completed during the detailed engineering phase. Only recent reports (from 2011 onwards) are summarized in this section.

13.1 Previous Test Work

Some mineralogy, grindability and gold recovery test work was performed prior to the start of the FS. The key reports from 2011 to 2013 are summarized in this section. The reference documents are listed below:

- SGS Lakefield Research Limited, *An Investigation into Gold Recovery from Hardrock Project Ore, Final Report-12400-001*, March 1, 2011;
- SGS Canada Inc., *The Recovery of Gold from the Hardrock Project – Phase 2 Samples, Final Report-12400-002* -, December 11, 2012;
- McClelland Laboratories, Inc., *Whole Ore Cyanidation Testing - Project AF Drill Hole Reject Composites, MLI Job No. 3817*, September 24, 2013;
- SGS Canada Inc., *QEM Automated Rapid Mineral Scan,* Report 14117-001 – MI6000-OCT13 -, October 31, 2013.

13.1.1 Gold Recovery Test Work at SGS Lakefield (Phase 1)

Samples were sent to SGS Lakefield Research Limited in March of 2010. Two composites were prepared and were subjected to head analyses, mineralogy, Bond Work Index determination, gravity separation, gravity tailings flotation and whole ore, gravity tailings and flotation concentrate cyanidation.

13.1.1.1 Head Assays

Composites 1 and 2 were submitted for gold analysis according to the metallic sieve protocol. Two, 1 kg samples of each composite were submitted for coarse gold analysis (+/- 106 µm or 150 mesh fractions). The fine fraction was assayed in duplicate (analysis "a" and "b"). The results are presented in Table 13.1.

Table 13.1: Gold Head Analyses by Metallic Sieve

Sample	Head Grade Au (g/t)	+ 106 µm			- 106 µm Au (g/t)		
		Distribution		Au			
		Mass (%)	Au (%)	(g/t)	Mean	a	b
Composite 1 (A)	3.98	3.03	6.01	7.90	3.86	3.99	3.73
Composite 1 (B)	3.92	2.22	4.37	7.72	3.84	3.62	4.05
Composite 2 (A)	3.42	2.57	9.09	12.1	3.20	3.23	3.16
Composite 2 (B)	3.13	1.77	1.92	3.41	3.13	3.21	3.05

The variations in gold content in the coarse fraction between Composite A and Composite B and in gold content in the coarse fraction of Composite 2 (A) and Composite 2 (B) suggests the presence of fine free gold.

13.1.1.2 Mineralogy

A sample of each composite underwent an analysis of the rock-forming components using light microscopy, XRD, chemical analysis and SEM techniques. Table 13.2 lists Composite 1 and Composite 2 constituents. Other trace constituents include Fe-Ti oxides, amphibole, apatite and other sulphides.

Table 13.2: Constituents of Composite 1 and Composite 2

Sample	Composite 1 (wt%)	Composite 2 (wt%)
Quartz	26.2	32.5
Plagioclase	24.4	8.3
Ankerite	11.2	6.2
Chlorite	10.4	5.6
Muscovite	9.8	6.9
Pyrite	4.7	6.9
Clays	2.8	2.3
Biotite	2.7	1.8
Iron Oxides	1.8	18.8
Arsenopyrite	1.2	0.4
Siderite	1.2	7.2
Calcite	1.0	0.1
Pyrrhotite	0.7	1.8

13.1.1.3 Grindability Test Work

A standard Bond ball mill grindability test was completed on each composite (closing screen size of 150 µm). The results are shown in Table 13.3. Composite 1 falls into the moderately hard category while Composite 2 can be considered of medium hardness according to SGS Lakefield’s database.

Table 13.3: Composites 1 and 2 Bond Ball Mill Grindability Tests Results

Sample	Work Index (kWh/t)	Hardness Percentile
Composite 1	16.0	65
Composite 2	14.6	51

13.1.1.4 Gravity Separation

Gravity separation tests, including a Knelson Concentrator and a Mozley table, were performed to examine the amenability of the ore to gravity concentration and produce gravity tailings for cyanidation and flotation tests.

The effect of grind size was not investigated in these tests as all the test feeds were approximately 80% passing 100 µm. Gravity gold recovery ranged from 11.3% to 23.6% for Composite 1 and between 9.2% and 16.1% for Composite 2.

13.1.1.5 Flotation

The gravity tailings were subjected to flotation testing. The objective of the initial kinetic rougher flotation tests was to evaluate the impact of grind size on gold recovery and determine the test conditions required to generate bulk concentrate for further test work. The purpose of the tests was to recover gold in a sulphide rougher concentrate.

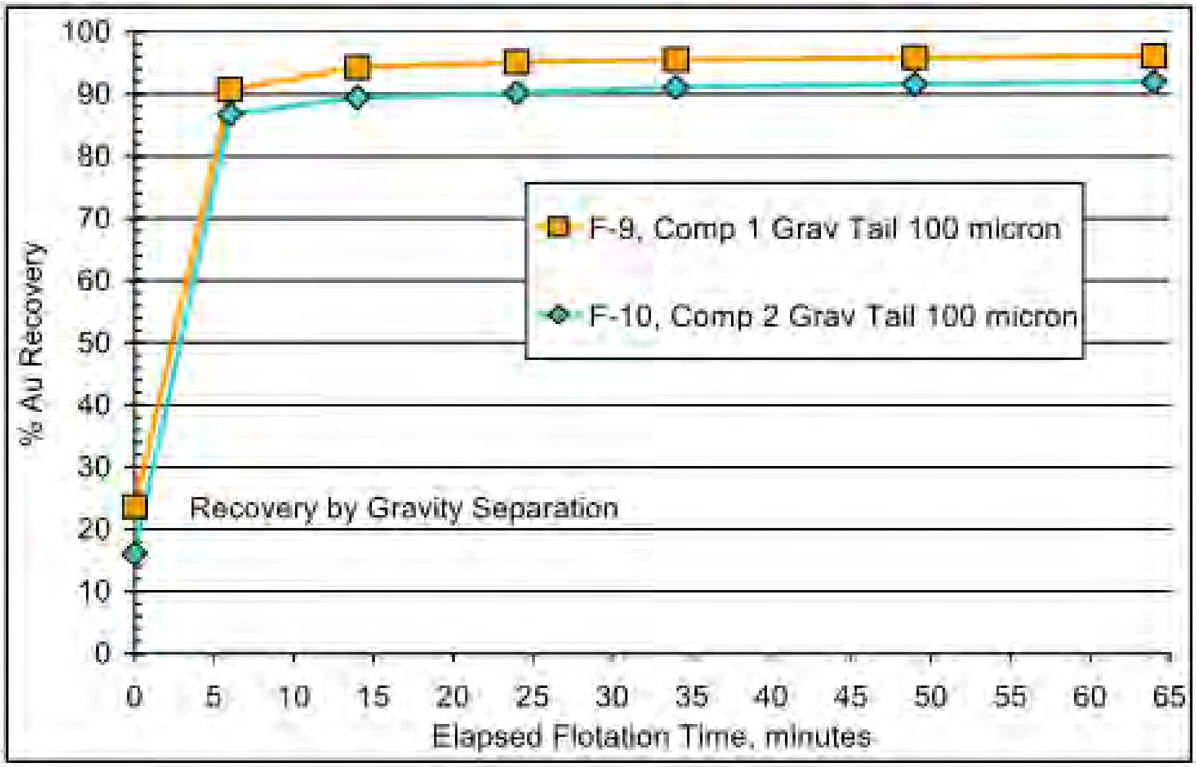
Flotation tests were carried out at grinds of 47, 70 and 95 µm (P80) for Composite 1. A concentrate mass recovery of 17% to 41% was achieved with gold grades ranging from 16.4 g Au/t at 95% overall recovery (coarsest grind) to 7.3 g Au/t at 98% overall recovery (finest grind). The tailings gold grade ranged from 0.19 g Au/t (P80 of 95 µm) to 0.15 g Au/t (P80 of 47 µm).

For Composite 2, flotation tests were performed at P80's of 51, 75 and 108 µm. Approximately 23% to 33% mass was recovered to the concentrate. Concentrate gold grades ranged from 11.3 g Au/t at 93% overall recovery (P80 of 108 µm) to 9.1 g Au/t and 95% overall recovery (P80 of 51 µm). The tailings gold grade was 0.22 g Au/t for the 51 µm sample and 0.29 g Au/t for the 108 µm sample.

Bulk flotation tests were conducted to generate concentrate for cyanidation. The results of the 10 kg bulk tests on the P80 of 100 µm gravity tailings were comparable to the 2 kg flotation tests on similar feed. The correlation between gold (non-gravity recoverable) and sulphide sulfur recovery indicates an association.

Figure 13.1 presents the results of the bulk flotation tests on Composite 1 and Composite 2.

Figure 13.1: Gravity Tailings Bulk Rougher Kinetics Results



Title: Gravity Tailing Bulk Rougher Kinetics Results	Author: SGS Lakefield Date: 11/03/01

13.1.1.6 Cyanidation

Whole ore cyanidation tests were conducted to examine cyanide leach amenability. The effect of particle size on gold extraction was also investigated. Bottle roll tests were completed at three grind sizes. Gold extraction ranged from 69% to 79% for Composite 1 and from 81% to 84% for Composite 2. Increased extraction with fine grinding was pronounced for feed P80's larger than approximately 70 µm. Below this size, the gain in recovery was less significant.

Gravity tailings cyanidation tests aimed to determine the ore amenability to cyanide leaching but also examined the effect of regrind on gold extraction. Bottle roll tests were performed under the same conditions as the whole ore tests. Cyanide extraction ranged from 64% to 70% for Composite 1 and between 75% and 82% for Composite 2. As observed in the previous test, regrind fineness was also less beneficial for regrinds below 70 µm. Combined gold recovery from gravity concentration and cyanidation was approximately 68% to 73% for Composite 1 and 78% to 83% for Composite 2.

Bottle roll tests were carried out on flotation concentrates and on reground concentrates (P80 of 10 µm). Gold extraction by cyanidation increased with finer grinding for both composites. For Composite 1, recovery increased from 60.1% to 67.3% at 11 µm. For Composite 2, recovery increased from 77.7% to 87.2% at 11 µm. Combined with the gravity recovery, the overall gold recovery with reground flotation concentrate reached 72.2% for Composite 1 and 81.9% for Composite 2.

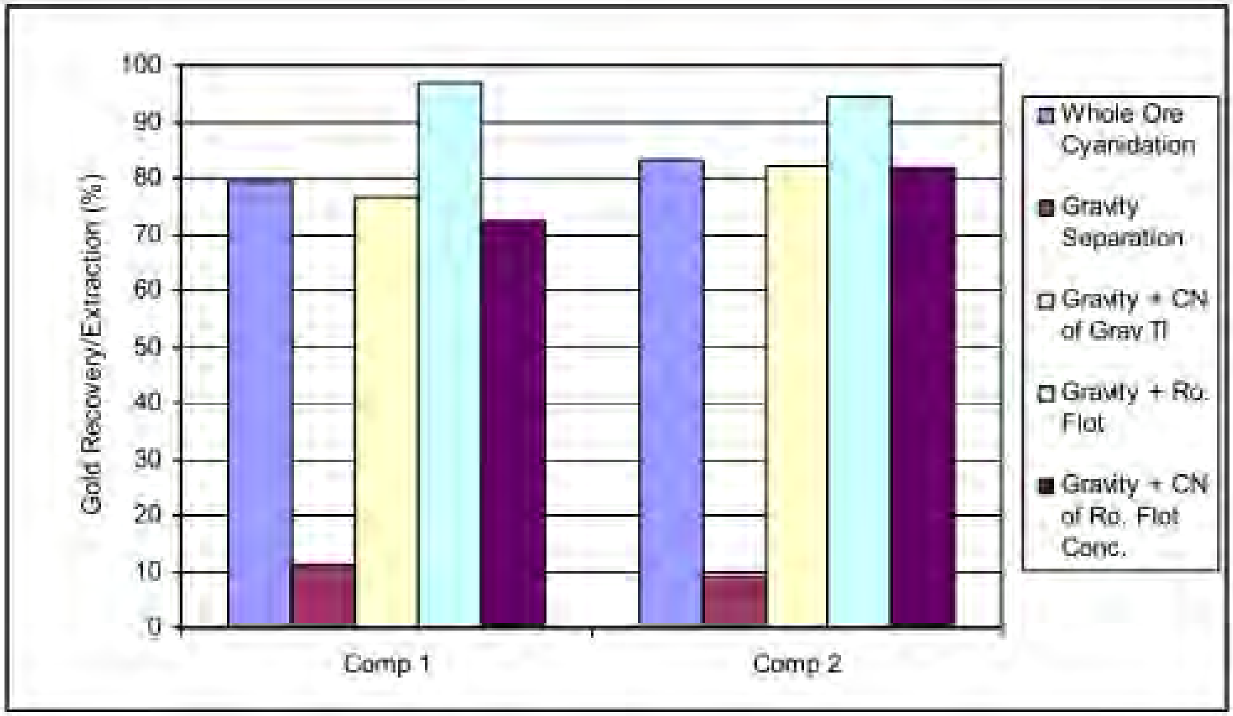
Table 13.4 below summarizes the whole ore, gravity tailings and flotation concentrate cyanidation tests results. Although recoveries vary for each process, the final tailings gold grades and calculated head grades are similar for both the Composite 1 and Composite 2 tests series.

Figure 13.2 presents the results of the combined methods.

Table 13.4: Cyanidation of Whole Ore, Gravity Tailings and Flotation Concentration

Test	Feed	CN Leach				Combined Recovery (%)	Final Tailings (g Au/t)
		Reagent Consumption (kg/t of Whole Ore)		Grind P80 (µm)	Recovery (%)		
		NaCN	CaO				
CN-2	Composite 1 Whole Ore	1.5	0.4	59	79.2		1.01
CN-8	Composite 1 Gravity Tail	0.8	0.5	68	62.1	73.3	0.99
CN-14	Composite 1 Flot. Conc.	0.9	0.7	11	48.6	72.3	1.04
CN-5	Composite 2 Whole Ore	1.3	0.5	66	83.2		0.51
CN-11	Composite 2 Gravity Tail	0.8	0.8	68	72.9	82.1	0.53
CN-16	Composite 2 Flot. Conc.	1.0	1.0	11	65.8	81.9	0.52

Figure 13.2: Comparison of Combined Results



Title: Comparison of Combined Results	Author: SGS Lakefield Date: 11/03/01

Gravity separation followed by gravity tailings cyanidation achieved similar results to whole ore cyanidation. Gravity separation followed by flotation yielded the highest recoveries but assumes that the flotation concentrate can be sold as smelter feed.

13.1.2 Gold Recovery Test Work at SGS Lakefield (Phase 2)

Samples were sent to SGS Lakefield in May 2011. This second phase of work followed the previous test work campaign completed on Composite 1 and Composite 2. For Phase 2, two new composites were prepared (Composite IF1 and Composite P2). Composites IF1 and P2 were subjected to gold deportment by mineralogy analysis, Bond grindability testing, gravity recoverable gold determination and whole ore flotation evaluation. Moreover, gravity tailings flotation and flotation rougher concentrate cyanidation tests were included in the program.

13.1.2.1 Head Assays

Composites IF1 and P2 were submitted for gold analysis according to the metallic sieve protocol. Two 1 kg samples of each composite were submitted for coarse gold analysis (+/- 106 µm or 150 mesh fractions). The fine fraction was assayed in duplicate (analysis “a” and “b”). The results are presented in Table 13.5.

Table 13.5: Gold Head Analyses by Metallic Sieve

Sample	Head Grade (g Au/t)	+ 106 µm			- 106 µm (g Au/t)		
		Distribution		Au (g/t)			
		Mass (%)	Au (%)		Mean	a	b
Composite IF1 (A)	4.53	1.96	3.86	8.91	4.44	4.32	4.57
Composite IF1 (B)	4.66	1.91	3.11	7.58	4.60	4.62	4.58
Composite P2 (A)	6.02	2.39	5.77	14.5	5.81	5.70	5.92
Composite P2 (B)	5.37	2.87	5.91	11.0	5.20	4.98	5.43

The variations in gold content in the coarse fraction between Composite IF1 and Composite P2 and in gold content in the coarse fraction of IF1-A and IF1-B and between P2-A and P2-B suggests the presence of fine free gold.

13.1.2.2 Mineralogy

A sample of each composite underwent a gold deportment study to provide the mode and occurrence of the microscopic gold. The gold chemical composition was analyzed using SEM-EDS (Scanning Electron Microscopy-Energy Dispersive Spectroscopy). The major gold mineral in the samples was native gold. In Composite IF1, the overall gold was 22% liberated, 20% attached and 58% locked. In Composite P2, 28% of the gold was liberated, 31% was attached and 41% was locked. The study determined that gold in the samples could effectively be recovered by gravity methods.

13.1.2.3 Grindability Test Work

A standard Bond rod mill grindability test (closing screen size of 14 mesh / 1,180 µm) was completed on a separate sample made by combining ore from three zones. The composite was also subjected to a standard Bond ball mill grindability test (closing screen size of 150 µm).

With a rod mill work index ("RWI") of 17.3 kWh/t and a ball mill work index ("BWI") of 16.5 kWh/t, the sample can be considered moderately hard with respect to both parameters according to SGS Lakefield's databases.

13.1.2.4 Gravity Recoverable Gold

A gravity recoverable gold ("GRG") test was performed on a sample from each Composite IF1 and P2. The GRG test estimates the maximum amount of gold that can be recovered by gravity. Plant recoveries are typically lower.

For IF1, it was found that 8% of the gold could be recovered to a gravity concentrate at a grinding product size of 570 µm (P80). A 14% recovery is reached at 241 µm and 24% at 60 µm. For P2, 9% could be recovered at 570 µm, 17% at 267 µm and 31% at 106 µm.

13.1.2.5 Gravity Separation

Gravity separation tests (Knelson/Mozley) were performed on IF1 and P2 to produce gravity tailings for flotation tests, for bulk flotation tests followed by concentrate cyanidation and for a cyanidation test. Gold recovery varied from 38% to 39% for IF1 and between 17% and 40% for P2. The grind sizes for all five tests ranged from 80 to 101 µm.

13.1.2.6 Flotation

Whole ore flotation tests were carried out to evaluate the effect of rougher concentrate cleaning on overall concentrate mass reduction and final concentrate gold grade and recovery.

For Composite IF1, the cleaning stages reduced the second cleaner concentrate mass to 12% with a grade of 30.3 g Au/t and an 85% recovery. After regrinding to 45 µm (P80), the mass pull was 14%, the gold grade was 27.0 g Au/t and the recovery was increased to 89%.

For Composite P2, the second cleaner concentrate showed 7% mass pull, a grade of 54.8 g Au/t and an 81% gold recovery. With regrinding to 25 µm, the mass pull increased slightly to 8.5%, the grade and gold recovery were higher at 56.8 g/t and 88%. A locked-cycle test was undertaken on Composite P2. An average grade of 27.2 g Au/t and 22.0% sulfide was achieved with a 92.5% gold recovery and a 94.8% sulfide recovery.

The cleaner flotation tests on gravity tailings also demonstrated the material could be effectively cleaned. Similar gold grade and recovery were achieved using gravity tailings as with whole ore. The results of the whole ore and gravity tailings cleaner flotation tests are summarized in Table 13.6 and the locked-cycle test projected results are presented in Table 13.7.

Table 13.6: Whole Ore and Gravity Tailings Cleaner Flotation Tests

Sample		Test No.	Regrind P_{80} (µm)	2nd Cleaner Concentrate			Gravity Conc. Grade Au (g/t)
				Recovery (wt %)	Grade Au (g/t)	Recovery (overall) Au (%)	
Composite IF1	Whole Ore	F1 F3	n/a 45	12.5 13.8	30.3 27.6	84.6 88.7	n/a n/a
	Gravity Tailing	F5 F6 F7	n/a 23 10	8.5 7.9 8.1	28.8 28.7 31.8	89.1 88.6 91.3	8.32
Composite P2	Whole Ore	F2 F4	n/a 22	7.3 8.5	54.8 56.8	81.3 87.8	n/a n/a
	Gravity Tailing	F8 F9 F10	n/a 19 9	5.9 4.9 4.8	48.9 54.2 58.6	88.6 87.4 86.2	1.21

Table 13.7: Locked-Cycle Metallurgical Projected Results

Product	Mass		Assay		Distribution	
	g	%	Au (g/t)	S^{2-} (%)	Au (%)	S^{2-} (%)
1st Cleaner Concentrate	1,464.9	18.3	27.2	22.0	92.5	94.8
1st Cleaner Scavenger Tails	1,291.1	16.1	0.72	1.14	2.2	4.3
Rougher Tails	5,247.0	65.6	0.44	0.06	5.4	0.9
Head	8,003.0	100.0	5.39	4.25	100.0	100.0

13.1.2.7 Pressure Oxidation

Assessment of pressure oxidation ("POX") as a pre-treatment to cyanidation was performed on a rougher concentrate sample generated from Composite 1 during the previous phase of the test work program. Only 70% of the sulfides were oxidized but it was sufficient to make the sample amenable to cyanide leaching. The results of the four tests showed that even at a coarse grind of 123 µm (P80), pressure oxidation increased gold extraction to 97% with a 94% overall gold recovery (including the flotation stage).

13.1.2.8 Cyanidation

Cyanide leach tests were performed on whole ore and flotation rougher concentrate samples. Standard bottle roll tests were conducted. The sodium cyanide concentration and aeration method were varied in the flotation concentrate cyanidation tests. The effect of regrind and lead nitrate were also evaluated.

The highest extractions were achieved at the finer grinds. A 10 µm grind resulted in a 98% extraction for IF1 while a 15 µm grind yielded 95% recovery for P2. However, cyanide consumption was also highest for these tests. The sodium cyanide concentration and aeration method did not impact gold extraction.

For Composite IF1, there was no benefit in including a flotation stage as 77% extraction was achieved after 72 hours of whole ore leaching. Cyanidation of rougher flotation concentrate achieved 75% overall recovery. For Composite P2, a flotation stage increased overall recovery to 87% compared to 75% after 72 hours of whole ore leaching. These results are summarized in Table 13.8.

Table 13.8: Whole Ore Cyanidation vs. Flotation Concentrate Cyanidation

Process	Test	Grind P_{80} (µm)	72h Au Extraction (%)	Recovery		Residue/Tailings Grade		CN Consumption (kg/t)	
				Flotation (%)	Overall (%)	CN Au* (g/t)	Overall Au (g/t)	CN Unit	Overall
IF1 Whole Ore	CN-18	93	76.5	-	76.5	1.00	1.00	4.01	4.01
IF1 Flot. + CN of Flot. Conc.	CN-17	93	78.5	95.4	74.9	2.87	1.21	8.70	2.98
P2 Whole Ore	CN-20	123	75.0	-	75.0	2.23	2.23	1.08	1.08
P2 Flot. + CN of Flot. Conc.	CN-19	123	94.3	92.2	86.9	0.89	0.63	4.60	1.37

*Note: *Average of duplicate residue assays*

13.1.3 Whole Ore Cyanidation Testing at McClelland

Drill hole reject composites (13) were sent to McClelland Laboratories to undergo whole ore cyanidation tests. The objectives of the program were to confirm previous testing results and to examine grind size and cyanide concentration impacts on whole ore leaching.

13.1.3.1 Results

The tests consisted of standard bottle roll tests with or without carbon addition. The direct head assays of the thirteen samples ranged from 0.40 g Au/t to 7.37 g Au/t with an average of 3.20 g Au/t. The cyanidation tests were performed on three different grind sizes (P80): 125, 75 and 37 µm. The summary of the tests performed at 75 µm and without carbon addition are presented in Table 13.9.

Table 13.9: Whole Ore Cyanidation Tests Results

Sample	C_{Org} (%)	S (%)	Au Recovery (%)	g Au/t Ore				Reagent Requirements kg/t Ore	
				Extracted	Tail	Calc'd Head	Head Assay	NaCN Cons.	Lime Added
EP134T-A	0.03	1.25	91.7	1.76	0.16	1.92	1.85	0.58	2.9
EP134T-B	<0.01	2.37	86.5	2.43	0.38	2.81	2.75	0.42	2.2
HR124	0.03	1.08	95.4	5.62	0.27	5.89	6.87	0.28	2.6
HR133-A	0.03	3.27	85.6	2.14	0.36	2.50	2.33	0.51	2.8
HR133-B	0.01	0.84	93.8	2.57	0.17	2.74	2.47	0.32	2.8
HR142	0.05	8.09	76.6	5.44	1.66	7.10	7.37	0.99	3.9
HR145-A	0.03	0.20	86.9	0.53	0.08	0.61	0.69	0.49	2.1
HR145-B	0.01	1.42	89.1	1.56	0.19	1.75	1.30	0.38	4.1
HR148	0.01	0.23	86.0	0.43	0.07	0.50	0.40	0.39	1.8
MM005T-A	0.01	2.17	92.6	2.13	0.17	2.30	2.50	0.38	2.1
MM005T-B	0.02	0.87	86.2	1.44	0.23	1.67	1.68	0.50	1.9
MM351-A	0.04	12.70	63.8	4.43	2.51	6.94	6.69	0.90	4.1
MM351-B	0.06	5.41	77.4	3.77	1.10	4.87	4.74	1.06	3.8

All thirteen composites were amenable to cyanidation under the tested conditions. Gold recovery was between 85% and 95% for the composites with low sulfide sulfur content (less than 2.5%). Three composites showed higher sulfide sulfur levels (5.4% to 12.7%) and yielded lower gold recoveries (63.8% to 77.4%). Cyanide consumption was also higher for these three samples.

Gold recovery increased with finer grind sizes (2.1% increase between 120 µm and 75 µm and 4.3% between 75 µm and 37 µm) but was not affected by cyanide concentration. Preg-robbing characteristics were not observed, and recoveries were similar with or without activated carbon. Gold leaching was complete in approximately eight hours and recovery rates were fast.

13.1.4 QEM Rapid Mineral Scan at SGS

A Global Composite sample was subjected to a QEM Rapid Mineral Scan at SGS Minerals in Lakefield, Ontario in October of 2013. The results are presented in Table 13.10.

Table 13.10: QEMSCAN Modals on Global Composite

Survey Project / LIMS Sample	Global Composite Mineral Mass (%)
Quartz	28.6
Plagioclase	19.3
Sericite/Muscovite	13.8
Chlorite	9.53
Ankerite	9.48
Magnetite	7.96
Pyrite	2.62
Biotite	1.72
Siderite	1.46
Calcite	1.33
Pyrrhotite	1.20
Hematite	0.69
Other Micas/Clays	0.60
Other Oxides	0.46
Other	0.42
K-Feldspar	0.39
Apatite	0.28
Arsenopyrite	0.11
Chalcopyrite	0.01

13.2 Feasibility Study Test Work

This section includes any test work program that was performed during the Preliminary Economic Assessment ("PEA") and during the FS. As the FS progressed, additional test work was initiated and is described here. Primarily, high pressure grinding rolls ("HPGR") tests were required to confirm the ore amenability for high pressure grinding, to select the equipment and estimate the operating costs. The key reports from 2014 and 2015 are summarized in this section. The reference documents are listed below:

- SGS Canada Inc., *An Investigation into the Grindability Characteristics of Samples from the Hardrock Deposit, Report 1 (Grindability)-14117-001*, August 26, 2014;
- SGS Canada Inc., *An Investigation into The Hardrock Deposit, Final Report-14117-001*, October 8, 2014;
- SGS Canada Inc., *The HPGR Amenability of Samples from The Hardrock Deposit, Report 2 – Rev 1- 14117-001*, March 6, 2015;
- JKTech Pty Ltd., *Revised SMC Test Report*, April 2014;
- FLSmidth, *Thickening and Rheology Tests on Gold Ore Composite*, June 2014.

13.2.1 Grindability Test Work

Dilution samples (5), PQ core samples (3) and core interval samples (53) were submitted for comminution testing at SGS Canada Inc. in Lakefield, Ontario. In addition, nine variability composites and one Global Composite sample were prepared using the core samples. The Global Composite is considered most representative of the run-of-mine during the project's life. The samples were submitted for JK drop-weight tests, SMC tests, Bond low-energy impact tests, Bond rod mill and ball mill grindability tests, ModBond tests and Bond abrasion tests.

13.2.1.1 Grindability Tests Results

The grindability tests results for the Composite samples, the PQ core samples and the Dilution samples are presented in Table 13.11.

Table 13.11: Composites, PQ Core and Dilution Samples Comminution Tests Results

Type	Name	Interval Number	CWI (kWh/t)	Relative Density	JK Parameters			BWI (kWh/t)	Mod Bond (kWh/t)	Ai (g)
					A	b	Axb			
Composites	Global							15.2		
	A							15.9		
	B							15.3		
	C							15.9		
	D							15.8		
	E							15.1		
	F							14.5		
	G							16.4		
	H							14.3		
	I							15.0		
PQ Core	PQ Iron Formation (DWT)		12.0	3.26	75.1	0.43	32.3			
	PQ Iron Formation (SMC)			3.24	84.1	0.40	33.6			
	PQ Greywacke (DWT)		10.2	3.26	59.6	0.76	45.3			
	PQ Greywacke (SMC)			3.11	75.7	0.54	40.9			
	PQ Porphyry with Minor Greywacke (DWT)		14.6	2.93	75.1	0.32	24.0			
	PQ Porphyry with Minor Greywacke (SMC)			2.76	76.3	0.34	25.9			
Dilution Samples	Greywacke			2.77	94.6	0.24	22.7	15.5	16.0	0.154
	Iron Formation			2.95	81.2	0.35	28.4	10.5	11.1	0.091

Type	Name	Interval Number	CWI (kWh/t)	Relative Density	JK Parameters			BWI (kWh/t)	Mod Bond (kWh/t)	Ai (g)
					A	b	Axb			
	Gabbro			2.78	65.7	0.48	31.5	14.5	14.8	0.102
	Porphyry			2.68	92.0	0.27	24.8	16.0	16.5	0.194
	Ultramafic			2.96	66.7	0.89	59.4	10.2	10.2	0.069

The results were computed for each lithology to calculate the 90th percentile values as presented in Table 13.12.

Table 13.12: Comminution Test Results per Lithologies

Samples	Mod Bond 90th percentile (kWh/t)	DWI 90th percentile
Greywacke (S3E) & Gabbro (I1A)	15.5	11.7
Iron formation (C2A)	15.5	12.3
Porphyry (I3P)	16.4	10.7
Overall	15.6	11.7

Fifty-three core interval samples, made of material from various lithologies representing the entire deposit, were submitted to comminution testing. The samples show little variability between them. The summary of the results is presented in Table 13.13.

Table 13.13: Core Interval Samples Comminution Tests Results

Description	JK Parameter		RWI (kWh/t)	BWI (kWh/t)	Mod Bond (kWh/t)
	Rel. Density	Axb			
Average	2.98	29.2	16.5	14.9	14.4
Std. Dev.	0.21	3.4	0.2	1.0	1.2
Rel. Std. Dev.	7	12	1	7	8
Minimum	2.71	41.0	16.3	13.2	11.3
Median	2.92	28.8	16.4	15.4	14.6
Maximum	3.35	24.1	16.8	16.0	16.5

In terms of resistance to impact breakage (Axb), the samples were found to be hard to very hard. Their abrasion resistance (t_a) fell into the very hard category. The Bond low-energy indices characterize the samples as medium to moderately hard.

The rod mill work indexes were all similar and fell into the moderately hard category. The ball mill work indexes ranged from soft to moderately hard. Finally, the abrasion indices denoted a mild to medium abrasive ore.

13.2.2 Characterization and Recovery Test Work

The samples used for the grindability tests were submitted to head grade determination, mineralogy, magnetic separation, flotation, gravity separation, cyanidation with cyanide destruction, carbon modelling, solid-liquid separation and environmental testing. The dilution samples were only assayed for direct head grade and were not submitted to any metallurgical test work. In addition, six low-grade composites and a master composite representing the lithological ratios for the first three years of operation were prepared and tested. The proportion of each lithology in the prepared samples is shown in Table 13.14.

Table 13.14: Global, Master, Variability and Low-Grade Samples Composition

Composite	Lithology Constitution (%)				
	Wacke to Greywacke S3E	Iron Formation C2A	Gabbro I1A	Porphyry I3P	Quartz-Feldspar-Porphyry I3R
Global	46.2	33.5	5.3	15.1	
Master	43.8	35.1	3.6		17.5
A	100				
B		55.8	11.4	32.8	
C	96.3	3.7			
D		72.0	28.0		
E	78.3		21.7		
F		100			
G				100	
H		100			
I	100				
S3E-0.5-WCE	100				
S3E-0.7-WCE	100				
I3P-0.5-WCE				100	
I3P-0.7-WCE				100	
C2A-0.5-WCE		100			
C2A-0.7-WCE		100			

13.2.2.1 Characterization and Recovery Tests Results

13.2.2.1.1 Head Grade Determination

Composite head grades were determined by metallic sieve analyses and a weighted average was calculated from the test work (Table 13.15). The direct and calculated head grades all correlate well except for Composite C and Composite F.

Table 13.15: Composite Samples Direct and Calculated Head Grade

Sample Name	Direct g Au/t	Calculated (From Test Work) g Au/t
Composites		
Global	1.74	1.92
Master	1.94	2.08
A	2.56	2.62
B	2.04	2.19
C	1.71	2.04
D	1.68	1.58
E	1.18	1.39
F	1.36	2.01
G	1.59	1.59
H	2.65	2.59
I	2.29	2.07
Dilution Samples		
Greywacke	0.06	-
Iron Formation	<0.01	-
Gabbro	0.08	-
Porphyry	0.06	-
Ultramafic	0.04	-
Low-Grade Composites		
S3E-0.5-WCE	0.55	0.50
S3E-0.7-WCE	0.67	0.72
I3P-0.5-WCE	0.46	0.49
I3P-0.7-WCE	0.75	0.67
C2A-0.5-WCE	0.34	0.38
C2A-0.7-WCE	0.85	0.82

13.2.2.1.2 Mineralogy

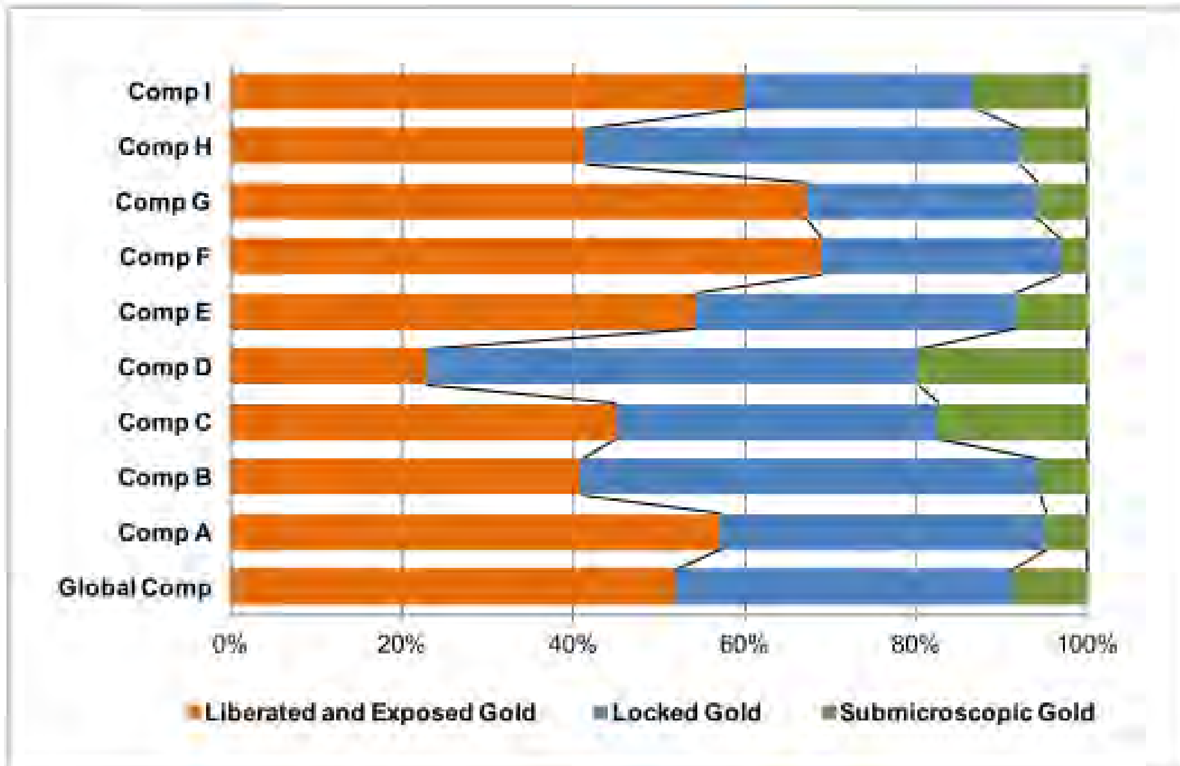
The Global and Variability composites were submitted to a microscopic (> 0.5 µm) and submicroscopic (< 0.5 µm) gold deportment study. The gold mineral association and distribution are presented in Table 13.16. The gold occurrence by distribution based on an approximate P80 of 300 µm is shown in Figure 13.3.

Table 13.16: Gold Deportment Results

Composite	Gold Distribution (%)		Gold Associated Minerals (% - Normalized to 100%)							
	Sub microscopic Au	Microscopic Au	Py	Apy	Py-Sul	FeOx	Py-Silc	Silc	Carb	Other
Global	8.6	91.4	75.8	7.75	8.94	2.97	4.16	-	-	0.33
A	4.8	95.2	58.6	14.3	5.69	3.14	14.9	-	1.38	1.96
B	5.7	94.4	58.4	4.66	1.33	8.24	20.8	1.85	1.38	3.43
C	17.4	82.6	83.4	1.43	0.64	3.81	7.52	-	2.85	0.36
D	19.7	81.0	78.7	4.58	-	13.6	-	0.58	2.27	0.28
E	8.3	92.7	34.3	-	-	17.2	23.8	22.4	-	2.25
F	3.2	96.9	74.9	3.42	-	10.6	4.62	3.53	-	2.93
G	5.7	94.3	90.3	5.38	0.99	1.18	0.59	-	1.42	0.19
H	7.8	92.2	87.9	2.54	-	0.92	0.72	7.93	-	-
I	13.2	86.8	5.45	12.61	-	-	0.23	80.79	0.43	0.51

Note: Py-pyrite (including greigite); Apy – arsenopyrite and with other sulphides; Py-Sul – Pyrite with other sulphides; FeOx – iron oxides; Py-Silc – pyrite with silicates; silc – Silicates; Carb – carbonate minerals and mixture

Figure 13.3: Gold Occurrence (by Distribution)



Title: Gold Occurrence (by Distribution)	Author: SGS Canada Inc. Date: 14/07/21

13.2.2.1.3 Magnetic Separation

Davis Tube testing was performed on the Global and Variability Composites to identify the presence of magnetic minerals. The results showed a large variation in the weight recovery to the concentrates: 0% for Composite G (100% Porphyry), up to 27% for Composite F (100% Iron Formation) and around 10% for the Global Composite. The Global Composite was also subjected to low intensity magnetic separation ("LIMS") and wet high intensity magnetic separation ("WHIMS") testing to evaluate the possibility of removing iron minerals without incurring gold losses. A LIMS stage was effective in removing significant amounts of iron but resulted in a 7.4% gold loss. The WHIMS stage did not significantly split the iron and gold distribution.

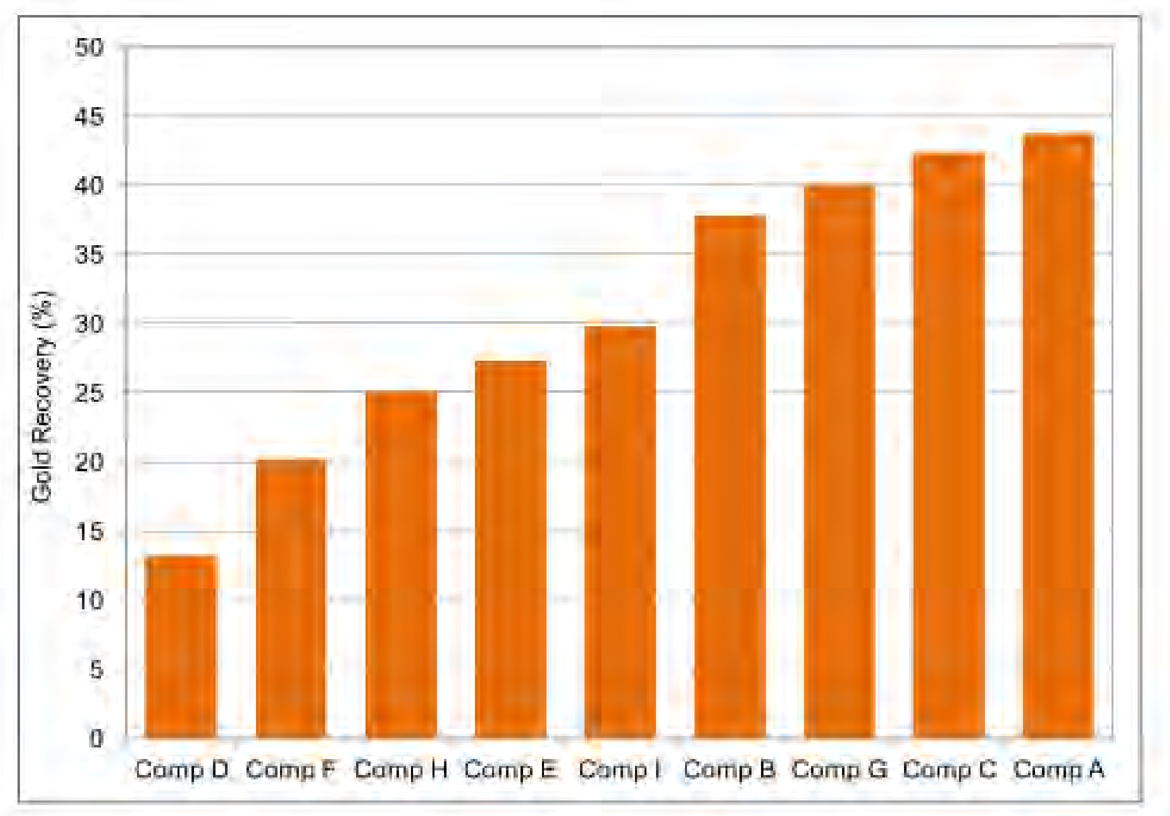
13.2.2.1.4 Gravity Recovery

All composites were subjected to gravity separation testing using a Knelson Concentrator and a Mozley table. Based on a series of gravity recovery tests completed at various grind sizes (P80), the Global

Composite recovery varied from 15% at 129 µm to 42% at 61 µm, the Master Composite recovery from 18% at 105 µm to 30% at 61 µm and the Low-Grade Composite recovery from 5% to 39% at 110 µm.

The Variability Composites were submitted for a single gravity separation test at a target grind of 80 µm (P80). The gold recovery varied from 13% to 44%. The results are shown in Figure 13.4.

Figure 13.4: Variability Composites Gravity Recovery Results



Title: Variability Composites Gravity Recovery	Author: SGS Canada Inc. Date: 14/10/08

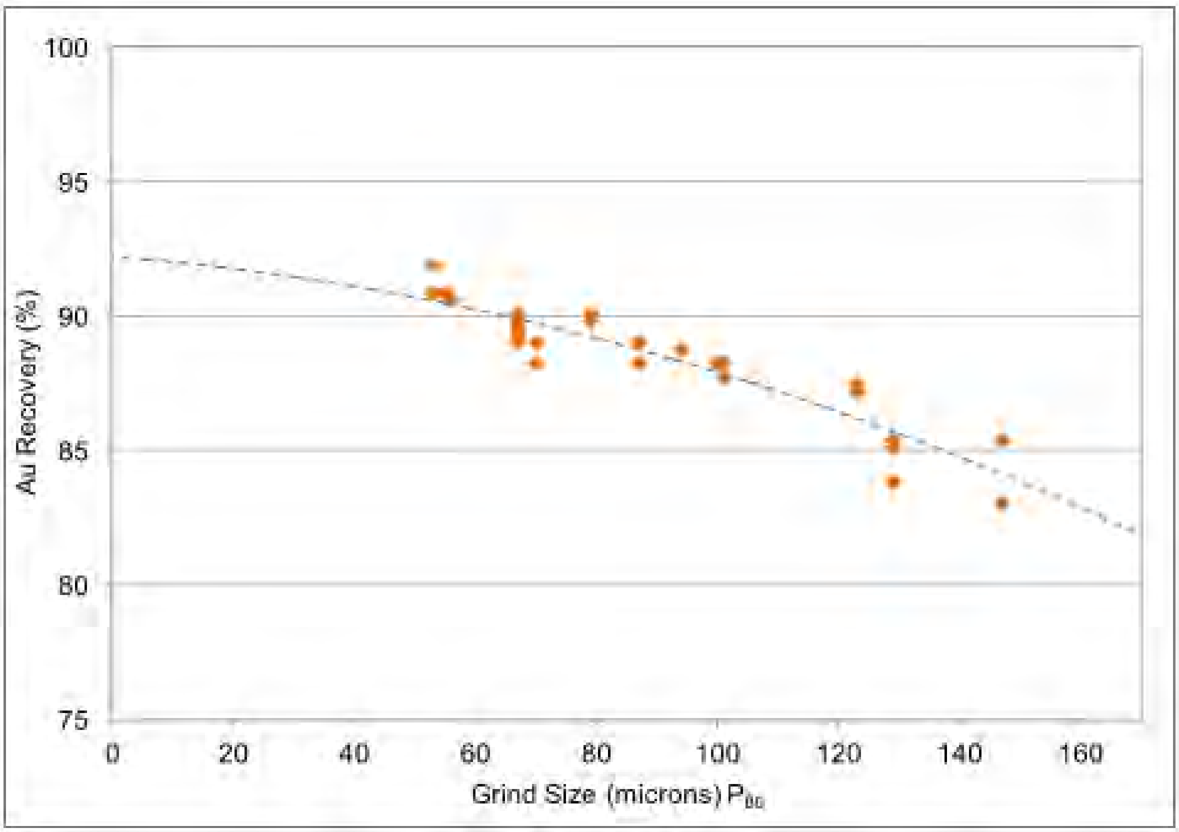
The Global Composite was also submitted for an extended Gravity Recoverable Gold ("E-GRG") test. The amount of gravity recoverable gold in the sample was assessed at 47.2%.

13.2.2.1.5 Cyanidation Testing

The Global Composite was subjected to developmental cyanidation testing. The program included whole ore versus gravity tailings leaching, effect of pre-aeration, grind size and percent solids. The results can be summarized as follows:

- Whole ore leach extraction: 85-93%;
- Gravity tailings leach recovery: 81-90%;
- Leach kinetics increased with a finer grind size and oxygen sparging;
- Oxygen sparging yielded lower cyanide consumptions compared to air sparging;
- Variations in slurry percent solids (33% to 50%) did not affect gold extraction;
- Cyanide concentration and pre-oxygenation duration did not significantly affect gold extraction;
- A finer grind improves gold recovery (Figure 13.5).

Figure 13.5: Gold Recovery as a Function of Grind Size (Global Composite)



Title: Gold Recovery vs. Grind Size	**Author: SGS Canada Inc.** **Date: 14/10/08**

Gravity tailings of the Global Composite and the Variability Composites underwent cyanidation testing at P80's of approximately 80 µm and 60 µm. The finer grind resulted in better gold extractions for all the samples (86 to 95% recovery versus 78% to 90%).

The Master Composite was submitted to leach optimization testing. The effects of grind size, residence time, lead nitrate addition, pH and carbon concentration were examined. The grind size had the most impact on gold extraction while a 2% gold recovery increase was observed when increasing the retention time from 32 to 72 hours. The gold recovery was between 85% and 89%.

The Low-Grade Composites were also submitted to cyanidation testing. Gold recovery and leach kinetics were improved at finer grind sizes. The gold recovery ranged from 80% to 95%.

The optimized leach conditions defined during the tests are as follows:

- Slurry density: 50% w/w;
- pH: 10.5-11.0;
- Dissolved oxygen: > 15 mg/L;
- Cyanide concentration: 0.5 g/L NaCN (maintained);
- Retention time: 36 hours.

13.2.2.1.6 Carbon Circuit Modelling

SGS Canada Inc. uses the semi-empirical models developed by Mintek SA (South Africa's national mineral research organization) to simulate Carbon-in-Leach ("CIL") and Carbon-in-Pulp ("CIP") circuits. The approach to CIL and CIP modelling involves conducting batch gold leaching and carbon adsorption tests with representative samples and commercially available activated carbon. The leach rate is determined through a classic bottle roll experiment by taking timed samples over a 72-hour period. The gold adsorption rate is determined by adding carbon to the slurry and again taking samples for 72 hours. Equilibrium adsorption isotherms are then established.

The Master Composite gravity tailings sample from test G-24 was used for the CIL/CIP modelling. The test revealed that leaching of the Master Composite Sample was complete after 24 hours. The sample showed relatively slow adsorption kinetics but very favorable equilibrium loading. The simulation results are presented in Table 13.17.

Table 13.17: Leach / CIP Modelling Results

Parameter	Value
Number of Leach Tanks	6 @ 4,200 m^3 each
Slurry Flow Rate	548 t/h at 55% solids
Number of Adsorption Tanks	6
Slurry Time in Each CIP Tank	0.3 h
Carbon in Each of the 6 Adsorption Tanks	12.6 t
Carbon Concentration in Adsorption Tanks	80 g/L
Gold on Carbon / Gold in Feed	1427
Carbon Advance Rate to Elution / Regeneration	7.9 t/d
Gold on Loaded Carbon	2,310 g/t
Gold on Eluted Carbon	50 g/t
Gold Locked Up on Carbon in Plant	48 kg
Ramp-up Time	11 days
Soluble Gold Losses	0.007 mg/L

13.2.2.1.7 Cyanide Destruction

The bulk leach product of the Global and Variability Composites were subjected to a single-stage cyanide destruction test to determine the samples amenability to detoxification using the SO_2/Air process. The objective of the test was to achieve weak acid dissociable cyanide (CN_{WAD}) levels below 1 mg/L.

The Global and Variability Composites A, B, G and I were the most difficult to treat. A retention time of 120 minutes, 30 to 45 mg/L of copper sulfate and more than 7.0 g of sulfur dioxide per gram of CN_{WAD} were required to meet the target. Variability Composites D, E and F also required 120 minutes of retention time, but reagents addition was lower (20 to 30 mg/L of copper sulfate and 5.7 to 6.1 g of sulfur dioxide per gram of CN_{WAD}). Finally, Variability Composites C and H only required 60 to 90 minutes of retention and 5.5 g of sulfur dioxide per gram of CN_{WAD}. It was also found that there is a strong relationship between the residual iron and the total cyanide (CN_T). The residence time and copper sulfate addition can be increased to further reduce total cyanide levels.

A two-stage cyanide destruction test was carried out on the Global Composite. The CN_{WAD} and CN_T were reduced to the targeted 1 mg/L in 90 minutes by adding 45 mg/L of copper sulfate and 7.32 g of sulfur dioxide per gram of CN_{WAD}. A shorter retention time (60 minutes) during test CND12-4 led to an increased concentration of CN_T in the cyanide destruction discharge solution (refer to Table 13.18).

Table 13.18: Two-Stage Cyanide Destruction Discharge Solution Analysis

Test No.	Solution	Analysis (mg/L)									
		Fe	Cu	CN_T	CN_F	CN_{WAD}	CNS	CNO	NH_3	NO_2	NO_3
CN-94	Final Barren	1.76	6.87	258	222	204	40	39	1.00	-	-
CND12-2	Final Destruction (R2)	0.26	0.11	0.63	0.08	0.08	46	120	12.2	< 0.3	< 0.6
CND12-4		2.22	0.11	6.07	0.04	0.04	55	190	5.3	< 0.3	< 0.6

Note: CND12 was a two-stage cyanide destruction, the final solution is the discharge from the second reaction vessel.

13.2.2.1.8 Solid-Liquid Separation and Rheology

The Global Composite and Variability Composites C, F and G cyanide destruction discharge samples were subjected to flocculant selection, static settling, dynamic settling and underflow rheology tests.

The objective of the flocculant screening test was to identify the right type of reagent for the separation process and to find a widely available and inexpensive reagent that would suit all the samples. The flocculant performance was evaluated in terms of relative effectiveness regarding particle aggregation, floc formation, resulting structure characteristics and supernatant water clarity. All the samples responded well to a low charge density anionic flocculant.

For the static tests, standard Kynch tests were conducted at variable slurry percent solids and reagent dosages. The non-optimized static settling tests results were used to define the starting conditions (feedwell solids density and relevant flowrates) for the settling tests.

The optimized dynamic settling parameters and results (flocculant dosage, unit area, solids and hydraulic loading, rise rate and residence time) are presented in Table 13.19.

Table 13.19: Dynamic Settling Test Results

Sample I.D.	Flocculant (BASF)	Dosage (g/t dry)	Dry Solids SG	U/F[1] (% wt)	U/F Extended (% wt)	TUFUA[2] ($m^2/t/d$)	THUA[3] ($m^2/t/d$)	Net Rise Rate ($m^3/m^2/day$)	Solids Loading ($t/m^2/day$)	Net Hydraulic Loading ($m^3/m^2/day$)	Res. Time (h)	Overflow (Visual)	TSS[4] (mg/L)
CND-1 Global Composite	Magnafloc 10	15	2.88	64.5	63.9	0.090	0.042	61.1	0.462	2.54	1.12	Clear	27
CND-2 Variability Composite C		17	2.82	63.5	63.7	0.080	0.019	68.6	0.519	2.86	0.95	Clear	10
CND-3 Variability Composite F		15	3.19	70.0	71.5	0.080	0.026	68.8	0.520	2.87	1.04	Clear	12
CND-4 Variability Composite G		18	2.74	64.2	67.1	0.100	0.030	54.6	0.415	2.28	1.19	Clear	43

Notes: All values were calculated without a safety factor. Key underflow rheology data were included in the rheology section.

Common Test Conditions:

Autodiluted Thickener Feed % solids = 15% w/w solids

Solution S.G – 1.000

[1] Ultimate Underflow (UF) Density

[2] Thickener Underflow Unit Area

[3] Thickener Hydraulic Unit Area

[4] Total Suspended Solids of the Overflow

The rheology tests were performed on the underflow samples generated under optimized settling conditions. The critical solids density ("CSD") for each sample is presented below. The CSD is the solids density at which a small increase in density causes a significant decrease in flowability. It also predicts the maximum solids density that is achievable in an industrial thickener and practical for pumping.

All the underflow samples displayed Bingham plastic behavior and the CSD for all four samples varied between 6%5 to 69% solids.

Table 13.20: Underflow Rheology Test Results

Sample I.D.	CSD	Yield Stress (Pa)		Flow Behaviour & Range (wt % solids)
	(wt % solids)	Unsheared	Sheared	Thixotropy
CND-1 Global Composite	66	33	14	60.5-68.9
CND-2 Variability Composite C	65	31	15	60.0-68.0
CND-3 Variability Composite F	69	35	10	63.1-73.4
CND-4 Variability Composite G	67	40	14	61.4-70.4

Note: CSD: Rheology-determined Critical Solids Density

13.2.3 Thickening and Rheology Tests

Additional thickening and rheology test work was carried out by FLSmidth in June 2014 to determine the sizing and operating parameters of a pre-leach thickener. The objective was to reach a 55% underflow density and a 50 ppm to 75 ppm solids concentration in the overflow.

13.2.3.1 Thickening and Rheology Results

FLSmidth tested five types of flocculant and the results show that an anionic polyacrylamide flocculant with a very high molecular weight and very low charge density yielded the best settling rates and overflow clarity. The flocculant recommended dosage is between 15 g and 25 g of flocculant per tonne of dry solids.

The settling flux tests determined that a feedwell percent solids of between 8% to 11% provides the best conditions for flocculation. The continuous fill tests yielded a recommended solids loading of 25 t/d/m^2 or a unit area of 0.04 m^2/t/d for the Composite Sample. The rheology tests determined that a 50% to 55% solids thickener underflow could be achieved in less than two hours with design yield stress lower than 50 Pa. The results are summarized in Table 13.21.

Table 13.21: Thickening and Rheology Tests Results Summary

Thickener Operating Parameters	Gold Ore Composite
Recommended Feedwell Suspended Solids Concentration (wt%)	11
Recommended Total Flocculant Dose (g/t)	25
Recommended Minimum Unit Area (m^2/t/d)	0.04
Design Overflow Clarity (ppm)	<40
Rheological Characteristics	
Est. Bed Solids at 0.5 h Retention Time (wt%)/ Est. Yield Stress (Pa)	57/<25
Est. Bed Solids at 1 h Retention Time (wt%)/ Est. Yield Stress (Pa)	58/<25
Est. Bed Solids at 2 h Retention Time (wt%)/ Est. Yield Stress (Pa)	60/<25
Est. Bed Solids at 4 h Retention Time (wt%)/ Est. Yield Stress (Pa)	61/<25
Est. Bed Solids at 6 h Retention Time (wt%)/ Est. Yield Stress (Pa)	73/120
High Rate Thickeners Sizing Basis	
Design U/F Solids (wt%)	50 – 60
Design U/F Retention Time (hr)	2 or less
Design Yield Stress (Pa)	25

13.2.4 HPGR Test Work

The HPGR testing program objectives were threefold. First, laboratory scale tests (batch and locked-cycle) were performed to determine the amenability of the ore to HPGR milling and yield data to allow a preliminary sizing to be done. Then, abrasion tests were completed to provide the data necessary to predict the service life of the rolls. Finally, a large-scale pilot plant test was completed to adequately size the equipment. Bond grindability testing was included in the scope of work to evaluate the BWI reduction of the HPGR product compared to the feed.

ThyssenKrupp is affiliated with SGS Minerals for the HPGR laboratory scale tests (“Labwal”) and the tests could be done in Canada. The abrasion tests (“Atwal”) were performed at ThyssenKrupp’s Resource Technologies Research Center in Germany. The pilot plant test was carried out in Germany.

Samples from each major lithology (Greywacke, Iron Formation and Porphyry) were prepared and sent to the laboratory for the Labwal tests. A representative composite sample was made from these lithology samples. The pilot plant composite sample was prepared at the same time to ensure the samples used for the laboratory scale tests and the future pilot scale tests would have the same characteristics. Table 13.22 below shows the sample preparation details.

Table 13.22: HPGR Tests Samples Preparation Details

Samples	Material Weight Distribution (kg)						Comp. Ratio (%)
	Received	Stored*	HPGR Testing	ATWAL	Compositing	Left Over	
Greywacke	969	594	165	210	594	0	50.5
Iron Formation	791	416	165	210	343	73	29.1
Porphyry	710	335	165	210	240	95	20.4
HPGR Comp	0	-	1,178	0	1,178	0	100
Total	**2,471**	**-**	**1,673**	**630**	**-**	**168**	**-**

*Note: *Material set aside for the composite*

13.2.4.1 Labwal Tests Results

The results of the Labwal tests are summarized below. The locked-cycle tests were performed using the optimal batch test conditions. One of the parameters used to determine the optimal conditions was the HPGR product fineness as a function of applied pressure. The tests results were used in SGS's comminution circuit simulations to size the HPGR.

Table 13.23: Labwal Tests Results

Sample Name	HPGR Batch Test						HPGR Locked-Cycle Test					
	Operating Press. (bar)	t/h	Net kWh/t	N/m^2	m_f	P_{80} (mm)	t/h	Net kWh/t	N/mm^2	m_f	CL (%)[1]	P_{80}
Greywacke	35	2.9	1.04	1.75	255	5.259	-	-	-	-	-	-
Greywacke	60	2.7	1.66	2.99	239	4.321	-	-	-	-	-	-
Greywacke	72	2.7	2.02	3.59	236	3.904	1.8	2.60	3.25	230	46	2.218
Iron Formation	36	3.1	0.97	1.79	273	4.731	-	-	-	-	-	-
Iron Formation	60	3.0	1.55	3.00	263	4.074	1.9	2.06	2.76	260	52	2.226
Iron Formation	72	2.9	1.80	3.57	255	4.024	-	-	-	-	-	-
Porphyry	34	2.6	1.01	1.70	233	5.243	-	-	-	-	-	-
Porphyry	58	2.5	1.69	2.87	221	4.184	1.7	2.31	2.74	224	52	2.067
Porphyry	70	2.4	1.96	3.48	216	4.060	-	-	-	-	-	-
HPGR Comp	-	-	-	-	-	-	1.8	2.59	3.22	240	48	2112

Note [1]Circulating Load:

13.2.4.2 Atwal Tests Results

The results of the Atwal tests are summarized in Table 13.24. The most abrasive sample was the Greywacke followed by the Porphyry and the Iron Formation that showed similar wear rates. According to these results, all the samples were classified as low to medium abrasive when dry (1% moisture) or wet (3% moisture).

Table 13.24: Atwal Tests Results

Ore	Test #	Moisture (%)	Grinding Force (N/mm^2)	Wear Rate (g/t)
Greywacke	A1	1.0	4.0	17.7
Greywacke	A2	3.0	4.0	20.5
Iron Formation	A1	1.0	4.0	15.6
Iron Formation	A2	3.0	4.0	17.3
Porphyry	A1	1.0	4.0	16.6
Porphyry	A2	3.0	4.0	17.0

13.2.4.3 Bond Ball Mill Grindability Tests Results

Bond grindability tests were performed at 106 µm on the four HPGR feed samples as well as on the four corresponding HPGR locked-cycle test products. Three additional tests were performed on the HPGR products using the particle size distribution of the HPGR feed samples (HPGR Adjusted Prod. samples).

The HPGR feed samples varied in terms of hardness from medium (Iron Formation) to moderately hard (Greywacke and Composite) to hard (Porphyry). When comparing the BWI values, the HPGR products were considerably softer and all fell into the medium hardness category, except for the Porphyry sample that went from hard to moderately hard. Results are summarized in Table 13.25.

Table 13.25: Bond Ball Mill Grindability Tests Results

Sample Name	Mesh of Grind	F_{80} (µm)	P_{80} (µm)	Gram per Revolution	Work Index (kWh/t)	% Reduction	Hardness Percentile	Feed Passing (%)
Greywacke	150	2,477	79	1.16	16.1	-	70	10.5
Greywacke HPGR Product	150	2,166	80	1.43	13.8	14	44	14.8
Greywacke – HPGR Adjusted Prod.*	150	2,520	79	1.31	14.6	10	52	10.3
Iron Formation	150	2,417	78	1.27	14.9	-	56	10.3
Iron Formation HPGR Product	150	2,256	77	1.44	13.4	10	40	15.8
Iron Formation – HPGR Adjusted Prod.*	150	2,440	80	1.36	14.3	4	49	10.3
Porphyry	150	2,392	80	1.09	17.1	-	77	7.3
Porphyry HPGR Product	150	2,173	82	1.22	15.9	7	68	13.6
Porphyry – HPGR Adjusted Prod.*	150	2,426	81	1.19	16.1	5	70	6.9
HPGR Comp	150	2,368	79	1.19	15.8	-	66	9.8
(HPGR Comp) HPGR Product	150	2,162	76	1.39	13.8	13	43	15.4

*Note: * Represent a different sample preparation approach explained in Subsection 13.2.4.3.*

SGS Canada Inc. developed a method that accounts for the effect of the increased quantity of fines in the HPGR product to better estimate the power reduction to grind from 100% passing 6 mesh to 100% passing 150 mesh. Based on their method, the HPGR product would require 17% to 23% less power (compared to a standard feed).

A different method was suggested to SGS by an external comminution specialist at SimSAGe. The BWI test samples were prepared to reproduce the size distribution of the Bond ball mill grindability tests performed on the HPGR feed. Based on this modified procedure, the HPGR products required 7% to 12% less power (compared to a standard feed) to grind from 100% passing 6 mesh to 100% passing 150 mesh.

13.2.4.4 Pilot Plant Tests Results

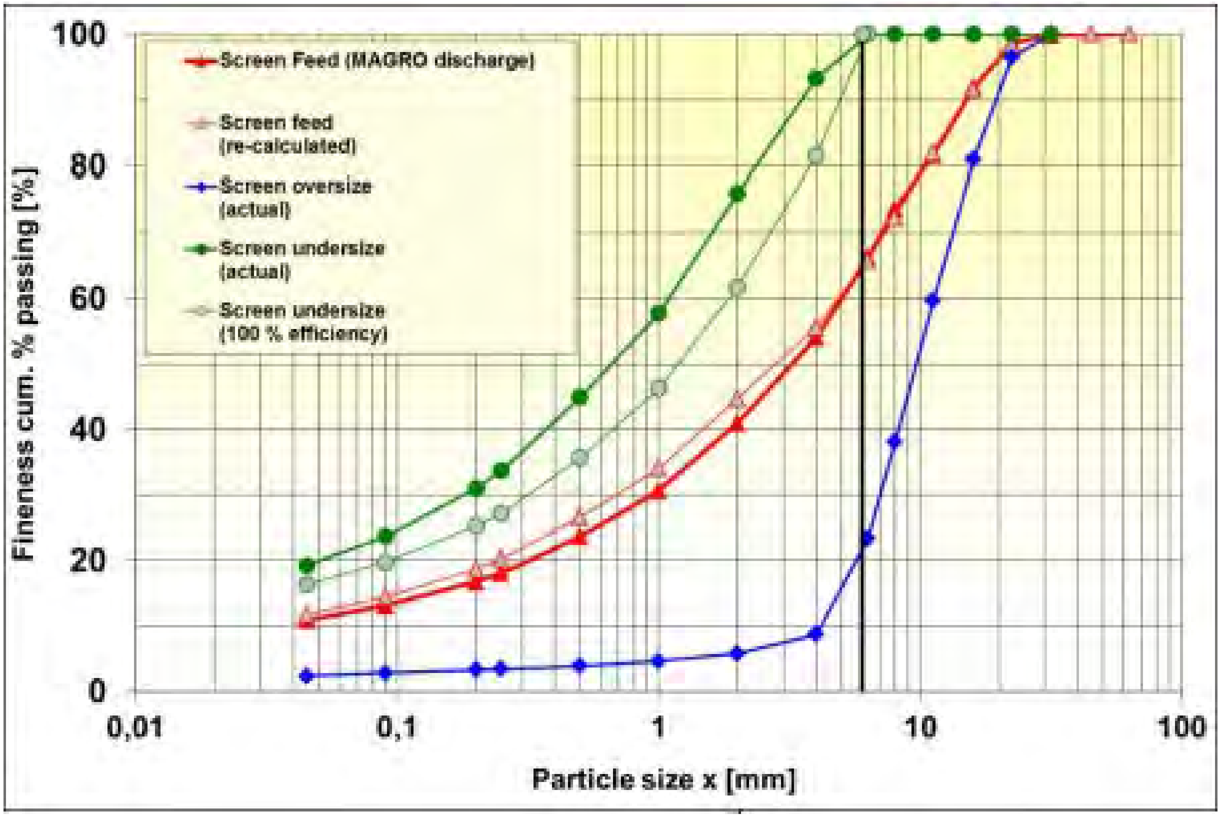
Pilot plant tests were carried out on about 950 kg of Au ore sample from the Hardrock Deposit. The sample material is a composite made of 50.5% Greywacke, 29.1% Iron Formation and 20.4% Porphyry. The ore sample was provided as drill cores that had been pre-crushed to match the feed size requirements of the units.

Pilot plant tests were conducted using a semi-industrial high-pressure grinding roll with 0.95 m diameter rolls, 0.35 m wide. Process data obtained from test work allow the sizing of industrial scale machines. The objectives in sizing HPGRs are to meet the throughput requirements and to achieve a certain product fineness. The key parameters are therefore the specific throughput rate and the specific energy consumption required to obtain the desired comminution result. The specific throughput rate varied between 306 and 320 t s/m^3 h; it was slightly dependent on the specific press force. The specific energy consumption varied between 1.4 and 2.6 kWh/t depending on the applied specific press force.

Bond tests were conducted on a conventionally crushed fresh feed sample from the provided sample as well as on the HPGR cycle products. The Bond test was conducted using a closing mesh size of 90 µm. The Bond Work Index was 10% lower after HPGR treatment: 14.73 kWh/t on crushed material compared to 13.28 kWh/t on HPGR product.

The pilot tests allowed the prediction of the expected industrial size distribution of the HPGR discharge and of the screen undersize product for a closed-circuit operation. Locked-cycle tests were conducted to simulate a continuous operation. The circulation factor was consistent in the first three cycles indicating that the circuit was stabilized. The pilot plant test third cycle size distribution is presented in Figure 13.6.

Figure 13.6: Pilot Plant Test Third Cycle Size Distribution



Title: Pilot Plant Test Third Cycle Size Distribution	Author: ThyssenKrupp Date: 15/05/05

13.3 Detailed Engineering Test Work

This section summarizes the test work program that was performed during the detailed engineering to obtain recovery data while varying feed and test conditions and further refine the block model gold recovery equation. The reference document is listed below:

- SGS Canada Inc., *An Investigation into Gold Recovery for the Hardrock Deposit, Final Report - 17074-01*, March 5, 2019.

13.3.1 Characterization and Recovery Test Work

Samples used for the recovery tests were submitted to head grade determination, gravity separation, cyanidation, solid-liquid separation and rheology testing. Seventeen near surface composite samples from the 2018 reverse circulation drilling campaign were used for the test work. The material used for the composites is 85 individual intervals of crushed rejects.

13.3.1.1 Characterization and Recovery Tests Results

13.3.1.1.1 Head Grade Determination

Head characterization included a gold screened metallics protocol, carbon speciation, sulphur speciation, arsenic individual elemental analysis, and a semi-quantitative Inductively Coupled Plasma ("ICP") scan analysis.

The gold head grades of the composites were determined by metallic sieve and a weighted average was calculated from the test work. Calculated head grades were obtained from the cyanidation leach test results. The direct and calculated gold head grades correlate well. The gold head grade results are presented in Table 13.26 as well as the sulphur speciation assays results and arsenic ("As") results.

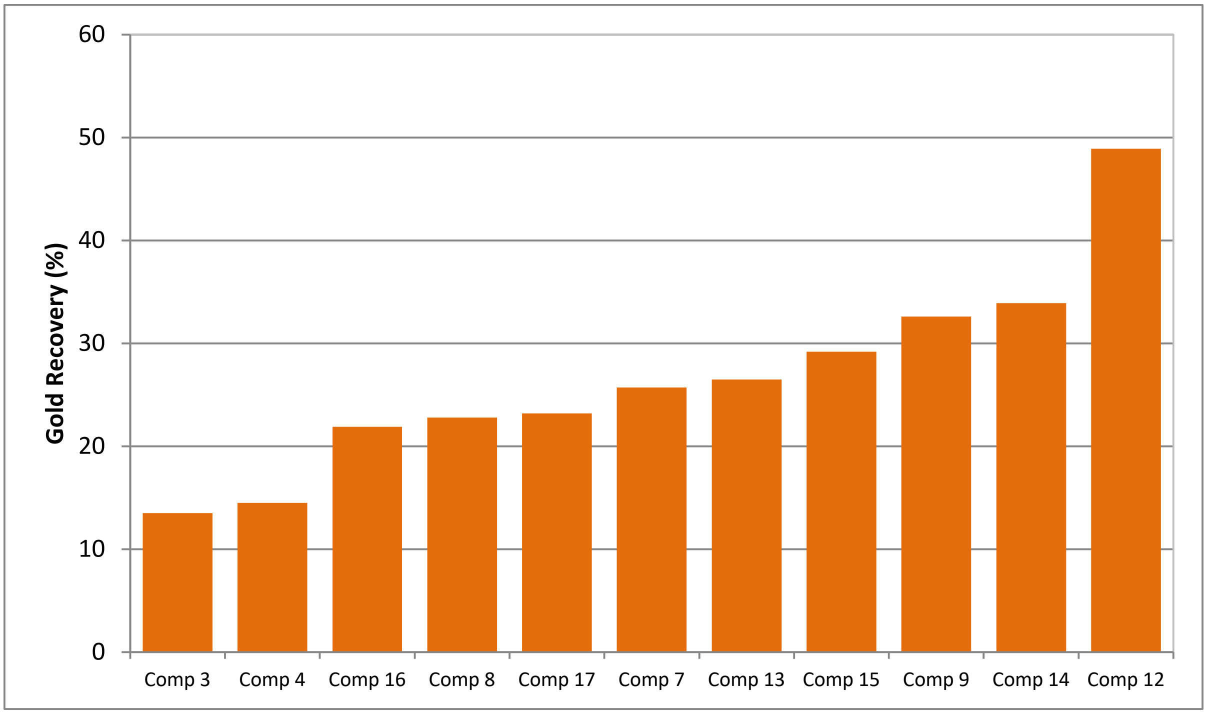
Table 13.26: Composite Samples Direct and Calculated Gold Head Grade

Sample Name	Direct g Au/t	Calculated (From Test Work) g Au/t	S^{2-} %	As %
Composite 1	0.55	0.46	1.50	0.060
Composite 2	0.57	0.52	0.23	0.002
Composite 3	1.66	1.64	1.46	0.038
Composite 4	0.73	0.82	0.57	0.011
Composite 5	0.51	0.46	1.03	0.072
Composite 6	0.50	0.77	0.44	0.006
Composite 7	1.06	1.19	0.95	0.059
Composite 8	1.55	1.49	0.44	0.008
Composite 9	1.35	1.32	0.54	0.017
Composite 10	0.40	0.50	0.99	0.052
Composite 11	0.44	0.34	0.30	< 0.001
Composite 12	19.1	22.4	0.96	0.035
Composite 13	1.02	1.04	0.35	0.002
Composite 14	0.99	1.01	0.62	0.008
Composite 15	0.73	0.80	0.43	0.007
Composite 16	1.09	0.87	0.88	0.038
Composite 17	1.15	0.80	0.58	0.023

13.3.1.1.2 Gravity Recovery

Eleven composites were subjected to gravity separation testing using a Knelson Concentrator and a Mozley table. Ten (10) composites were ground to a P80 of 90 µm and one (1) sample (Composite 9) was ground to a P80 of 110 µm. The results are shown in Figure 13.7.

Figure 13.7: Composites Gravity Recovery Results



Title: Composites Gravity Recovery	Author: SGS Canada Inc. / Soutex Date: 19/03/05

13.3.1.1.3 Cyanidation Testing

The composites were subjected to confirmatory cyanidation testing. The program included whole ore versus gravity tailings leaching, effect of pH, sodium cyanide concentration, grind size and dissolved oxygen content.

The baseline test conditions as defined previously were used:

- Slurry density: 50% w/w;
- pH: 10.5-11.0;
- Dissolved oxygen: > 15 mg/L;
- Cyanide concentration: 0.5 g/L NaCN (maintained).

Modifications were made to the test conditions to validate the influence of each parameter. The parameters were varied as follows:

- Cyanide concentration was varied from 0.10 g/L to 0.5 g/L;
- pH was varied from 9.8-10.3 to 10.5-11;
- The final DO varied from 7 to 33 mg/L;
- The gravity tailings P80 varied from 70 to 110 µm.

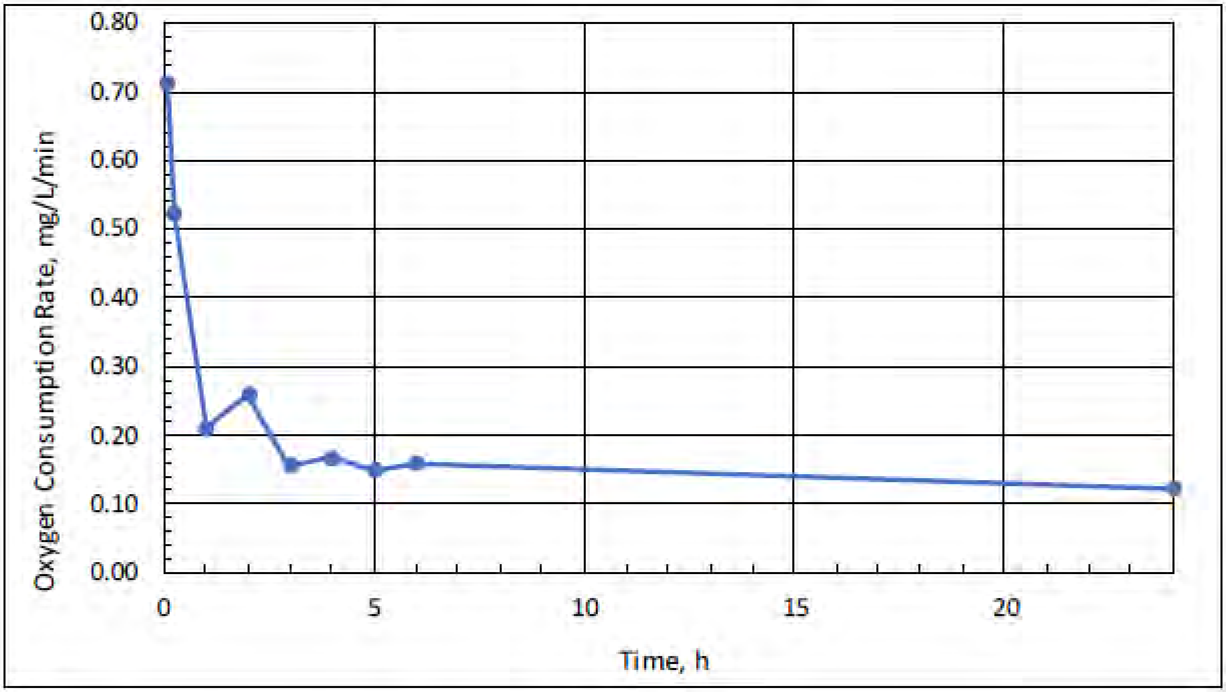
The results can be summarized as follows:

- Whole ore leach extraction at the baseline conditions was between 85 and 99%;
- Gravity tailings leach recovery at the baseline conditions was between 88 and 99%;
- Gold recovery increased with a finer grind size;
- Gold recovery increased with increasing cyanide concentration up to about 0.35 g/L;
- The pH did not significantly affect the recovery results.

13.3.1.1.4 Oxygen Uptake Test

An oxygen uptake test was performed to provide baseline oxygen consumption information of one of the gravity tailings samples. Composite 4 was used for the sample at a P80 grind size of 93 µm obtained after the gravity separation. Oxygen was sparged into the pulp sample at a rate to reach 15 mg/L and then it was regulated to maintain the dissolved oxygen target concentration in the range of 15 mg/L to 20 mg/L. The oxygen consumption rate is presented in Figure 13.8.

Figure 13.8: Oxygen Uptake Test



Title: **Composite 4 Gravity Tailings Oxygen Consumption Plot**	**Author: SGS Canada Inc.** **Date: 19/03/05**

13.3.1.1.5 Solid-Liquid Separation and Rheology

As part of the test program, two (2) ground composite samples were subjected to solid-liquid separation and rheology testing. Both composite samples were tested at particle size K80 targets at 90 µm and 72 µm. The composites were subjected to flocculant selection, pH optimization, static and dynamic thickening, and underflow rheology.

Flocculant scoping tests were conducted on all samples at a pH of 10.5 using a range of anionic, non-ionic, and cationic flocculants. The results indicated that all samples responded well to BASF Magnafloc 10.

An optimization test was conducted on each sample to examine the effect of pH on the settling response and supernatant clarity. The pH optimization tests were conducted on each sample at pH's of 9.0, 9.7, and 10.5 at 15 g/t dosage of BASF Magnafloc 10 flocculant. Based on the results of the initial settling rate ("ISR") and total suspended solids ("TSS") of the supernatant, a pH of 10 was identified as optimal.

Non-optimized static tests were conducted to define the starting conditions for the dynamic settling tests. Dynamic settling tests were realized on the composite while varying the flocculant dosages or the thickener unit areas. The flocculant used was the BASF Magnafloc 10 at a diluted thickener feed of 10% w/w. The dynamic settling parameters and results are presented in Table 13.27.

Table 13.27: Dynamic Thickening – Overall Results Summary

Sample I.D.	Dosage[1] (g/t dry)	Unit area[2] (m^2/t/d)	Solids Loading (t/m^2/day)	Net Rise Rate (m^3/m^2/day)	Underflow Density % w/w Solids	Overflow TSS mg/L[3]	Residence Time, h
BIF-90 µm	10 - 20	0.08 - 0.13	0.52 - 0.32	110.0 - 67.7	63.0 - 69.8	95 - 27	0.89 - 1.45
BIF-72 µm	10 - 20	0.07 - 0.14	0.60 - 0.30	125.7 - 62.9	62.0 - 67.6	272 - 24	0.79 - 1.58
GWK-90 µm	20 - 25	0.07 - 0.13	0.60 - 0.32	125.0 - 67.3	58.0 - 64.9	149 - 40	0.77 - 1.42
GWK-72 µm	25 - 30	0.08 - 0.16	0.52 - 0.26	109.4 - 54.5	59.5 - 65.5	96 - 43	0.82 - 1.64
BIF/GWK 90 µm Blend	15 - 25	0.04 - 0.13	1.04 - 0.32	220.0 - 68.1	58.7 - 65.8	192 - 36	0.42 - 1.38

[1] Flocculant dosage range that was tested in the laboratory dynamic thickening tests
[2] Unit area range that was tested in the laboratory dynamic thickening tests
[3] Overflow total suspended solids (TSS) range, expressed in mg/L

The results can be summarized as follows:

- Overflow TSS decreased as the unit area increased;
- Underflow solids density increased as the unit area increased;
- Overflow TSS decreased as the flocculant dosage increased up to about 20-25 g/t.

Rheology characterization was realized to study the relationship between the solids specific gravity and slurry solid content. The deviation of the actual specific gravity versus the specific gravity of the dry material defines the slurry interparticle interaction coefficient. All underflow samples exhibited insignificant inter-particle interactions, meaning that the dry solids specific gravity was comparable to their densities in the slurry phase. All underflow samples exhibited a Bingham plastic rheological behavior and were generally thixotropic.

13.4 Conclusions and Recommendations

13.4.1 Grinding

Grindability tests have been performed on a sufficient number of samples to properly assess the comminution characteristics of the Hardrock Deposit. Generally, the ore falls into the high hardness end of the spectrum. The test data from the various tests need to be manipulated to estimate values that represent the run-of-mine composition (weighted averages). These results are used as a basis for plant design.

13.4.2 High-Pressure Grinding

The HPGR Labwal tests showed that the Hardrock Deposit is amenable to high pressure grinding and yielded a net power consumption of 2.6 kWh/t. The abrasion tests determined that the ore falls into the low to medium abrasiveness categories. Bond ball mill grindability comparative tests done on the HPGR feed and product revealed that a 7% to 12% power reduction could be expected when grinding a HPGR product.

13.4.3 Magnetic Separation

The magnetic separation tests revealed that a variable amount of magnetic minerals is present in the different composites and that gold losses associated with the removal of the magnetic fraction can be significant. The tests also expose the fact that large amounts of gold bearing ore could potentially be rejected from the process if magnets are installed on relatively fine ore streams.

13.4.4 Gravity Recovery

Gravity recovery tests showed that gravity separation is an efficient method of recovering gold. Cyanidation of gravity tailings is an economical method of gold recovery and removal of a small portion of gold reduces cyanide consumption in the leach circuit and carbon circuit requirements.

13.4.5 Flotation

Comparing gold extraction by cyanidation of whole ore with cyanidation of flotation concentrate, there was no benefit seen by including the flotation stage because the expected recovery with the flotation process does not demonstrate improvement to the overall metallurgical performance.

13.4.6 Pressure Oxidation

Pressure oxidation as a pre-treatment ahead of cyanidation increased gold extraction to 97% (overall recovery of 94% including flotation) and compared favourably to cyanidation of finely ground rougher concentrate. Pressure oxidation, however, is a costly method for increasing gold extraction.

13.4.7 Cyanidation

The cyanidation tests revealed that overall gold recovery is improved at finer grinds and cyanide consumption is increased. The optimal leach conditions are defined as follows:

- Slurry density: 50% w/w;
- pH: 10.5-11.0;
- Dissolved oxygen: > 15 mg/L;
- Cyanide concentration: 0.35 g/L NaCN (maintained);
- Retention time: 30 hours.

13.4.8 Cyanide Destruction

The SO_2/Air process is effective at reducing cyanide levels to below 1 mg/L in the final tailings. A 90-minute retention time is required with the addition of 45 mg/L of copper sulfate and 7.32 g of sulfur dioxide per gram of CN_{WAD}.

13.4.9 Solid-Liquid Separation and Rheology

The pre-leach slurry can be thickened to 55% solids w/w by adding a low charge density anionic flocculant at a 15 g/t dosage.

13.5 Future Work

Additional tests are recommended to be conducted prior to operations commencing:

- Cyanide destruction optimization test work to confirm the reagents to be used and the operating conditions. Investigate the possibility of realizing the cyanide destruction and the precipitation of arsenic in two stages.

14. MINERAL RESOURCE ESTIMATES

14.1 Hardrock Mineral Resource Estimate

This Mineral Resource estimate ("MRE") is an update to the MRE conducted in 2016 prepared by G Mining Services Inc. ("GMS") for the Hardrock Project (the "Project"), the results of which were disclosed in National Instrument 43-101 ("NI 43-101") Technical Report issued on December 21, 2016.

Completion of the current MRE update involved the assessment of an updated drill hole database, which included data for an additional 481 reverse circulation ("RC") and 53 diamond drill holes ("DDH") completed since the last MRE, and an updated block model for mine planning purposes.

The 2019 MRE update was produced under the supervision of Mr. Réjean Sirois, P.Eng. of GMS, Vice President Geology and Resources, an independent "qualified person" ("QP") as defined in NI 43-101. Mr. Sirois visited the Project on multiple occasions in 2018 and 2019 to review drilling and sampling protocols. The effective date of the updated MRE is September 4th, 2019.

A single MRE was prepared from 17 sub-vertical mineralization domains (the "principal domains") and remaining mineralization was captured by Leapfrog™ RBF Grade Shells. (the "external grade shells"). Geovia GEMS™ was used for block modelling and estimation, and Leapfrog GEO™ was used for wireframing and geological interpretation.

Inverse Distance Cubed ("ID^3") was used to interpolate gold grades (g Au/t) into a block model using the modelled mineralization domains. Measured, Indicated and Inferred Mineral Resources are reported in the summary tables in Subsection 14.1.16. The MRE takes into consideration that the Hardrock Gold Deposit will be mined by both open pit and underground mining methods.

The MRE was prepared in accordance with CIM Standards on Mineral Resources and Reserves (adopted May 10, 2014) and is reported in accordance with Canadian National Instrument 43-101 - *Standards of Disclosure for Mineral Projects*. Classification, or assigning a level of confidence to Mineral Resources, has been undertaken with strict adherence to CIM Standards on Mineral Resources and Reserves. In the opinion of GMS, the Resource evaluation reported herein is a reasonable representation of the global Mineral Resources found in the Hardrock Project at the current level and spacing of sampling.

The MRE includes Inferred Mineral Resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is also no certainty that these Inferred Mineral Resources will be converted to the indicated and measured categories through further drilling, or into Mineral Reserves, once economic considerations are applied.

Resource estimation methodologies, results and validations are presented in this section of the Technical Report.

14.1.1 Drill Hole Database

In order to complete an updated MRE for the Hardrock Project, a database comprising a series of comma-delimited spreadsheets containing information for 481 reverse circulation grade control ("RCGC") drill holes and 53 DDH completed since the 2016 MRE was provided to GMS during 2018 and 2019 by Greenstone Gold Mines ("GGM"). The database included drill holes collar information (NAD83 / UTM Zone 16), surveys, assays, lithological, alteration, structural and geotechnical data. The data for the additional drill holes was subsequently imported and merged with the previous database used in the 2016 MRE in Geovia GEMS™ version 6.8.2 software ("GEMS") for statistical analysis, block modelling and resource estimation.

The current MRE is derived exclusively from the database described in Section 12 - Data Verification. GMS has reviewed the database and is satisfied with the integrity of the drilling database. Only minor errors were detected during data validation and were subsequently corrected were possible. Although minor discrepancies still remain in the drilling database (as outlined in Section 12 - Data Verification), GMS believes it is suitable for the purposes of resource estimation. A summary of the updated drill hole database is presented in Table 14.1

Table 14.1: Summary of MRE Drilling Database for the Hardrock Gold Deposit

Period	No. Surface Diamond Drill Holes	Surface Diamond Drilling (m)	No. RC Drill Holes	RC Drilling (m)
Prior to 2018	1,629	684,116.1	-	-
2018	-	-	405	20,014.5
2019	53	12,008.9	76	5,946.0
Total	**1,682**	**696,125.0**	**481**	**25,960.5**

The drill holes database for the 2019 MRE contains a total of 481 RCGC and 1,682 DDH (or surface drill holes). All 2,163 drill holes were used in the 2019 MRE update, representing the drill holes completed and validated at the data close-out date of May 14, 2019. Historical underground drilling was not considered in the 2019 MRE (consistent with the 2016 MRE).

Since the 2016 MRE, there has been significant drilling at the Hardrock Project. RCGC drilling in 2018 focused on near-surface mineralization of the SP-Zone, the F-Zone and the North 1-Zone, within the first three benches of the proposed open pit. The goal of this program was to demonstrate continuity of gold mineralization in the initial years of production, and to validate and de-risk the near-surface mineral resource.

After an interim mineral resource was produced in 2018, further diamond and RC drilling was undertaken to validate the interim mineral resource, focusing on a few gaps in the drill-spacing in and around the proposed open-pit, and benches 4 - 7 targeting the SP-Zone.

The drill holes cover the 5.7 km strike-length of the project at an irregular 50 m (X) by 25 m (Y) drill spacing within the extents of the proposed open pit with some infill drilling on a 25 m (X) spacing focusing on mineralized domains. This spacing tightens to 20 m (X) by 10 m (Y) around near surface mineralized domains targeted by the 2018/2019 RCGC drilling campaigns (Figure 14.1). For the deeper, underground portion of the deposit, drilling is sparser and is generally on 100 m (X) traverses with additional infill drilling to 50 m (X) x 50 m (Y) focusing on stronger mineralized areas. For the surrounds of the proposed open pit, the drill spacing is judged adequate to develop a reasonable geological model of the distribution of mineralization, and to quantify its volume and continuity with a reasonable level of confidence. Figure 14.2 illustrates a 3D view of drill hole spacing for the Hardrock Project.

A surface channel sample database with a total of 1,219 assays from 26 channel samples collected in 2014 was already integrated into the GEMS project from 2016. Channel samples were not used in the 2019 MRE as they are superseded by RC grade control drilling.

Figure 14.1: Plan View of Drill Hole Collars – Hardrock Project

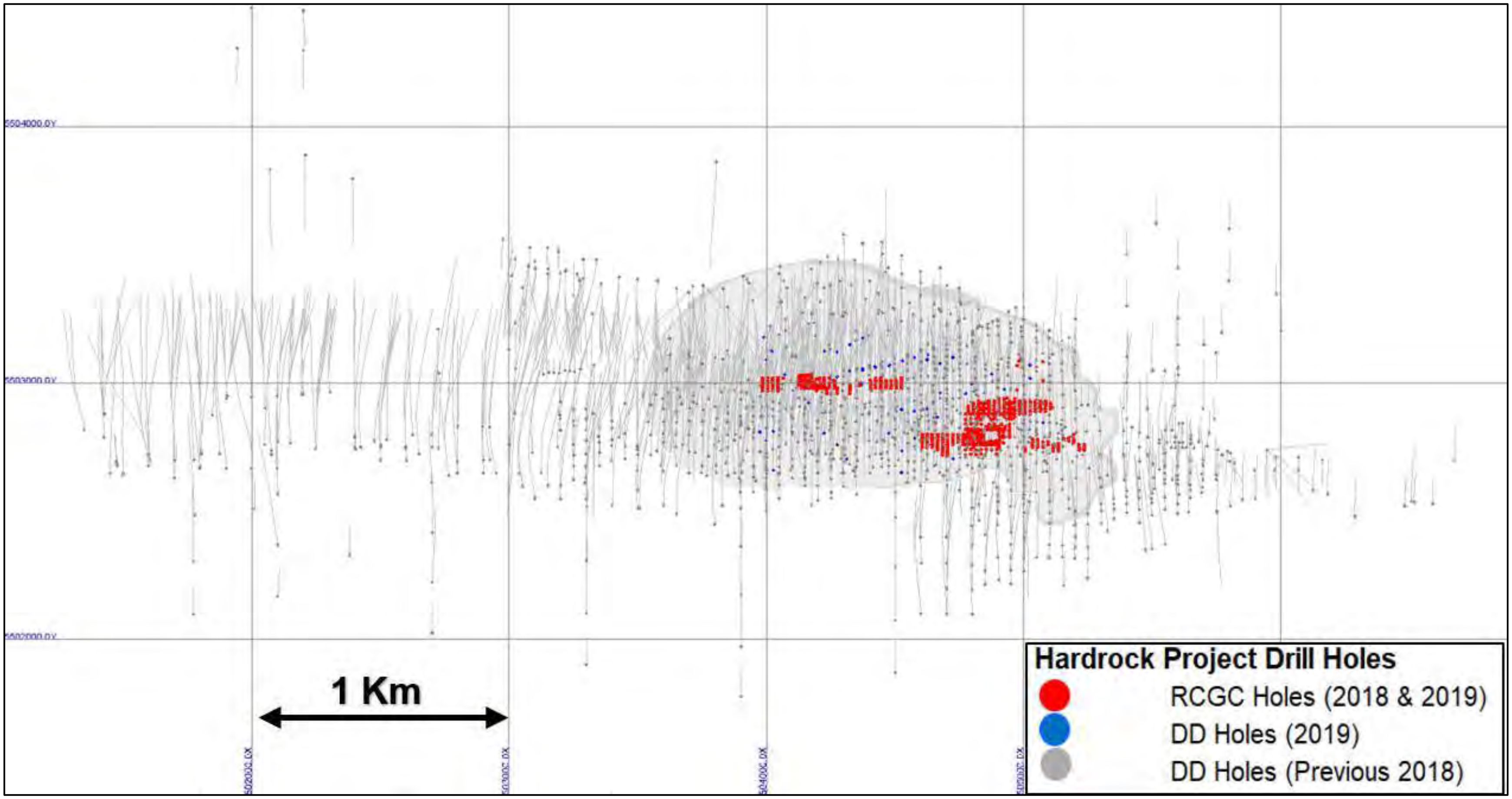
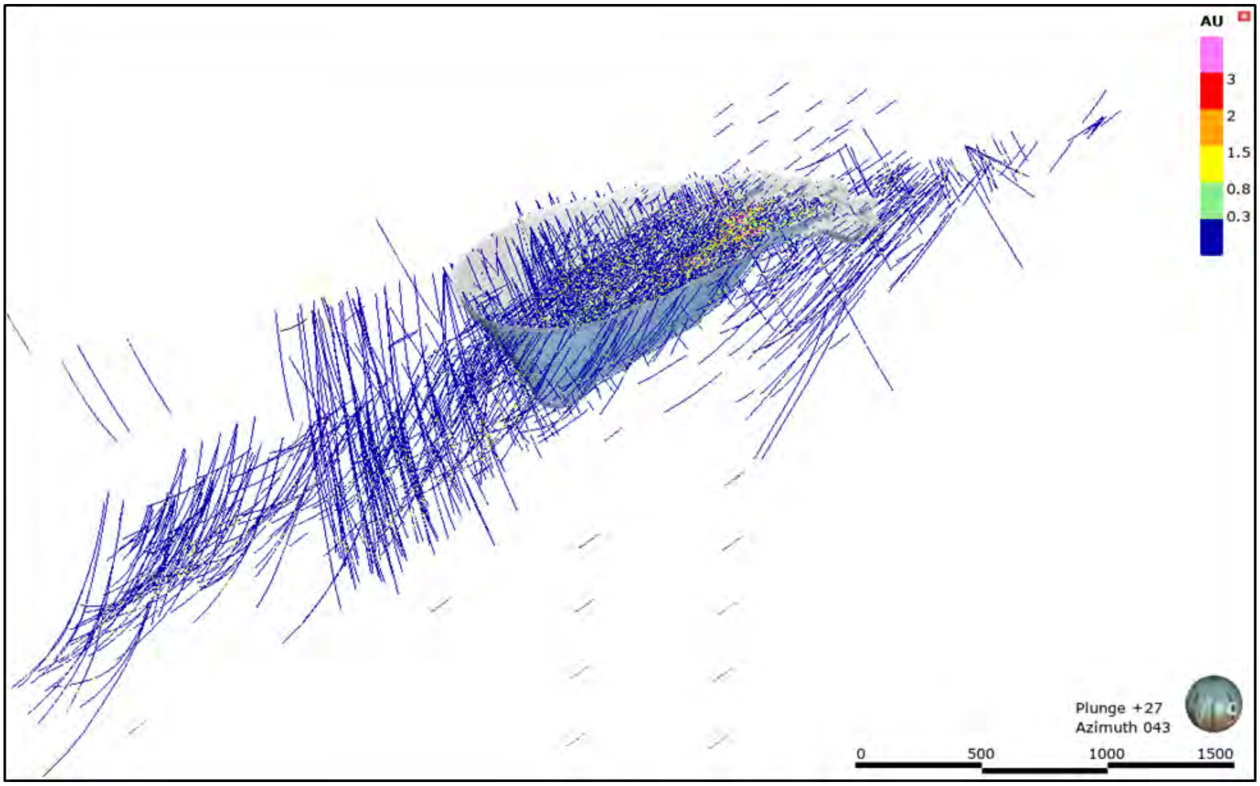


Figure 14.2: 3D View of Drill Holes, View towards North-East and 2016 FS Pit Design

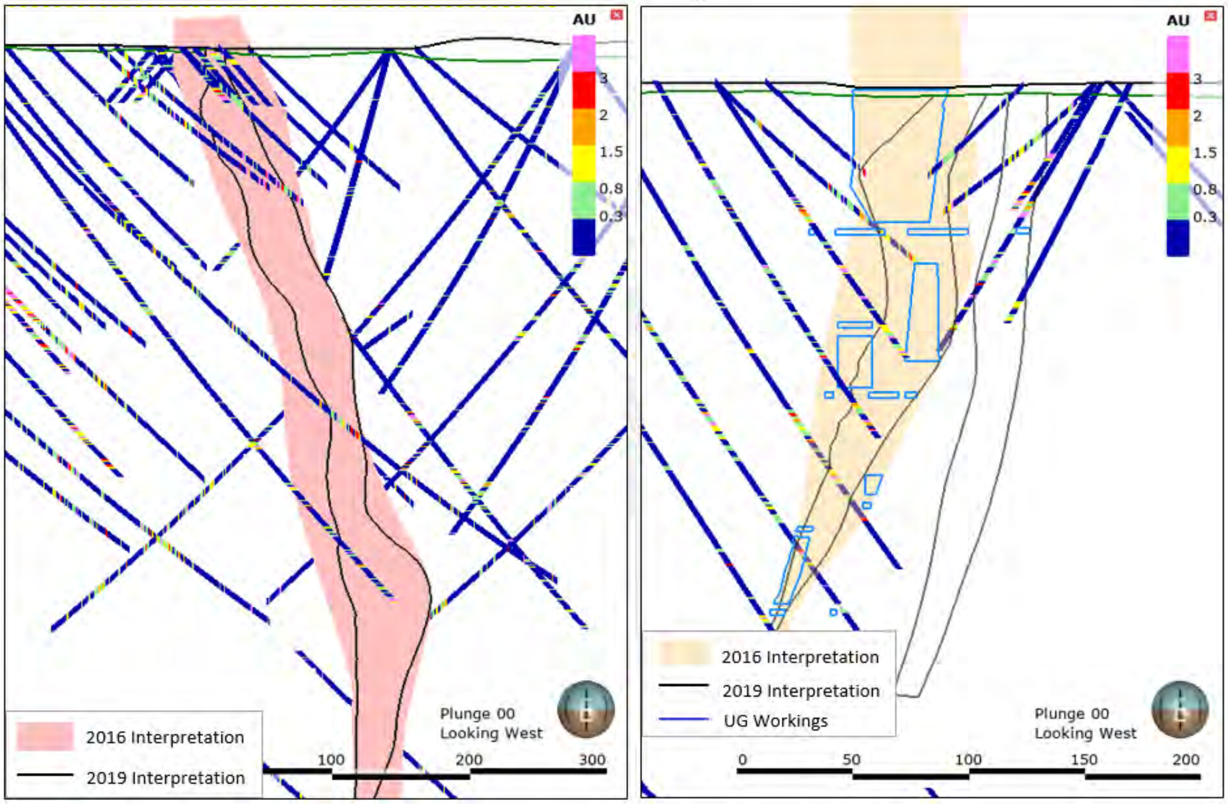


14.1.2 Geological Modelling Approach

For the 2016 MRE, a litho-structural model was developed that divided mineralization into structurally controlled, subvertical mineralization wireframes, and remaining mineralization was captured using lithological domains based on a sectional interpretation of the various lithologies (iron formation, greywacke, porphyry, gabbro and ultramafic). Although the structural wireframes demonstrated continuity between drilling sections, they contained significant dilution and required refining. The lithology wireframes contained large proportions of unmineralized samples (often greater than 75% of samples below the resource cut-off) and were inadequate for controlling gold grades during interpolation, resulting in what we believe excessive grade smearing.

For the 2019 MRE update, GMS refined the 2016 interpretation by minimizing dilution within the subvertical structural wireframes (Figure 14.3), adding any additional zones of cohesive mineralization, and splitting any original domains into separate wireframes to reduce internal dilution. For the remaining mineralization, Leapfrog RBF™ grade shells were employed which followed a trend based on of the 2016 lithological model.

Figure 14.3: 2019 Interpretation vs. 2016 Interpretation. Left: Section 504,250mE North 1-Zone. Right: Section 504,845mE North 2-Zone



14.1.2.1 <u>Principal Mineralization Domains</u>

A total of 17 mineralized domains were modelled in Leapfrog Geo™ software using as reference the mineralized domains defined in 2016 MRE. The new interpretation was completed on cross-section using an average grade of 0.3 g/t as the lower limit and 3D solids were built using the new hanging wall and footwall intervals. A minimum true thickness of 3.0 m was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed. Overlaps of wireframes are minimal and were handled by the "precedence" system used by Geovia GEMS™ for coding the block model and drill intercepts.

The SP Zone was modelled using a slightly different approach, as mineralization is anticlinal in nature. Using a 0.3 g/t lower limit, drilling intervals were defined manually on-section, and an indicator method was used to build a wireframe that adheres to the anticlinal structural trend in Leapfrog GEO™.

For the underground portion of the deposit, GMS made adjustments to the SP-Zone at depth, where data-spacing and lack of grade continuity did not permit a continuous geological interpretation.

Figure 14.4 shows a 3D view of the seventeen principal mineralized domains modelled by GMS for the current MRE. Figure 14.5 shows long-sections of six of the seventeen principal domains, and Figure 14.6 presents an example cross-section. The nomenclature is as follows:

- F-Zone (3105);
- North 1-Zone (3105);
- A splay off the original North 1 Zone (3210);
- Central-Zone (3300);
- A new iron-formation-related tabular domain at depth (3305);
- F2 Zone (3400);
- SP Zone (3500);
- North 2-Zone (3600)
- A splay of the original North 2 Zone domain (3605);
- A small domain north of the North 2 Zone (3610);
- A small domain east of the North 2 Zone (3620);
- North 3-Zone, north limb (3710)
- North 3-Zone, south limb (3720);
- Lower Zone (3800);
- A-Zone (3900);
- Tenacity Zone (4000);
- SP2 Zone (4100).

Figure 14.4: Isometric View of the 17 Principal Domains. Looking NE. and 2016 FS Pit Design shown for Scale

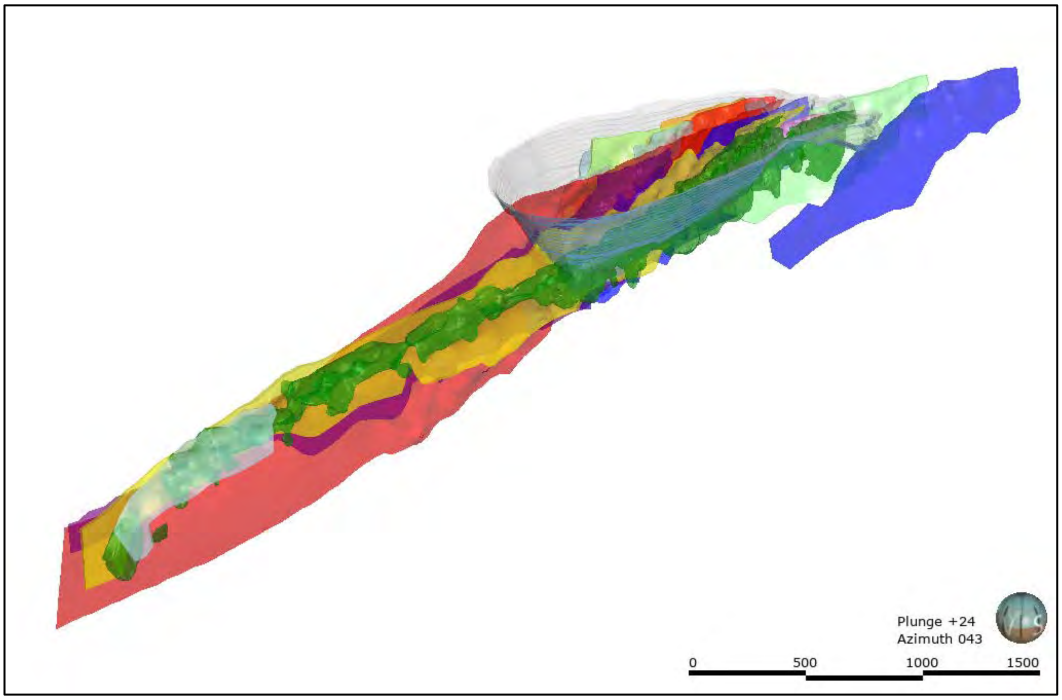


Figure 14.5: Long-Section Views (looking north) of Six of the 17 Principal Domains, showing 2016 FS Pit Design for Scale

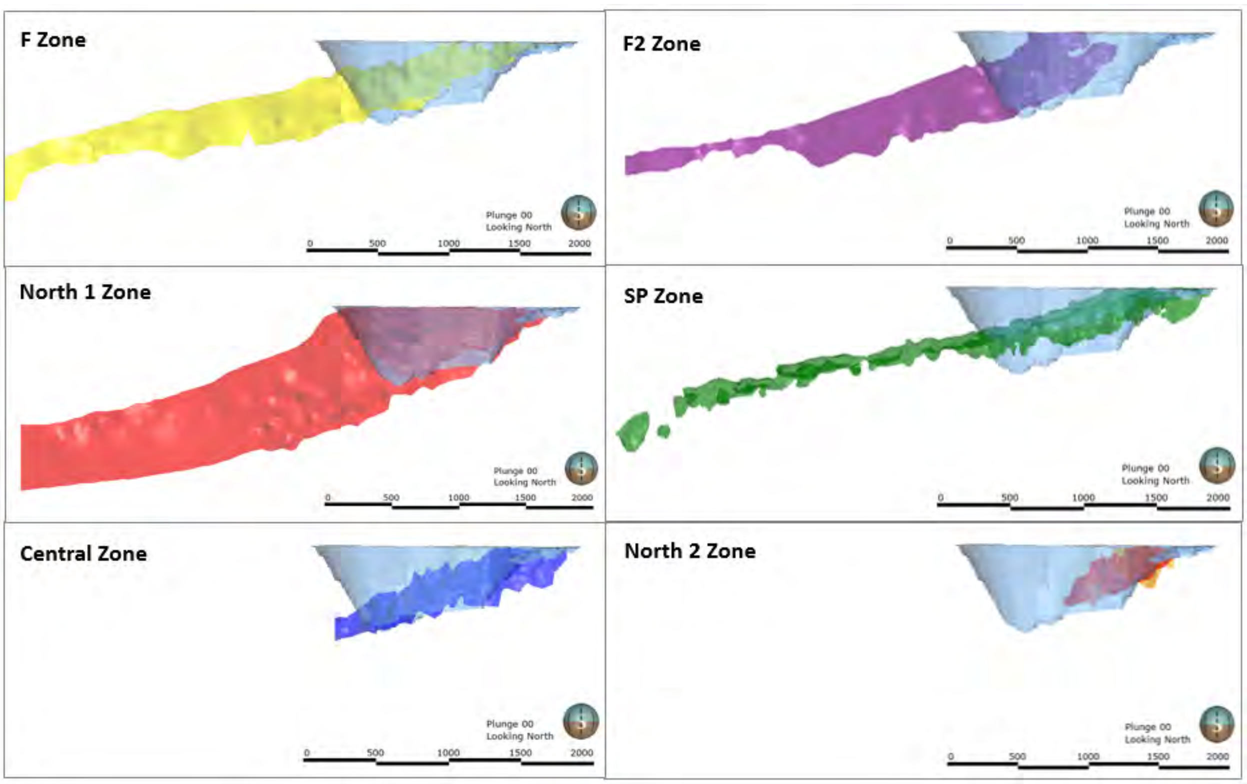
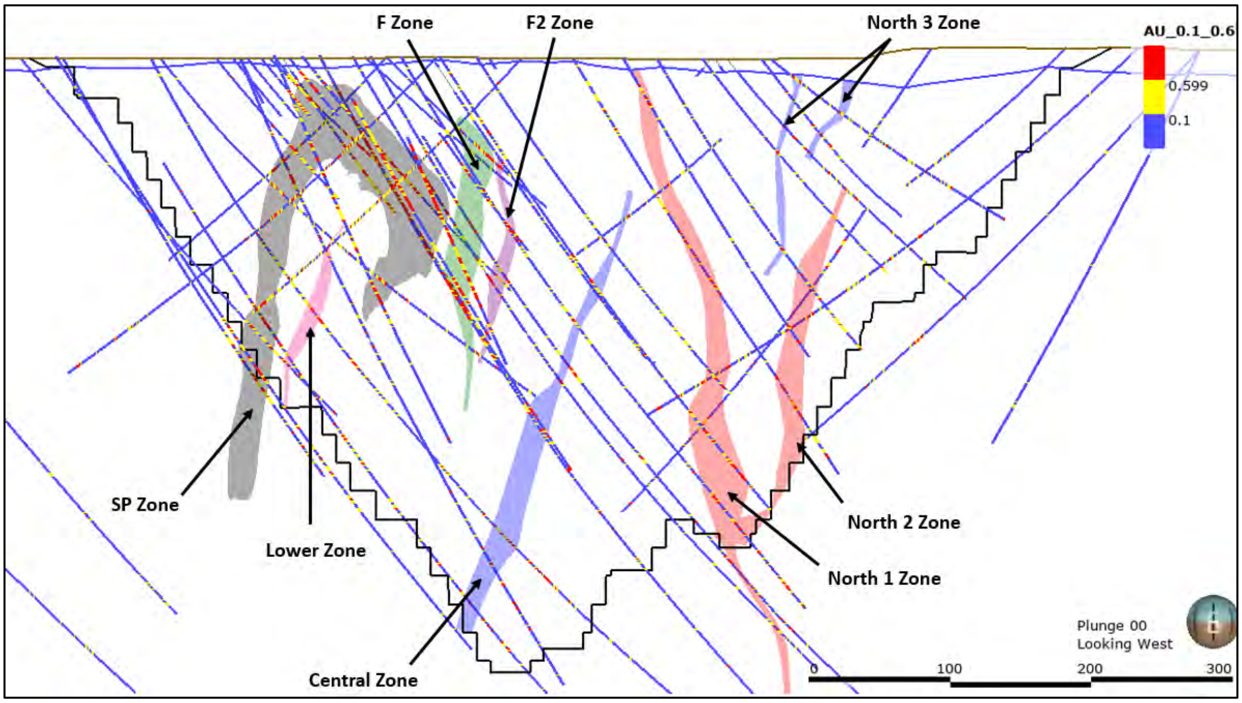


Figure 14.6: Section 504,600mE, Looking West: Principal Domains and Drilling (g Au/t), 2016 FS Pit Design shown for Scale



14.1.2.2 Internal Sub-domain Grade Shells

Six of these seventeen principal domains were further split into low-grade (<0.1 g/t), medium-grade (0.1 - 0.6 g/t), and higher-grade (>0.6 g/t) sub-domains to reduce grade smearing and ensure that internal waste within a given domain is well-represented. The internal grade shells were applied to the following domains:

- Domain 3105 (F-Zone);
- Domain 3205 (North 1-Zone);
- Domain 3300 (Central-Zone);
- Domain 3305 (New Domain);
- Domain 3405 (F2-Zone);
- Domain 3500 (SP-Zone).

Internal grade shells were employed with the goal of reducing grade-smearing. For example, around 60% of the assays within the North 1-Zone are still below 0.3 g/t (even after refinements were made), implying that without additional measures there would be significant mixing of mineralized and unmineralized samples during grade interpolation.

The thresholds of 0.1 g/t and 0.6 g/t were chosen based on statistical analysis. 0.1 g/t appears to be a natural limit between mineralized and non-mineralized material, and 0.6 g/t was chosen to ensure that continuity between medium and higher grades populations was preserved during the modelling in sections (Figure 14.7).

Table 14.2 represent the volume changes between the 2016 MRE and the 2019 MRE for each of the 17 principal domains.

Figure 14.7: Section 504,815mE (looking west) - Example of Internal Grade Shell Sub-domains for the SP Zone. Blue: < 0.1 g/t, Yellow: 0.1 – 0.6 g/t, Red: > 0.6 g/t.

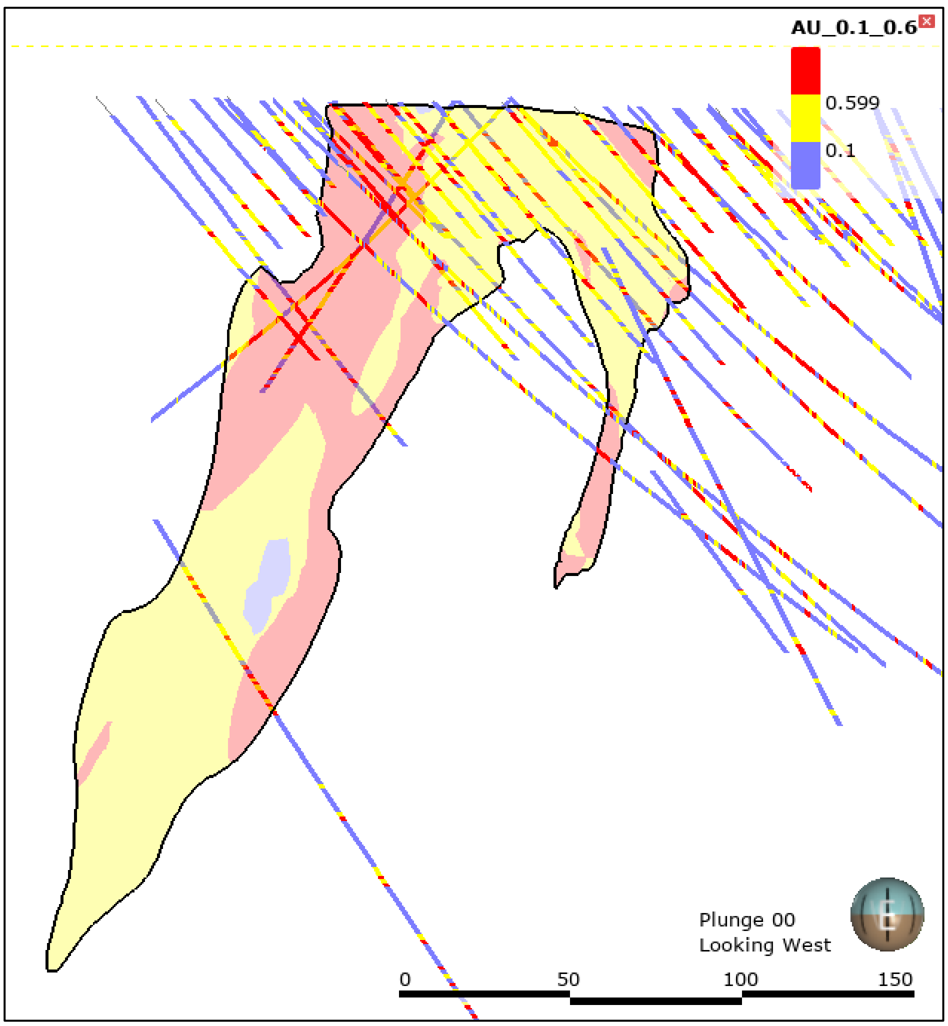


Table 14.2: Volume Changes – 2016 vs. 2019 Geological Modelling – All the Deposit

Domain	2016			2019			Volume Difference 2016 vs. 2019 (%)	Grade Difference 2016 vs. 2019 (%)
	Wireframe Volume ('000 m^3)	Average g Au/t (uncapped)	% Assays > 0.30 g/t	Wireframe Volume ('000 m^3)	Average g Au/t (uncapped)	% Assays > 0.30 g/t		
3105	19,890	1.60	50%	18,121	1.73	55%	-8.90%	8.00%
3205	37,638	1.02	32%	28,391	1.21	40%	-24.60%	18.90%
3210	New Domain			320	2.15	49%		-
3300	6,371	1.08	36%	5,690	1.33	47%	-10.70%	23.60%
3305	New Domain			1,419	1.67	47%		-
3405	13,090	0.78	32%	11,032	1.24	47%	-15.70%	58.00%
3600 + 3605	5,143	2.15	38%	4,197	2.22	48%	-18.40%	3.50%
3610	New Domain			426	2.12	48%		-
3620	New Domain			71	0.83	69%		
3710 + 3720	1,756	1.33	50%	2,280	1.61	56%	29.80%	20.70%
3800	337	0.82	35%	643	1.15	54%	90.60%	40.20%
3900	4,982	0.81	48%	3,575	1.04	61%	-28.20%	27.80%
4000	6,969	1.24	41%	4,858	1.56	51%	-30.30%	25.50%
4100	1,976	2.41	57%	2,282	2.44	61%	15.50%	1.10%
Total	**98,152**	**1.20**	**35%**	**83,305**	**1.48**	**48%**	**-15.10%**	**23.90%**
3500*	**13,034**	**1.20**	**53%**	**26,658**	**1.15**	**54%**	**104.50%**	**-3.80%**

Notes: SP-Zone (3500 rock code) interpretation largened significantly due to the inclusion of down-dip extensions of lower-grade material.*

14.1.2.3 External Grade Shells

All mineralization outside of the seventeen principal domain wireframes were captured by Leapfrog™ RBF Grade Shells. Three grade-controlled domains were chosen: low-grade (<0.1 g/t), medium-grade (0.1 - 0.6 g/t), and higher-grade (>0.6 g/t) shells. The construction of the grade shells was guided by a trend based on lithological wireframes and the 17 principal domains.

Gold values were temporarily capped to 10 g/t during the construction of the grade shells to reduce the "leapfrog bubble effect" and to prevent the overestimation of volume in data-sparse areas.

14.1.2.4 Structural Domain Subdivisions

Three external grade shell domains (500, 501 and 506) and two principal domains (SP-Zone and North 2 Zone) were subdivided into search ellipse sub-domains to guide the search ellipses around the complex orientations (with soft boundaries). This is discussed in more detail in Subsection 14.1.5.

14.1.2.5 Topographic and Bedrock Surfaces

A topographic surface was generated from LiDAR data provided to GMS. A base of overburden surface was generated in Leapfrog GEO™ from drill hole lithology information to evaluate the overburden thickness. Geotechnical drilling information collected in the vicinity of the historic tailings was also provided to GMS to enable the subdivision of tailings material into organics, tailings and fill. Wireframes were modelled for these three material types and incorporated into the block modelling process.

14.1.3 Assay Capping and Compositing

14.1.3.1 High-grade Capping

Basic univariate statistics were performed on raw assay datasets grouped by rock code. Assay capping was undertaken before compositing for each mineralized domain and sub-domain, and capping levels were chosen primarily using probability plots.

A total of 209 samples were capped using the determined capping limits, which represents a metal loss factor of 9.2%. Table 14.3 presents a summary of the statistical analysis for each domain for the raw assays. Figure 14.8 and Figure 14.9 show example probability plots for the high-grade sub-domains of the F-Zone and North 1-Zone.

14.1.3.2 Arsenic and Sulfur Database

Basic univariate statistics were performed on raw arsenic (As) and sulfur (S) assay datasets grouped by domain using raw analytical assay data, for a total of 16,144 samples for As and 16,370 samples for S.

Table 14.4 presents a summary of the median analysis and capping values applied on raw As and S assays for each mineralized domain.

Table 14.3: Summary Statistics of Raw Assays by Domain and Sub-domain

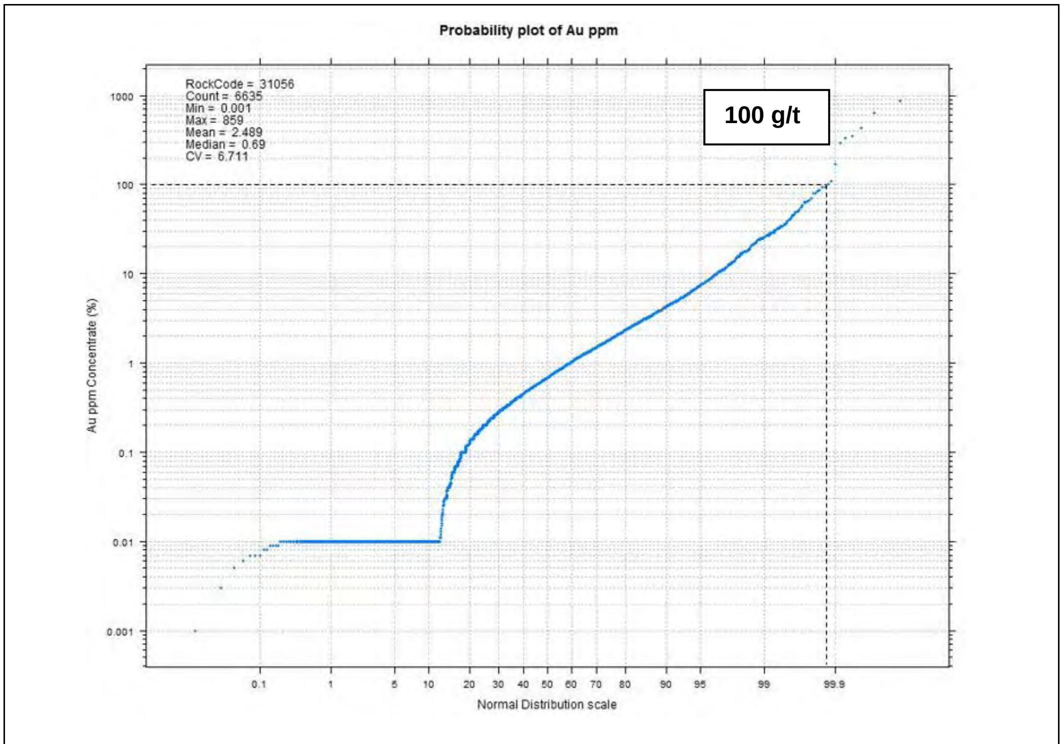
Zone	Principal Domain	Sub-domain	Number of Samples	Max (g Au/t)	Uncut Mean (g Au/t)	High Grade Capping	Cut Mean (g Au/t)	# Samples Cut	% Samples Capped	% Loss Metal Factor
F-Zone	3105	31050	123	1.29	0.04	1	0.04	1	0.81%	-6%
		31051	1,965	116.0	0.35	3	0.28	12	0.61%	-15%
		31056	6,635	859.0	2.49	100	2.13	9	0.14%	-10%
North 1-Zone	3205	32050	752	6.41	0.06	1	0.04	4	0.53%	-22%
		32051	5,209	14.1	0.27	6	0.27	6	0.12%	-1%
		32056	6,605	402.0	2.31	110	2.21	6	0.09%	-4%
	3210*	3210	547	53.4	1.97	20	1.66	11	2.01%	-14%
Central-Zone	3300	33000	80	0.87	0.08	0.87	0.08	-	-	0%
		33001	1,620	10.3	0.33	3.5	0.32	5	0.31%	-1%
		33006	1,876	436.0	2.56	60	2.01	13	0.69%	-20%
New Domain	3305	33051	176	77.2	0.72	4	0.30	2	1.14%	-54%
		33056	556	50.1	1.89	15	1.74	9	1.62%	-8%
F2-Zone	3405	34050	273	3.61	0.05	1	0.04	1	0.37%	-16%
		34051	1,733	7.70	0.31	3	0.30	8	0.46%	-2%
		34056	2,455	251.0	2.30	90	2.15	5	0.20%	-5%
SP-Zone	3500	35000	185	46.6	0.45	2	0.19	4	2.16%	-59%
		35001	6,640	51.1	0.35	4	0.33	10	0.15%	-1%
		35006	8,973	2,363.4	2.19	125	1.90	8	0.09%	-2%
North 2-Zone	3600	3600	2,135	2,000.0	3.26	50	2.32	3	0.14%	-27%
	3605*	3605	1,371	39.4	1.34	20	1.29	9	0.66%	-3%
New Domain	3610	3610	303	234.0	1.93	20	0.97	3	0.99%	-52%
New Domain	3620	3620	56	3.76	0.69	3	0.68	1	1.79%	-2%
North 3-Zone	3710	3710	1,196	511.0	1.77	20	1.11	5	0.42%	-32%
	3720	3720	325	36.1	1.79	15	1.62	4	1.23%	-9%
Lower-Zone	3800	3800	654	34.3	1.18	15	1.14	4	0.61%	-3%
A -Zone	3900	3900	1,106	59.3	1.37	20	1.21	10	0.90%	-5%
Tenacity-Zone	4000	4000	1,775	1,560.0	2.67	15	1.25	14	0.79%	-31%
SP2-Zone	4100	4100	499	156.0	2.48	30	2.06	5	1.00%	-20%
External Grade Shells	500	500	194,025	19.9	0.03	5	0.03	9	0.00%	-1%
	501	501	65,280	21.0	0.25	6.5	0.24	17	0.03%	0%
	506	506	6,981	2,870.0	3.38	140	2.50	11	0.16%	-17%
Total			**322,109**	**2,870.0**	**0.48**	**Variable**	**0.43**	**209**	**0.06%**	**-9.20%**

Notes:

** 2016 domain 3205 was split into two domains (3205 and 3210)*

** 2016 domain 3600 was split into two domains (3600 and 3605)*

Figure 14.8: Probability Plot Au (g/t) – High-grade Sub-domain (> 0.6 g/t) of the F-Zone (upper image) and North 1-Zone (lower image) Zone



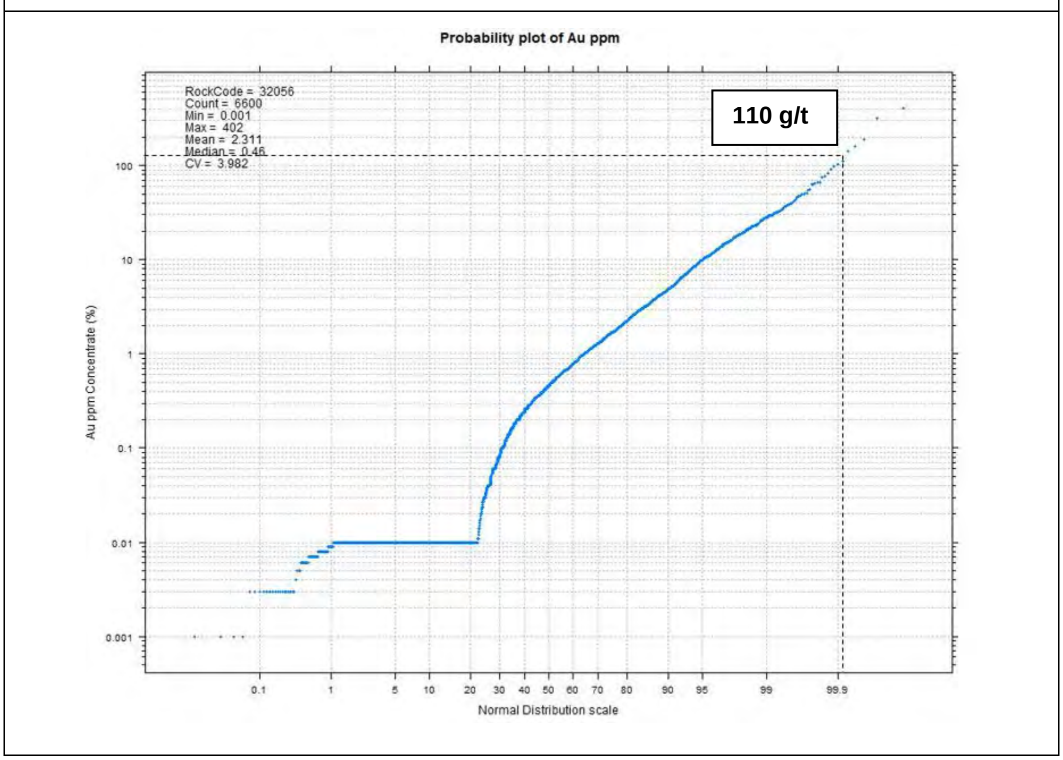


Figure 14.9: Probability Plot Au (g/t) – High-grade Sub-domain (> 0.6 g/t) of the F2-Zone (upper image) and SP-Zone (lower image) Zone

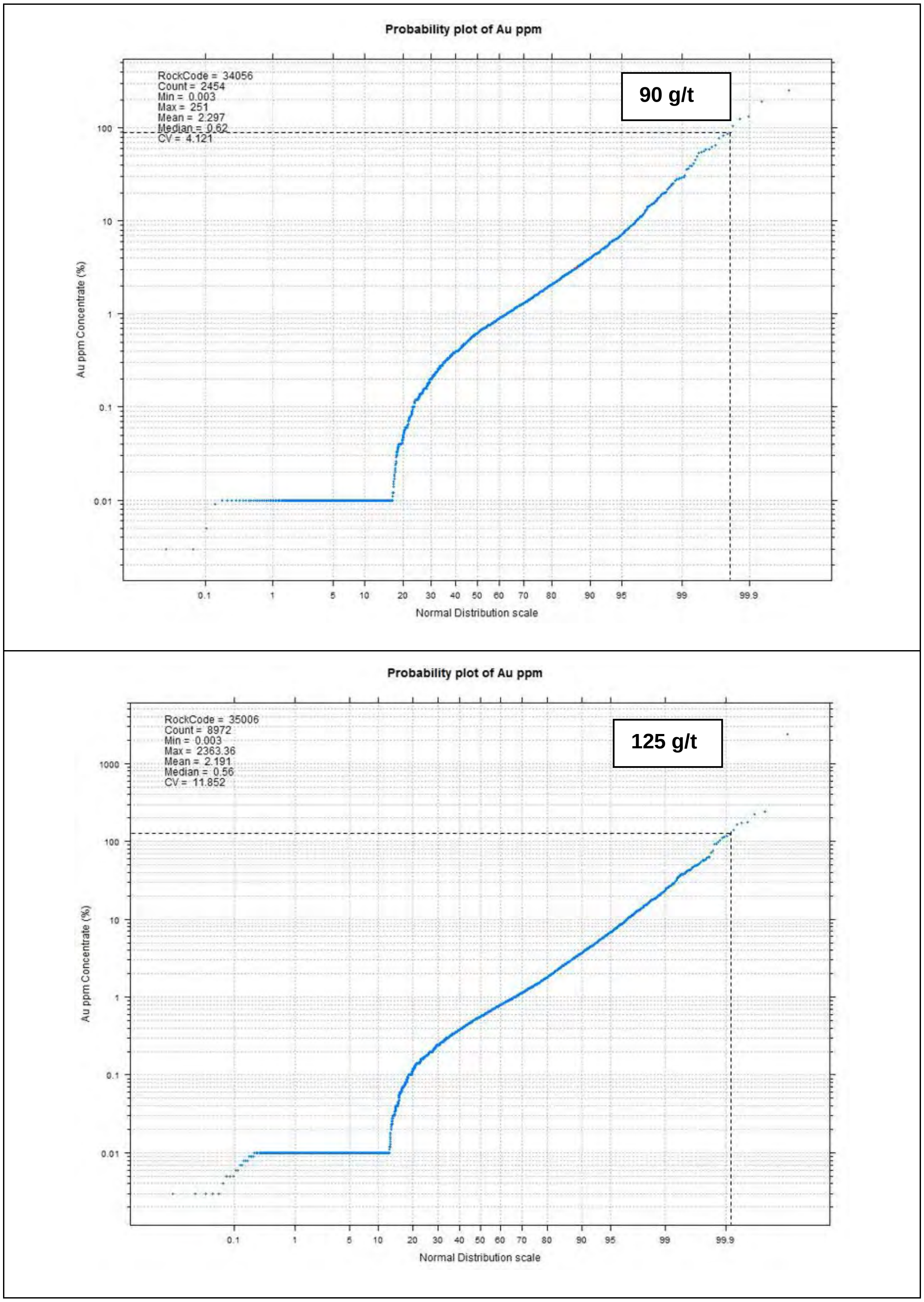


Table 14.4: Summary Statistics for As (ppm) and S (%) Raw Assays by Domain

Zone	Principal Domain	Block Code	Number of Assays (As)	Median As (ppm)	As (ppm) Capping Level	Number of Assays (S)	Median S (%)	S (%) Capping Level
F-Zone		31050	12	73	N/A	12	0.2	N/A
	3105	31051	111	61	N/A	117	0.5	1.1
		31056	672	78	10,000	677	0.7	4.4
North 1-Zone	3205	32050	17	234	N/A	18	0.2	N/A
		32051	340	23	6,000	360	0.5	7
		32056	653	204	10,000	662	1.1	14
	3210*	3210	91	397	3,500	91	1.4	7
Central-Zone	3300	33000	4	115	1,000	4	0.5	N/A
		33001	167	445	10,000	172	0.6	1.8
		33006	341	300	10,000	354	0.9	3.2
New Zone	3305	33051	32	200	N/A	32	0.2	1
		33056	102	7	1,200	102	0.6	8.9
F2-Zone	3405	34050	3	12	N/A	3	0.4	N/A
		34051	133	111	5,800	133	0.6	1.8
		34056	306	182	10,000	306	0.8	2.9
SP-Zone	3500	3500	1,781	291	N/A	1,781	0.6	N/A
North 2-Zone	3600	3600	261	104	10,000	261	2.5	N/A
	3605*	3605	167	365	6,000	177	0.7	10
New Zone	3610	3610	23	203	N/A	23	0.4	N/A
North 3-Zone	3710	3710	251	236	N/A	251	0.5	7.2
	3720	3720	59	258	N/A	59	0.5	6.6
Lower-Zone	3800	3800	95	91	N/A	95	0.5	1.9
A-Zone	3900	3900	49	12	N/A	49	0.3	0.6
Tenacity-Zone	4000	4000	22	3	N/A	22	0.7	N/A
External Grade Shells	500	500	6,494	32	4,500	6,613	0.2	3.8
	501	501	3,465	19	7,000	3,491	0.3	5
	506	506	493	42	6,000	505	0.5	11
TOTAL			**16,144**			**16,370**		

Notes:

** 2016 domain 3205 was split into two domains (3205 and 3210)*

** 2016 domain 3600 was split into two domains (3600 and 3605)*

14.1.3.3 Compositing

In order to minimize any bias introduced by the variable sample lengths, the capped gold assays of the drill hole data were composited. The typical original sample length of the diamond and RC drilling are 1.5 m and 2.0 m respectively. GMS chose a composite length of 2 m, which is the sampling interval used for the RC drilling in 2018 and 2019.

Composites of 2.0 m (down hole) were generated for all mineralized domains, with composite residuals retained. Domain boundaries were used during compositing (i.e. composites were broken on wireframe contacts, creating composite residuals less than 2.0 m in length).

GMS found that retaining composite residuals during grade estimation softens the boundary between the internal grade shells, as the internal grade shells are not snapped to sampling intervals. This results in some lower-grade composite residuals (< 0.6 g/t) being retained around the inside edge of the > 0.6 g/t internal grade shell (see Table 14.5). GMS also ran numerous sensitivity grade estimates and found that the inclusion of composite residuals resulted in a more conservative MRE (lower grade, and less overall ounces) compared to excluding the residuals < 0.5 m in length. These are discussed further in Subsection 14.1.15.

Table 14.5: Statistics of Composites Grouped by Length for the > 0.6 g Au/t Internal Grade Shell for the SP-Zone

Composite Length	Number of Composites	Minimum (g Au/t)	Maximum g Au/t	Mean (g Au/t)	Standard Deviation
< 0.5 m	267	0.000	6.460	0.411	0.678
0.5 – 1.0 m	234	0.000	8.802	0.557	0.921
1.0 – 1.99 m	486	0.000	51.581	1.101	3.510
2.0 m	5918	0.000	124.965	1.692	4.179

The total number of composites used in the MRE is 284,926. A grade of 0.00 g Au/t was assigned to missing sample intervals during compositing, however unsampled intervals within voids were removed from the estimation. Table 14.6 summarize the basic statistics of the gold composites used for the 2019 MRE.

Table 14.6: Summary Statistics for the 2.0 m Composites

Zone	Principal Domain	Sub-domain	Number of Composites	Max (g Au/t)	Mean (g Au/t)	Standard Deviation	Coefficient of Variation	Coefficient of Variation (Au>0.1 g/t)
F-Zone	3105	31050	104	0.88	0.05	0.12	2.64	0.81
		31051	1,608	3.00	0.28	0.36	1.30	0.85
		31056	5,079	100.00	1.86	4.22	2.27	2.12
North 1-Zone	3205	32050	648	0.66	0.04	0.09	2.06	0.60
		32051	4,054	6.00	0.26	0.40	1.53	1.00
		32056	4,747	77.27	1.94	4.37	2.25	1.98
	3210	3210	431	20.00	1.49	3.00	2.02	1.58
Central-Zone	3300	33000	73	0.78	0.08	0.14	1.79	0.68
		33001	1,246	2.73	0.32	0.37	1.13	0.78
		33006	1,345	59.97	1.81	4.54	2.51	2.26
New Zone	3305	33051	134	4.00	0.29	0.57	1.95	1.10
		33056	397	11.86	1.57	2.37	1.51	1.11
F2-Zone	3405	34050	230	0.44	0.04	0.06	1.59	0.52
		34051	1,365	2.85	0.29	0.35	1.19	0.80
		34056	1,775	59.00	1.93	4.52	2.34	2.16
SP-Zone	3500	35000	193	2.00	0.17	0.28	1.71	0.89
		35001	5,570	3.13	0.33	0.31	0.95	0.72
		35006	6,905	124.97	1.58	4.01	2.55	2.38
North 2-Zone	3600	3600	1,438	40.44	1.96	3.98	2.03	1.69
	3605	3605	936	20.00	1.10	2.10	1.91	1.53
New Zone	3610	3610	217	16.23	1.06	2.43	2.29	1.91
New Zone	3620	3620	43	2.26	0.66	0.54	0.81	0.77
North 3-Zone	3710	3710	821	15.05	1.06	1.74	1.64	1.43
	3720	3720	236	13.70	1.53	2.21	1.44	1.26
Lower-Zone	3800	3800	477	14.99	1.13	1.73	1.53	1.32
A-Zone	3900	3900	795	16.86	1.01	1.51	1.50	1.37
Tenacity-Zone	4000	4000	1,190	15.00	1.09	1.55	1.43	1.19
SP 2-Zone	4100	4100	337	23.62	1.90	3.40	1.79	1.47
External Grade Shells	500	500	185,708	5.00	0.03	0.06	2.59	1.11
	501	501	51,596	5.47	0.22	0.30	1.38	0.90
	506	506	5,228	105.09	1.93	4.91	2.54	2.15
Total			**284,926**					

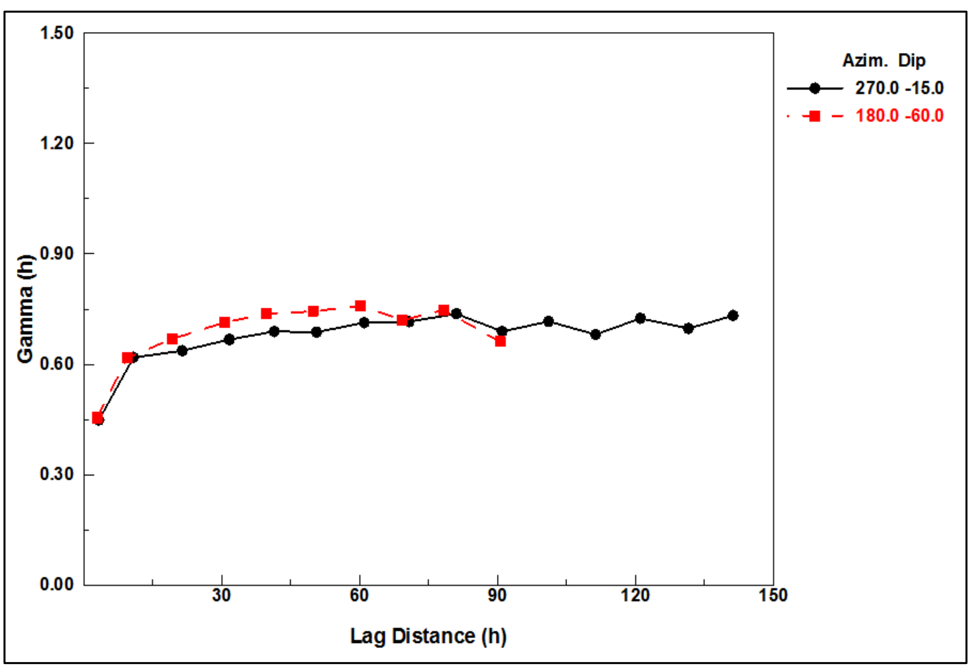
14.1.4 Variography

Three-dimensional (3D) directional variography was completed on the principal domains using the 2.0 m composites of the capped gold values. The variographic analysis was performed in SAGE2001™. The GMS approach to model the variograms is described as follows:

- Log-normal variograms, correlograms and pairwise-relative variograms were compared for each of the mineralized domains containing sufficient data to confidently estimate the ranges of the various axes of continuity. Internal grade shells were grouped during variography;
- Evaluation of the nugget effect based on the down hole variogram;
- Pairwise-relative variograms were chosen for modelling the major, semi-major and minor axes as they showed the clearest structure.

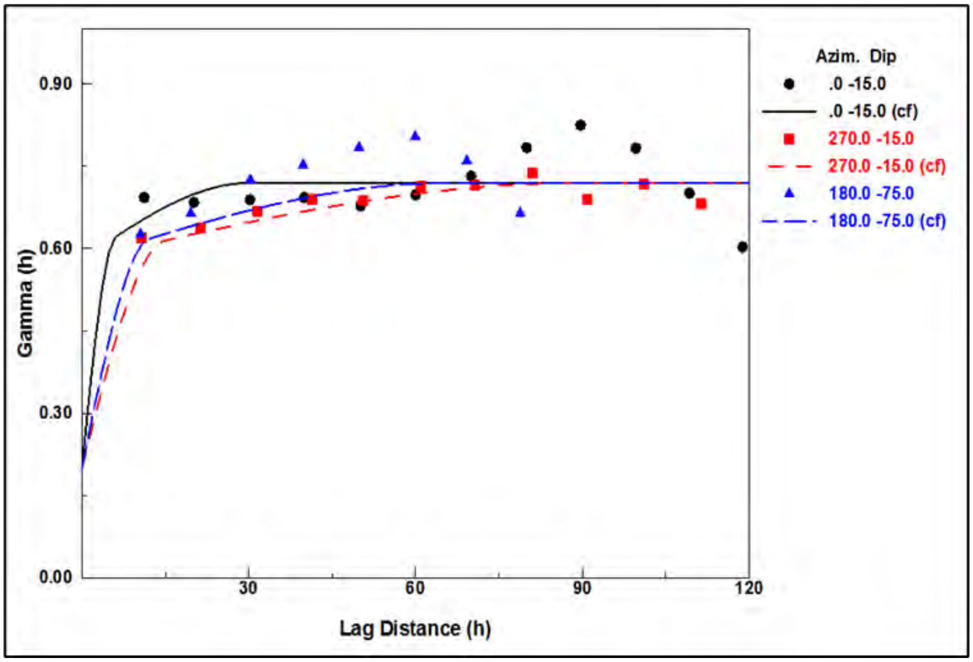
Figure 14.10 and Figure 14.11 illustrate an example of the directional pairwise-relative variogram and the variogram model for the SP-Zone. Variogram models for the F-Zone and SP-Zone were derived solely from the RCGC drilling, and variograms from the North 1, North 2, Central and F2-Zones were derived from all composites.

Figure 14.10: Pairwise-Relative Variogram for the SP Zone (black = major axis, red = semimajor axis)



*Notes: * Along Strike direction in black (270/-15) and Down Dip direction in red (180/-60)*

Figure 14.11: Pairwise-Relative Model for the SP Zone (Domain 3500)



*Notes: * S1 Range – Major = 15 m, Semi = 12m, Minor = 6 m, S2 Range – Major = 90 m, Semi = 65 m, Minor = 30 m*

The selected variogram model parameters are tabulated in Table 14.7.

Table 14.7: Variogram Model Parameters for Domain

Zone	Domain	Axis	Nugget (C0)	Sill 1 (C1)	Range 1 (R1)	Sill 2 (C2)	Range 2 (R2)
F-Zone	3105	X	0.3	0.35	8	0.15	85
		Y	0.3	0.35	8	0.15	85
		Z	0.3	0.35	6	0.15	35
North 1-Zone	3205	X	0.35	0.5	15	0.2	60
		Y	0.35	0.5	12	0.2	40
		Z	0.35	0.5	6	0.2	20
Central-Zone	3300	X	0.4	0.45	15	0.11	80
		Y	0.4	0.45	15	0.11	80
		Z	0.4	0.45	6	0.11	30
F2-Zone	3405	X	0.4	0.4	15	0.18	60
		Y	0.4	0.4	12	0.18	50
		Z	0.4	0.4	6	0	25
SP-Zone	3500	X	0.2	0.38	15	0.14	90
		Y	0.2	0.38	12	0.14	65
		Z	0.2	0.38	6	0.14	30
North 2-Zone	3600	X	0.45	0.5	15	0.13	75
		Y	0.45	0.5	12	0.13	50
		Z	0.45	0.5	6	0.13	25

14.1.5 Search Ellipsoids

Search ellipse dimensions were defined and based on observed variogram ranges as described below:

- First pass ≈ 80% of the variogram range;
- Second pass ≈ 100% to 120% of the variogram range;
- Third pass ≈ 150% to 200% of the variogram range (or ensuring that the majority of remaining blocks are interpolated;

- If anisotropy was observed in the variograms (i.e. major range was longer than semi-major range, or vice-versa) then these were applied to the search ellipse dimensions.

Each domain was estimated using a single search ellipse orientation, customized to the dip and dip direction of the domain wireframe. The SP-Zone was divided into four search ellipse sub-domains, and the North 2-Zone was divided into two search ellipse sub-domains (Figure 14.12), using soft boundaries, to ensure the search ellipse was orientated with the different directions of grade continuity.

Search ellipse orientations for the external grade shells (rock codes 500, 501 and 506) were assigned based on the six search ellipse sub-domains (Figure 14.13). They were modelled based on changes in dip of the iron-formations and stratigraphy.

Search ellipse orientations were determined for each domain using a combination of stereonets of wireframe face dips and strikes, and visualisation of search ellipses in three dimensions

Figure 14.12: Section Views (looking west) of Wireframe Subdivisions (soft boundaries) to Guide Search Ellipses. Left: SP-Zone, Right: North 2-Zone

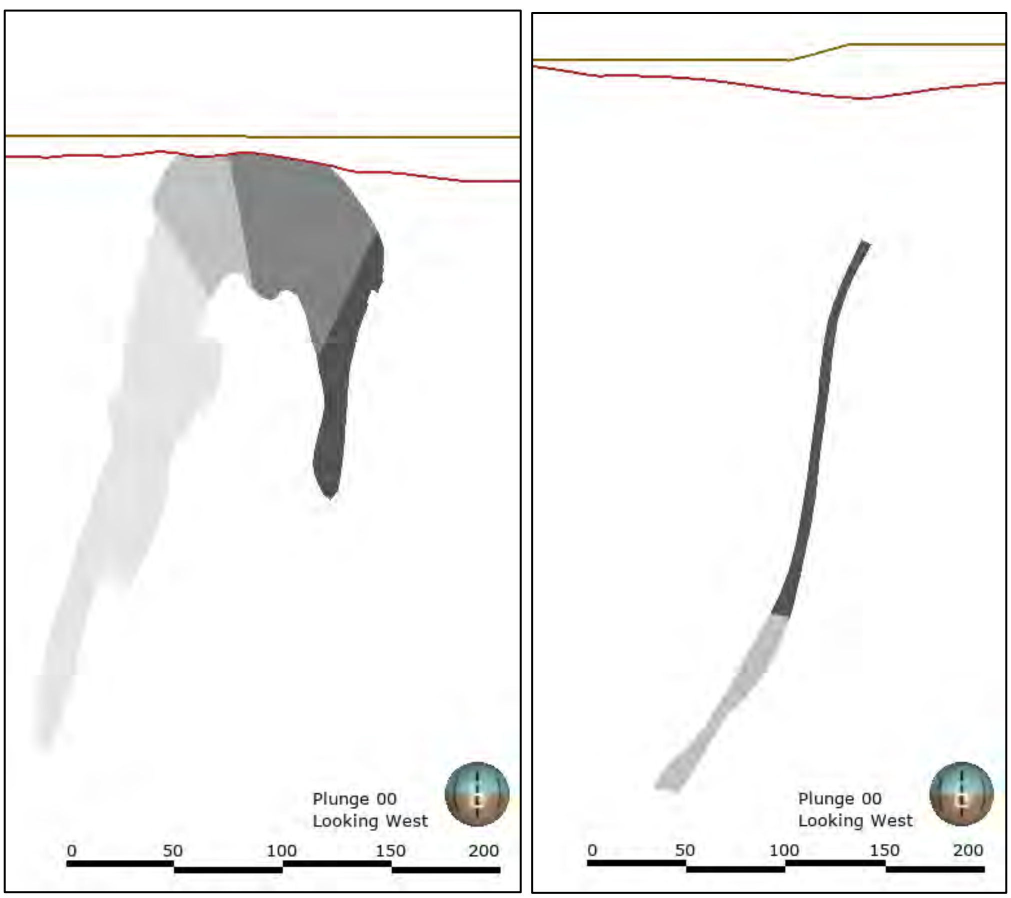
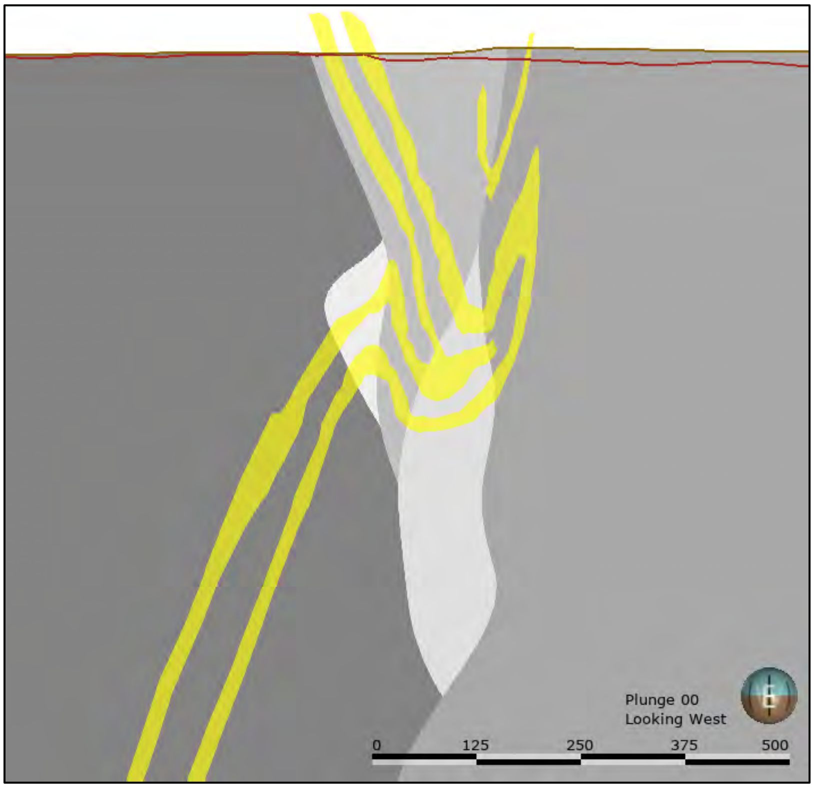


Figure 14.13: Section 504,325 mE (looking west) of Search Ellipse Sub-domains (grey shades) for External Grade Shell Domains 500, 501 and 506. Iron-Formations shown in Yellow

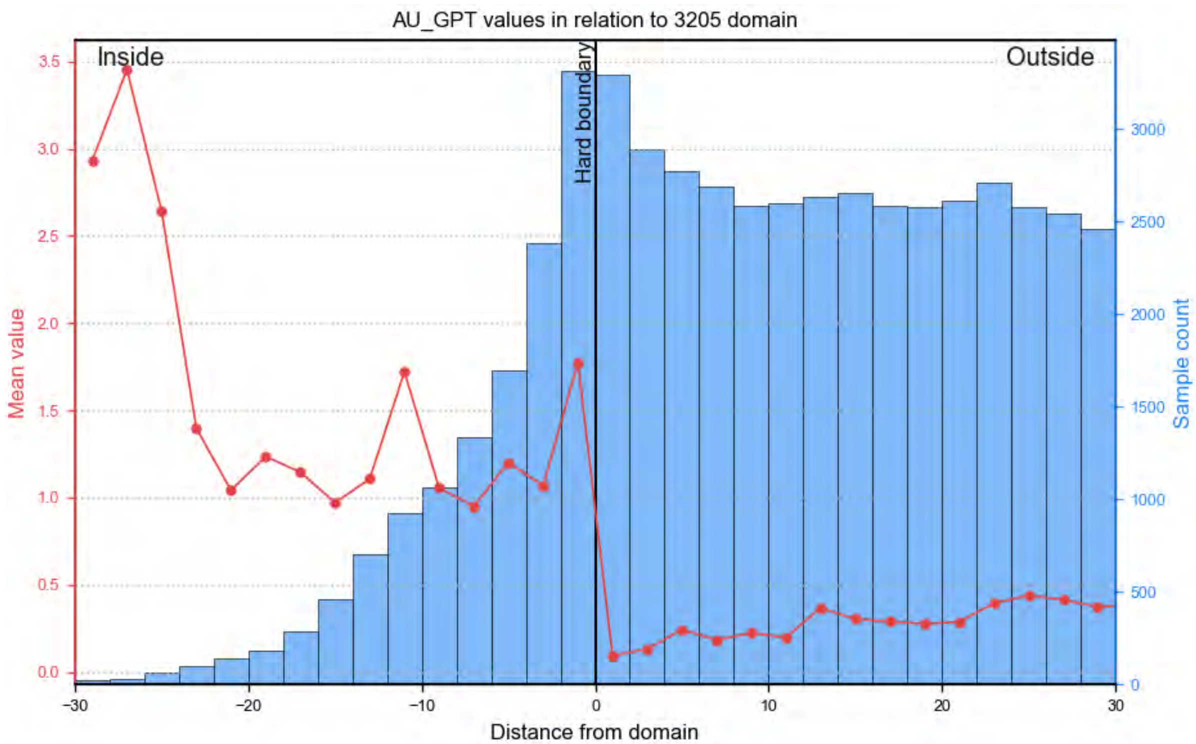


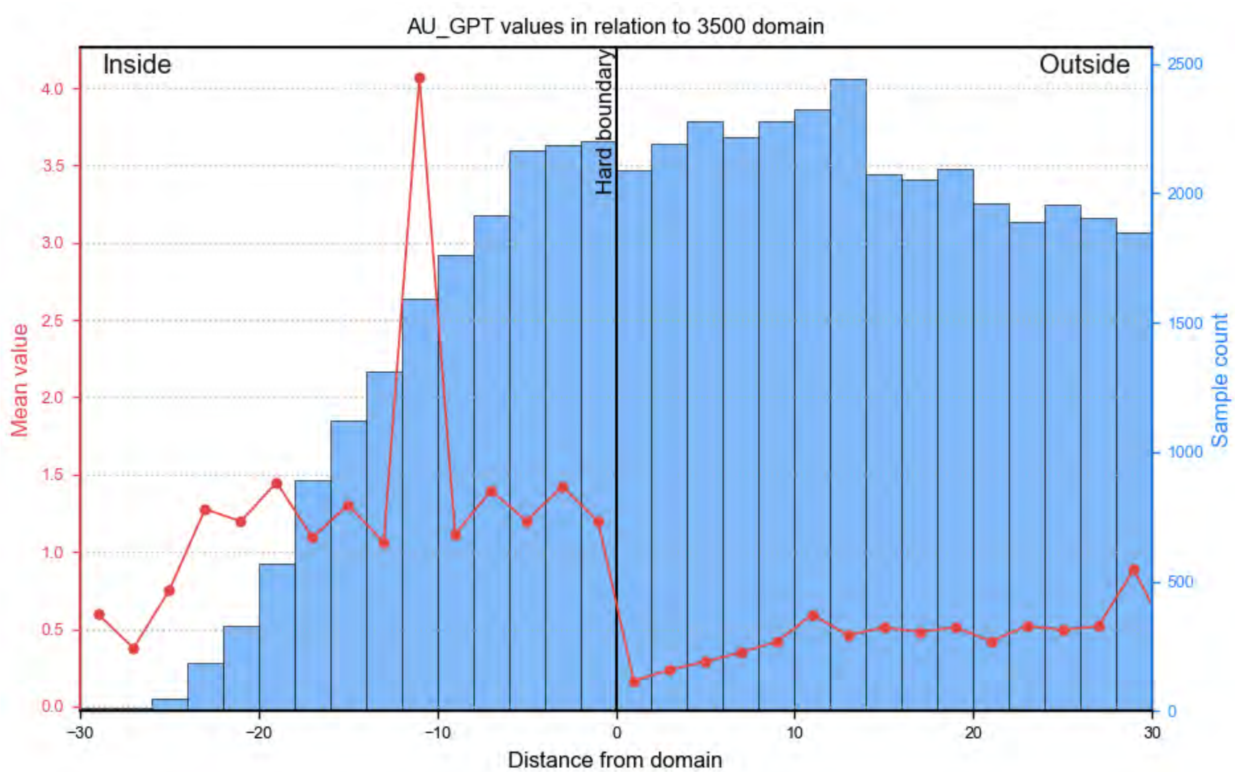
14.1.6 Domain Boundaries

GMS undertook contact analysis in Leapfrog GEO™ on the 17 principal domains and found that hard contacts exist in virtually all cases. This is consistent with the styles of mineralization observed at the Project. In regards to the internal grade shells of the 17 principal domains, GMS believes that the semi-soft boundary effect observed by including the composite residuals discussed in Subsection 14.1.3.3 is appropriate to ensure that some "softness" is retained during the estimation.

As the North 1 (3205) and North 2 (3600) Zones represent the two limbs of a syncline, GMS decided to treat the boundary between these two domains as completely soft. This ensures that blocks estimated within the nose of the syncline can "see" composites from both domains. Away from the fold nose, sufficient distance exists between the two domains to ensure that composites are not shared.

Figure 14.14: Contact Analysis Plots of the North 1-Zone (top) and the SP-Zone (bottom)





GMS applied hard boundaries during estimation for all domains apart from the North 1 and the North 2 Zones (as previously mentioned). Soft boundaries were used between the search ellipse sub-domains (SP-Zone, North 1-Zone and external grade shells). GMS also ran sensitivity grade estimations removing the internal grade shells for the 17 principal domains, and the results are shown in Subsection 14.1.15.

14.1.7 Treatment of High Grades

In order to control the influence of isolated high-grade composites during grade estimation, GMS used high-grade restraining (also know as "high-grade restraint" or "transition"). This method involves applying a second, smaller internal search ellipse to restrict the influence of high-grade composites above a user-defined value (a "threshold").

High-grade thresholds were chosen based on probability plots of capped gold grade composites with the objective of identify remaining outliers that require additional restraining. The size of the search restriction was determined as follows:

- Downhole variography indicated that the majority of variability is accounted for in the first 5 m in the sub-horizontal direction;
- Dimensions for the high-grade thresholds were kept consistent for the each of the three passes;
- Anisotropy was applied where necessary;
- High-grade threshold dimensions are generally ½ the first pass dimensions.

In the 17 principal domains, this method was applied only in the second and third estimation passes. The first pass was deemed sufficiently constrained in terms of search ellipse dimensions and other estimation parameters to not require high-grade restraining. For the external grade shells, high-grade restraining was retained for all estimation passes as these zones do not demonstrate sufficient grade continuity between drill sections and require a more conservative approach.

Table 14.8 summarizes the parameters of the final ellipsoids and threshold dimensions used for interpolation.

Table 14.8: Final Search Ellipsoid Parameters and Threshold Dimensions

	Rotation GEMS			Pass 1			Pass 1 HG Thresh.				Pass 2			Pass 2 HG Thresh.				Pass 3			Pass 3 HG Thresh.			
Domain	**Z**	**X**	**Z**	**X**	**Y**	**Z**	**X**	**Y**	**Z**	**HG Thresh. g Au/t**	**X**	**Y**	**Z**	**X**	**Y**	**Z**	**HG Thresh. g Au/t**	**X**	**Y**	**Z**	**X**	**Y**	**Z**	**HG Thresh. g Au/t**
3105	-5	90	15	50	50	15	None				75	75	25	20	20	5	40	120	120	40	20	20	5	40
3205	-5	-80	-20	40	30	15	None				70	50	25	20	15	5	50	100	75	35	20	15	5	50
3210	3	-66	0	40	30	15	None				70	50	25	None				100	75	35	None			
3300	-15	70	20	40	50	15	None				60	75	25	None				100	120	35	None			
3305	-12	62	15	50	40	15	None				70	45	25	None				100	70	35	None			
3405	-5	80	15	40	30	15	None				70	50	25	20	15	5	40	100	75	35	20	15	5	20
3500	Various			40	30	15	None				70	50	25	20	15	5	40	100	75	35	20	15	5	20
3600	-15	67	23	45	30	15	None				70	50	25	None				100	70	35	None			
3605	-13	71	5	50	40	15	None				75	60	25	None				100	80	35	None			
3610	-7	85	0	40	50	15	None				75	60	25	None				100	50	35	None			
3710	-17	73	30	40	40	15	None				75	75	25	None				100	100	35	None			
3720	-9	80	25	40	40	15	None				75	75	25	None				100	100	35	None			
3800	-2	73	15	40	40	15	None				75	75	25	None				100	100	35	None			
3900	-8	62	11	45	30	15	None				70	50	25	None				100	75	35	None			
4000	-7	74	15	40	40	15	None				75	75	25	None				100	100	35	None			
4100	-23	41	18	45	30	15	None				70	50	25	None				100	75	35	None			
500, 501,506	IF-Related			50	35	20	20	15	5	21	80	50	35	20	15	5	21	120	80	50	20	15	5	21
500, 501,506	Sediment-Related			50	50	15	20	20	5	21	75	75	25	20	20	5	21	100	100	40	20	20	5	21

14.1.8 Bulk Density Data

For the 2019 MRE update, a total of 6,937 bulk density measurements were provided by GGM and integrated into the Project database. Bulk density measurements were taken at the Geraldton core shack by GGM staff. Statistical analysis of the density data by estimation domain showed that slight differences exist between mineralized and unmineralized samples within a given domain (based on the internal grade shells). This could be the result of the varying levels of sulphide associated with mineralization affecting the overall bulk density.

For the 17 principal domains, GMS assigned the bulk density by estimation domain. For the external grade shells, bulk density was assigned based on the 2016 lithology interpretation. Median values were used to reduce the influence of outliers.

A density of 2.00 g/cm^3 was assigned to the overburden. A density of 2.05 g/cm^3 was assigned to the tailings, 1.60 g/cm^3 to organics and 1.80 g/cm^3 to fill material.

GGM completed a study of underground workings in 2016, which were classified by confidence level and backfill status. Densities ranging from 0.27 g/cm^3 up to 2.08 g/cm^3 were assigned based on the following types of backfill: Open (filled with water); Waste (mix of mining waste and reject from the processing plant; and Sand. These remain unchanged from the 2016 MRE.

GMS is of the opinion that the bulk density databases is of sufficient quality for Mineral Resource estimation at the Hardrock Deposit.

Table 14.9 presents the bulk density values assigned to the block model.

Table 14.9: Bulk Density Assigned to Block Model by Domain

Zone	Principal Domain	Block Code	Dominant Lithology	Bulk Density (g/cm^3)			
				No. Samples	Median	Min	Max
F-Zone	3105	31050 - 31051	Greywacke	27	2.73	2.58	2.89
		31056		59	2.74	2.61	3.43
North 1-Zone	3205	32050 - 32051	Iron Formation and Greywacke	64	2.76	2.56	3.75
	3210	32056 - 3210		90	2.81	2.57	3.73
Central-Zone	3300	33000 - 33001	Greywacke	46	2.78	2.54	3.43
		33006		40	2.76	2.25	3.59
New Zone	3305	33050 - 33051	Iron Formation	6	3.02	2.70	3.27
		33056	Iron Formation	15	3.12	2.68	3.40
F2-Zone	3405	34050 - 34051	Greywacke	23	2.75	2.55	3.37
		34056		48	2.77	2.59	3.33
SP-Zone	3500	35000 - 35001	Mixed (IF, Porphyry and Greywacke)	64	2.75	2.50	3.30
		35006		74	2.72	2.56	3.33
North 2-Zone	3600-3605	3600	Iron Formation and Greywacke	14	2.92	2.71	3.44
		3605		7	2.75	2.67	3.68
	3610 - 3620	3610 - 3620		14	2.72	2.61	2.92
North 3-Zone	3710 - 3720	3710 - 3720	Iron Formation and Greywacke	28	2.79	2.60	4.05
Lower-Zone	3800	3800	Mixed (IF, Porphyry and Greywacke)	8	2.77	2.68	3.27
A-Zone	3900	3900	Porphyry and Iron Formation	10	2.74	2.59	2.94
Tenacity-Zone	4000	4000	Greywacke	12	2.73	2.57	3.57
SP2-Zone	4100	4100	Greywacke	7	2.76	2.58	2.78
Porphyry	-	8100	Porphyry	539	2.73	2.31	3.62
Conglomerate	-	9100, 9200	Conglomerate	144	2.75	2.47	3.41
Conglomerate	-	10100 - 10400	Conglomerate	45	2.74	2.53	3.77
IF North 1	-	11100 - 11160	Iron Formation	242	3.06	2.54	3.80
IF North 2	-	11200	Iron Formation	210	2.76	2.45	3.61
IF North 3	-	11300	Iron Formation	38	2.76	2.52	3.48
Lower IF	-	12000	Iron Formation	159	2.78	2.50	3.71
Middle IF	-	13100 - 13200	Iron Formation	10	3.28	2.72	3.94
Upper IF	-	14100 - 14300	Iron Formation	95	2.73	2.58	3.72
Ultramafic	-	15000	Ultramafic	88	2.89	2.50	3.79
North Gabbro	-	16000	Gabbro	216	2.75	2.54	3.52
South Gabbro	-	17100 - 17400	Gabbro	392	2.75	0.28	3.50
Greywacke	-	18000	Greywacke	1311	2.74	2.15	3.63
Background	-	20000	-	2791	2.74	0.98	3.92

14.1.9 Block Model

A block model was constructed in Geovia GEMS™ using a regular block size and percentage attributes. The block model was extended to cover the planned open pit and all underground mineralization. The model has been pushed down to a depth of approximately 1,800 m below surface. The block model was not rotated (Y-axis oriented towards north). The block dimensions were chosen considering the sizes of the mineralized zones and the proposed mining bench height (10 m). Table 14.10 presents the properties of the block model.

Table 14.10: Block Model Properties

Description		Number of Blocks	Block Size (m)	Dimension (m)		Rotation	Origin (UTM NAD83, Zone 16)	
Volum_ID19	Colum	575	10	Width	5,750	0	East	501,050
	Row	340	5	Length	1,700		North	5,502,000
	Level	192	10	Height	1,920		Elevation	500

Notes: The block model origin is the upper south-west corner of the block model

The seventeen mineralized domains (and their internal grade shell sub-domains) as well as the external grade shell domains were coded in one block model folder using the majority 50/50 rule for the attribution of a block code. Precedence was respected during the process. Checks were undertaken to ensure that for a given domain, the volume of the coded blocks is accurate when compared to the volume of the input wireframe. A percentage attribute was calculated to remove the underground workings, overburden and the various tailings volumes from the MRE.

The block model folders are summarised below:

- Zones_5050 – Main folder containing MRE attributes;
- Fill – Fill component of tailings dump, zero grade;
- Organics – Organic component of tailings dump, zero grade;
- Openings_All – Underground workings with varying densities and grades due to backfill of tailings underground. Remains unchanged from 2016 apart from new void additions near-surface;
- Tailings – Tailings component of tailings dump, zero grade;
- Overburden – Overburden volume, zero grade;
- Air – Volume above topography surface, zero grade.

A series of attributes were added for the Zones_5050 folder in the block model and are presented in Table 14.11. These are incorporated into the block model to capture the various attributes needed during the block modelling development.

Table 14.11: Hardrock Block Model – Zones_5050 Folder

Folder	Attribute Name	Attribute Description
Zones_5050	RockType19_GS	Domain coding (500 to 36002)
	Density19	Specific gravity (2.72 to 3.28 g/cm^3)
	Percent19_NEW	Percent block attribute (subtracts UG workings and overburden)
	AU_CUT_19_GS_FINAL	Gold capped grades (in grams per tonne (g/t))
	CATEG_FINAL	Resource classification (1 = Measured, 2 = Indicated and 3 = Inferred)
	IN_PIT_2019	Within 2019 Optimized Shell (MII) boundary
	S_PCT_19	Sulfur grades (in percent (%))
	AS_PPM_19	Arsenic grades (in parts per million (ppm))
	DIST_19	Actual distance to closest composite

14.1.10 Grade Estimation

Two interpolation methods, Inverse Distance Cubed (“ID^3”) and Ordinary Kriging (“OK”), were considered to perform the Mineral Resource estimate. The ID^3 method was selected for the final resource estimation for all zones due to the levels of internal dilution that remain in the principal domains.

The 2 m composites were assigned rock codes and block codes corresponding to the mineralized domains or sub-domains (internal grade shells) in which they occur. Hard and soft boundaries were applied as described in Subsection 14.1.6. The search/interpolation ellipse orientations and ranges defined in the interpolation profiles used for the grade estimation correspond to those developed in Subsection 14.1.5 (Table 14.8). Other specifications to control grade estimation are as follows:

- Pass 1
  - Minimum of seven (7) and maximum of fifteen (15) composites in the search ellipse for interpolation;
  - Maximum of three (3) composites from any one drill hole;
  - Minimum of three (3) drill holes required for interpolation of the given block.
- Pass 2
  - Minimum of four (4) and maximum of fifteen (15) composites in the search ellipse for interpolation;
  - Maximum of three (3) composites from any one drill hole;
  - Minimum of two (2) drill holes required for interpolation of the given block.
- Pass 3
  - Minimum of three (3) and maximum of fifteen (15) composites in the search ellipse for interpolation;
  - Maximum of three (3) composites from any one drill hole;
  - Minimum of one (1) drill hole required for interpolation of the given block.

The estimation of block grades is illustrated on a plan view and a cross section on Figure 14.15 and Figure 14.16.

Figure 14.15: Plan View at 135 RL showing Estimated Block Grades of the Hardrock Deposit

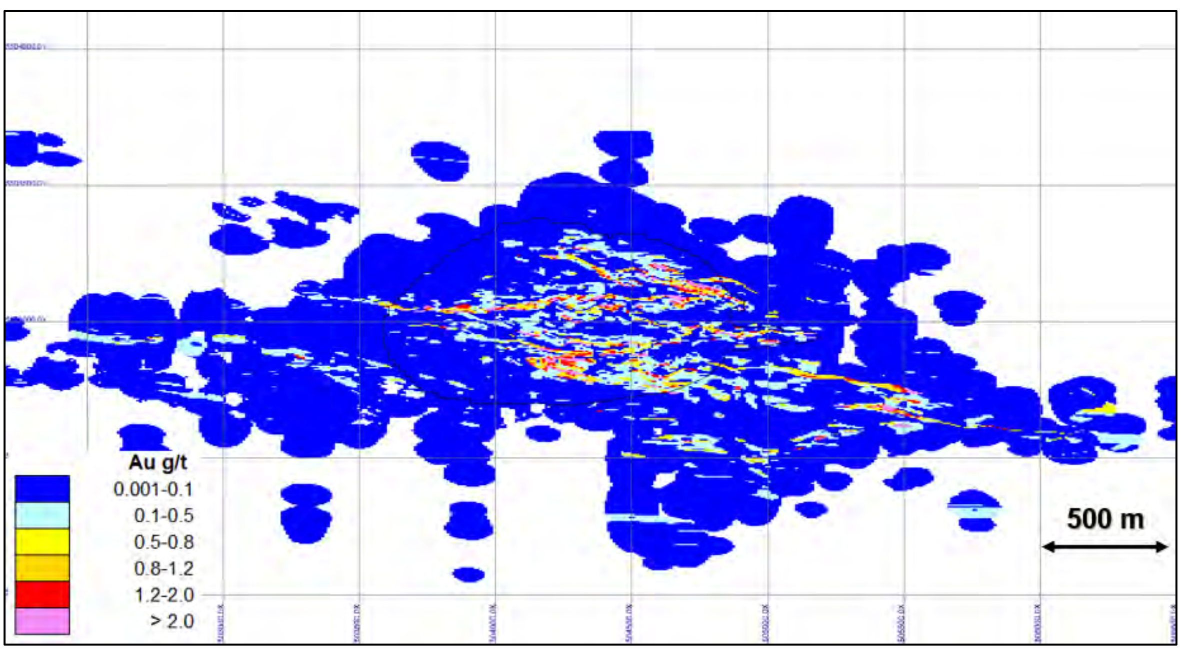
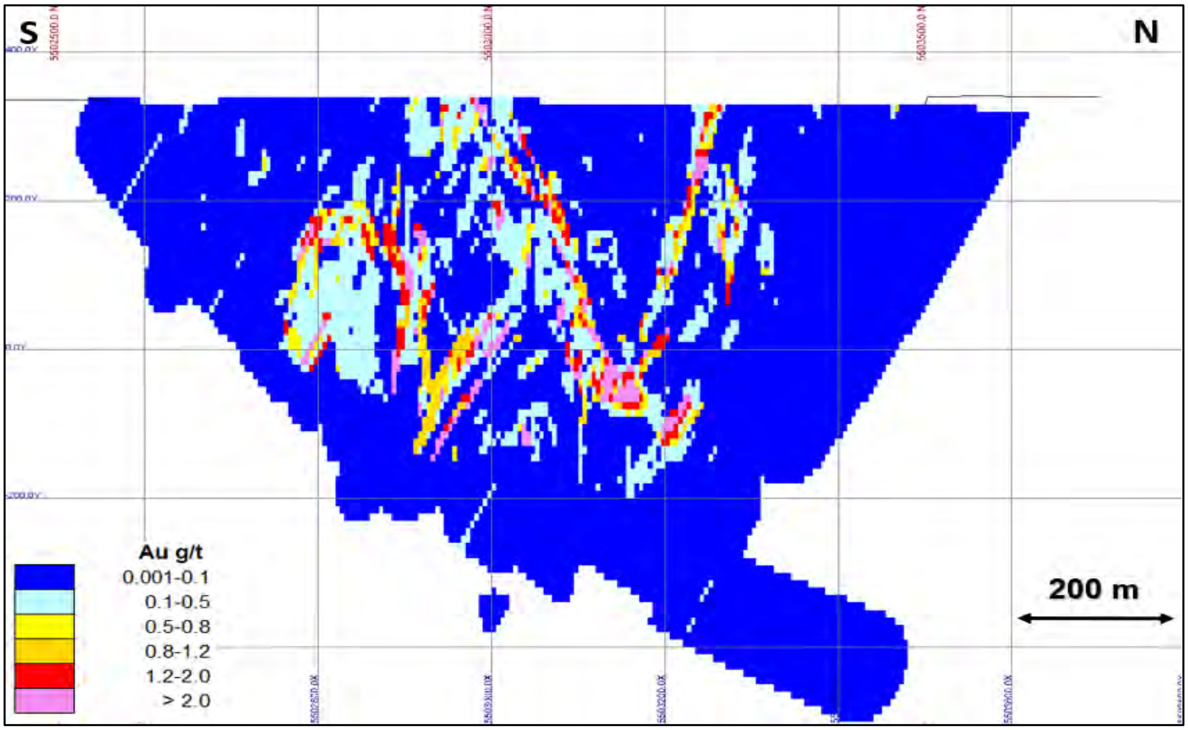


Figure 14.16: Section 504,325 mE (looking west) showing Estimated Block Grades of the Hardrock Deposit



14.1.11 Block Model Validation

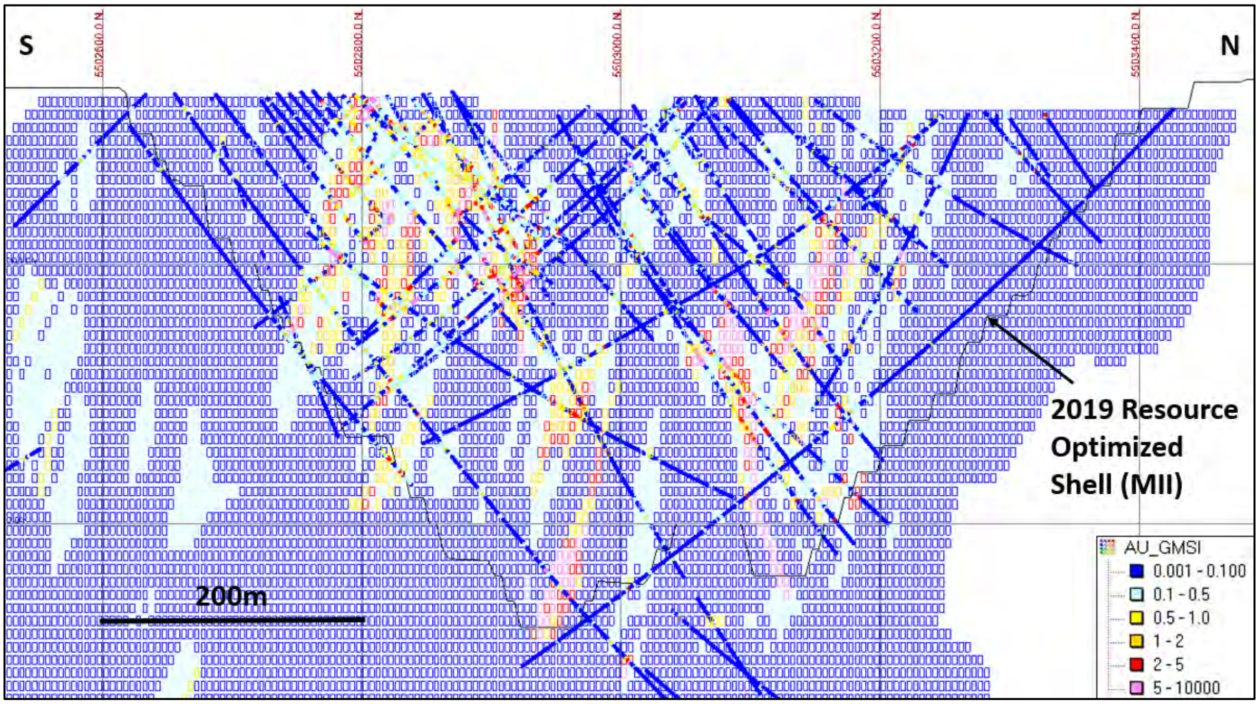
Various block model validation steps were taken to ensure that the block model is a robust representation of the composites. The following validations were undertaken:

- Ensure that the various block model percentages in the folders aggregate to 100%;
- Ensure that the volumes of the blocks in the various domains were representative of the input wireframes;
- Ensure that sufficient blocks are estimated in the various estimation passes;
- Visual checks on-section comparing composite gold grades against block gold grades;
- Global statistical checks comparing the gold grades of the block model against the declustered composite data;
- Local statistical checks to identify any over-smoothing or areas of grade over-extrapolation
- Sensitivity estimates to determine the impact of the capping, high-grade restraining, the interpolator and domain boundaries.

14.1.11.1 Visual Validation – Composite Grades vs. Block Grades

Visual comparisons of block grades and composites in cross section and plan view generally provide a correlation between block grades and drill intersections. No excessive over-extrapolation of grade was observed, and the block grades were found to be a good representation of the composite grades.

Figure 14.17: Blocks vs. Composites - Section 504,675 Em (looking west). The 2019 MII MRE Pit and Optimization



14.1.11.2 Global Statistical Validation

Table 14.12 compares the mean block and declustered composite grades for the mineralized domains considering Pass 1, 2 and 3 within the 2019 MII pit optimization of the deposit. For the seventeen principal domains, the declustered capped composite mean and block mean are very similar (1.19 g/t vs. 1.18 g/t).

Table 14.12: Comparison of the Block and Composite Mean Gold Grades within In-Pit Area for the Mineralized an External Grade Shell Domains

	Composites							Blocks				
Block Code	**No. Obs**	**Min**	**Max**	**Mean**	**Mean Declustered**	**CV**	**CV > 0.1 g/t**	**No. Blocks**	**Min**	**Max**	**Mean**	**CV**
31051	1,387	0.002	3.00	0.28	0.33	1.25	0.84	2,681	0.01	1.84	0.3	0.76
31056	3,838	0.001	100.00	1.77	1.88	2.29	2.16	9,751	0.02	36.08	1.63	1.06
32051	3,155	0.000	6.00	0.26	0.29	1.50	0.97	8,005	0.00	2.54	0.27	0.75
32056	2,986	0.000	77.27	1.91	2.00	2.40	2.10	13,851	0.01	42.01	1.86	1.09
33001	1,201	0.005	2.73	0.32	0.36	1.13	0.77	3,283	0.01	1.55	0.33	0.56
33006	1,253	0.003	59.97	1.78	1.85	2.55	2.31	5,687	0.01	39.57	1.73	1.27
33051	129	0.005	4.00	0.3	0.38	1.94	1.08	385	0.01	2.23	0.43	0.98
33056	397	0.003	11.86	1.57	1.74	1.51	1.11	2,268	0.01	10.35	1.74	0.68
34051	1,275	0.005	2.24	0.29	0.33	1.16	0.78	4,502	0.01	1.42	0.27	0.56
34056	1,183	0.000	59.00	1.74	1.85	2.28	2.13	4,885	0.01	33.46	1.77	1.06
35001	4,829	0.000	3.13	0.33	0.34	0.92	0.70	7,591	0.00	6.04	0.33	0.57
35006	5,709	0.000	124.97	1.55	1.51	2.57	2.41	13,511	0.00	52.48	1.54	1.23
3210	431	0.001	20.00	1.49	1.36	2.02	1.58	624	0.01	10.59	1.48	1.07
3600	1,436	0.000	40.44	1.95	1.98	2.03	1.69	5,552	0.01	26.95	1.81	1.07
3605	894	0.007	20.00	1.11	1.25	1.90	1.52	2,433	0.02	8.60	1.06	1.01
3610	213	0.000	16.23	1.05	1.28	2.33	1.92	824	0.01	8.52	0.91	1.09
3620	43	0.076	2.26	0.66	0.68	0.81	0.77	143	0.21	1.57	0.71	0.35
3710	821	0.001	15.05	1.06	1.18	1.64	1.43	3,796	0.01	9.89	1.09	0.82
3720	236	0.002	13.70	1.53	1.61	1.44	1.26	730	0.00	8.75	1.54	0.81
3800	477	0.007	14.99	1.13	1.30	1.53	1.32	1,275	0.02	6.03	1.15	0.74
3900	118	0.010	8.19	0.71	0.74	1.49	1.40	658	0.05	5.22	0.63	0.57
ALL WF	**33,054**	**0.000**	**124.97**	**1.10**	**1.19**	**1.86**	**1.56**	**97,002**	**0.00**	**52.48**	**1.18**	**1.38**
501	28,696	0.000	3.52	0.22	-	1.32	0.88	111,456	0.00	2.96	0.21	0.65
506	2,608	0.000	105.09	2.01	-	2.58	2.19	6,766	0.01	36.74	2.44	1.14

14.1.11.3 Local Statistical Validation - Swath Plots

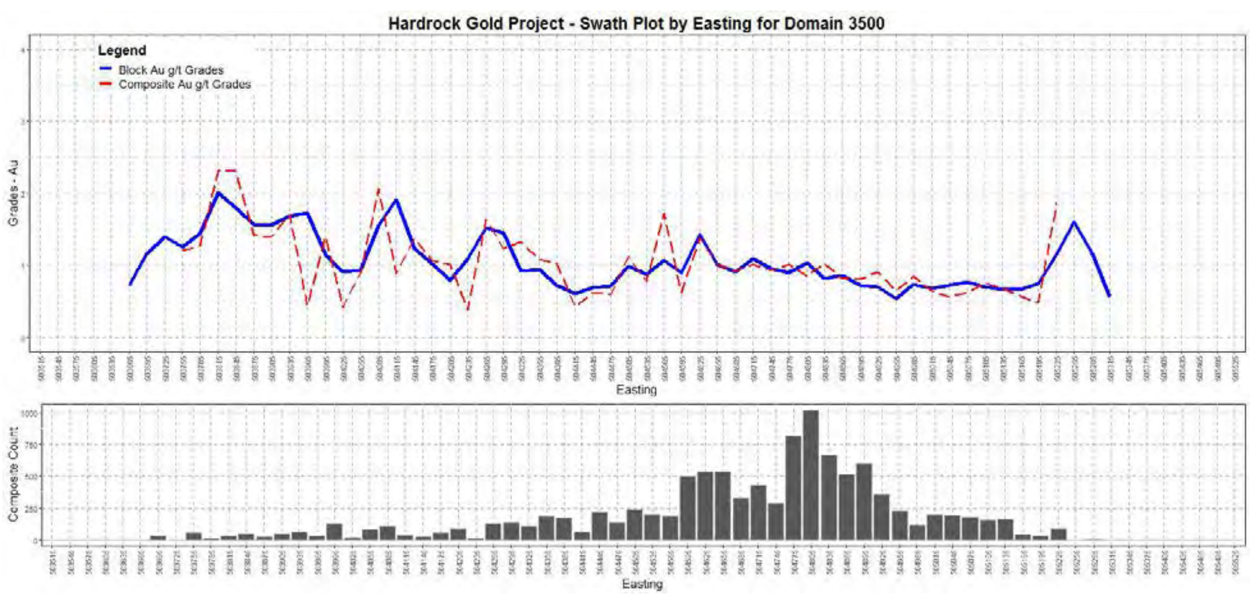
The swath plot method is considered a local validation, which works as a visual mean to compare estimated block grades against composite grades within a 3D moving window. It is used to identify possible bias in the interpolation (*i.e.* a section with significantly high gold content based on a low population of composites).

Swath plots were generated to assess the correlation between composites used in the interpolation of each block versus the total gold content estimated.

Swath plots were produced for all composites of the principal domains at increments of 30 m (Easting) for gold grades and for blocks estimated within Pass 1, 2 or 3 within the 2019 MII pit optimization of the deposit. Peaks and lows in estimated grades should generally follow peaks and lows in composite grades in well informed areas of the block model, whereas less informed areas can occasionally show some discrepancies between the grades.

Figure 14.18 illustrates a swath plot of gold grades for the SP-Zone by Easting (sub-domains were grouped together to produce swath plots) by easting. Peaks and lows in gold content generally match peaks and lows in composite frequency; no bias was found in the resource estimate in this regard.

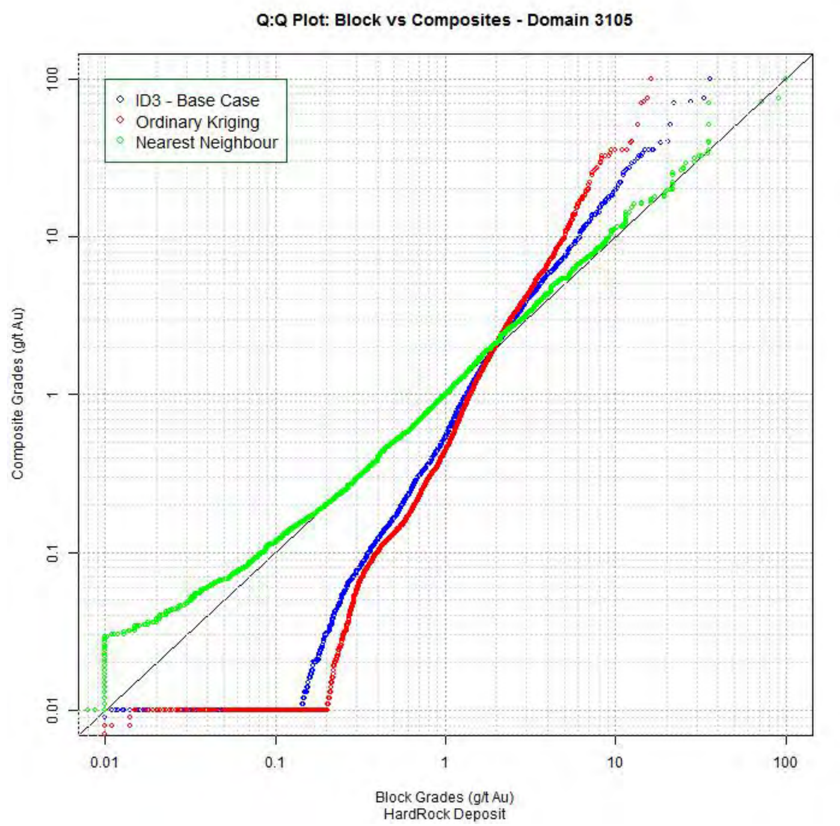
Figure 14.18: Swath Plot of Gold g/t for the SP-Zone by Easting (Pass 1 to 3) within the In-Pit Area



14.1.11.4 Grade Smoothing / Conditional Bias Validations

To determine the impact of the interpolator during grade estimation, GMS produced Q:Q plots comparing the 2 m composites with the Nearest Neighbour ("NN"), Inverse Distance Cubed ("ID3") and Ordinary Kriging Interpolators ("OK"). As expected, the nearest neighbour shows an almost identical distribution to the composite distribution. Higher levels of grade smoothing are observed when using the OK interpolant. Figure 14.19 shows an example Q:Q plot for the F-Zone (3105 rock code, sub-domains were grouped together to produce Q:Q Plots).

Figure 14.19: Q:Q Plot Comparing the Three Grade Interpolators for the F-Zone (3105) within the 2019 MII Pit Optimization



14.1.11.5 Sensitivity Grade Estimations

To assess the impact of the various steps taken in the MRE, GMS ran sensitivity grade estimates to compare with the final grade estimate. These are presented in Subsection 14.1.15.

14.1.11.6 Discussion on Block Model Validation

Globally, the Hardrock block model is a good representation of composite gold grades used in the estimation. Global statistical validations show no significant over/under-estimation of gold grades. Local statistical validations illustrate good local correlation between the interpolated blocks compared to the composite for gold grades, and no overestimation of grades was observed during the validation of estimated grades for the Hardrock gold deposit.

14.1.12 Mineral Resource Classification

14.1.12.1 Mineral Resource Classification Definition

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.

14.1.12.2 Resource Classification for the Hardrock Project

Mineral Resource classification of Measured, Indicated and Inferred category was undertaken for any blocks meeting all the conditions below:

- Measured Mineral Resources are defined as blocks within ~15 m of the 2018 and 2019 RCGC drilling;
- Indicated Mineral Resources are defined as blocks estimated in Pass 1 or 2, where the distance to the closest composite is less than 35 m;
- Inferred Mineral Resources are defined as blocks estimated in Pass 3, and blocks estimated in Pass 1 or 2 where the distance to the closest composite is greater than 35 m;

A grooming step was undertaken on the preliminary classification to ensure that the resource category is coherent for mine planning purposes. GMS is of the opinion that this step was necessary to homogenize (smooth out) the resource volumes in each category. In addition, any blocks located within the external grade shell domains (500, 501 or 506) in the Underground Resources (below the 2019 MII pit optimization) were recategorized to Inferred.

Figure 14.20 to Figure 14.22 show the mineral resource classification, as well as the 2019 Whittle-optimized pit shell delimiting the In-Pit and Underground Mineral Resources.

Figure 14.20: Plan View showing the Categorized Mineral Resources and the 2019 Whittle Optimized Shell (elevation 300 m)

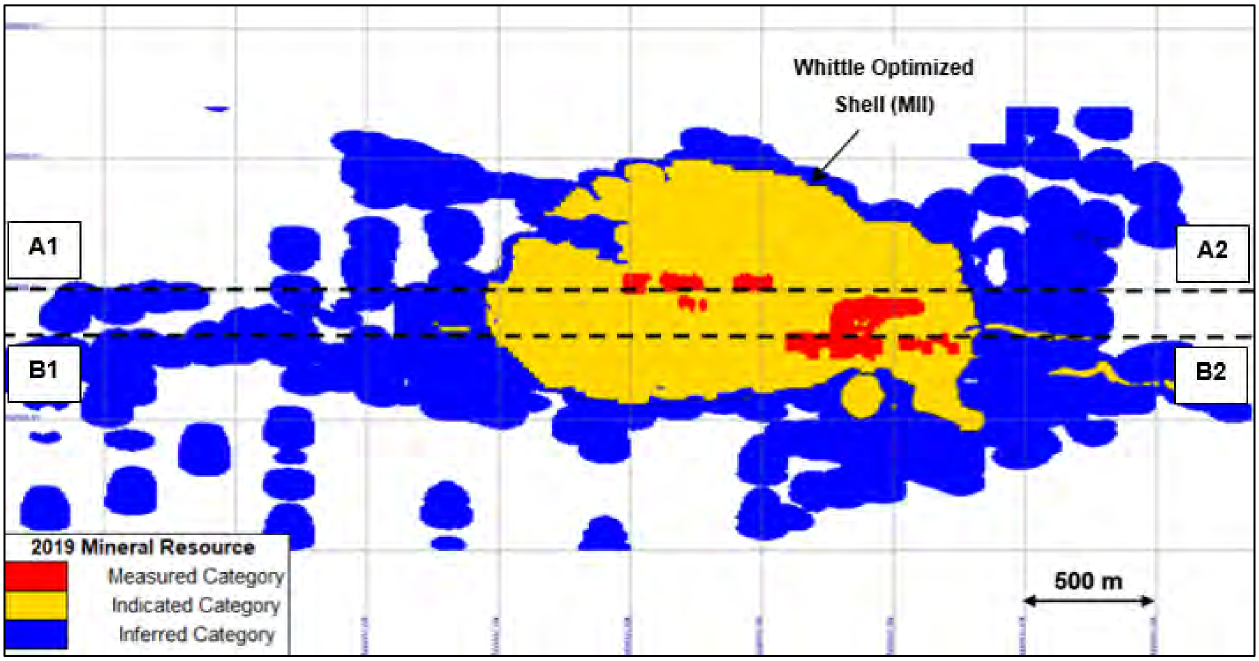


Figure 14.21: Longitudinal View showing the Categorized Mineral Resources and the 2019 Whittle Optimized Pit Shell (longitudinal view 5,503,000N)

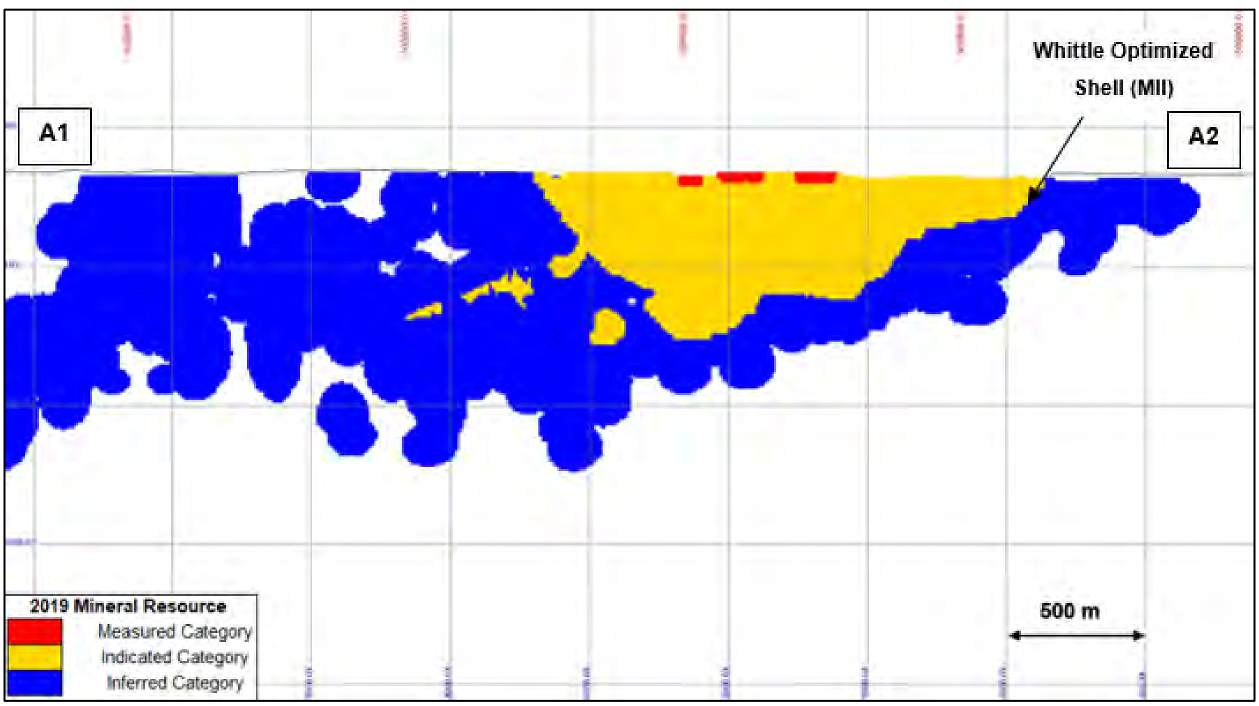
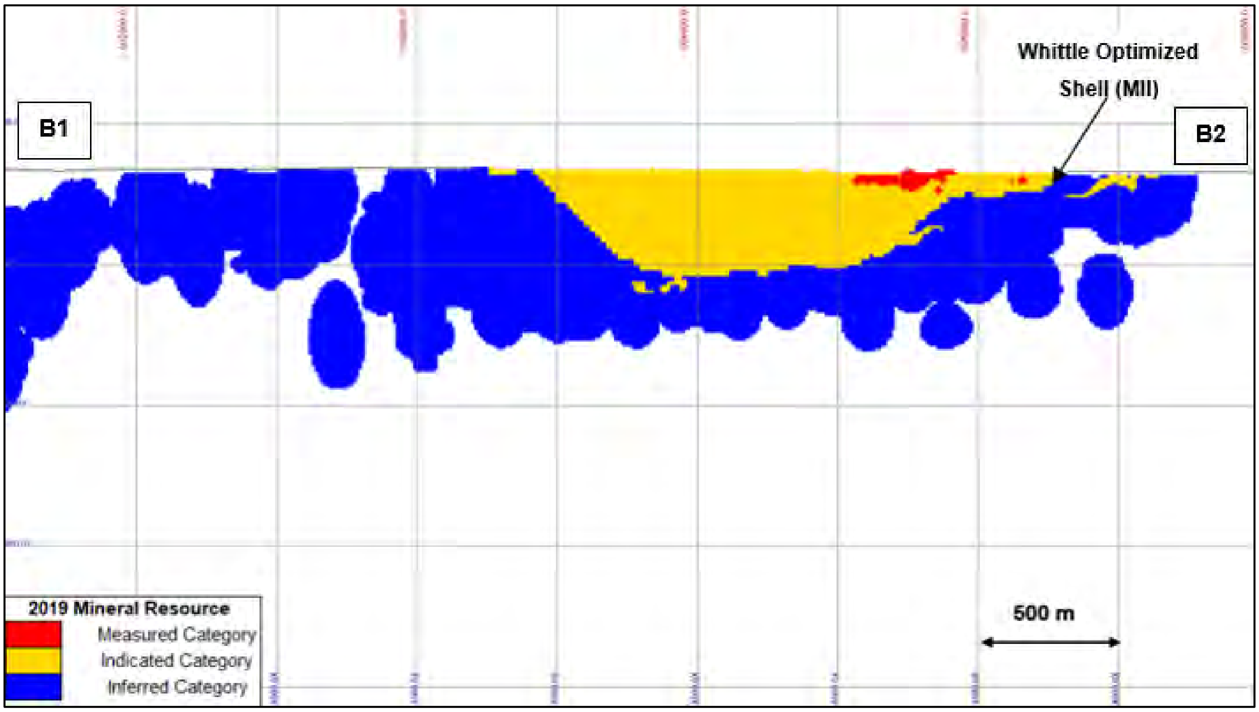


Figure 14.22: Longitudinal View showing the Categorized Mineral Resources and the 2019 Whittle Optimized Shell (longitudinal View 5,502,830N)



14.1.13 <u>In-Pit Constrained Mineral Resources (Inclusive of Mineral Reserves)</u>

To determine In-Pit Mineral Resources at the Hardrock Project, pit optimizations were conducted at various gold prices using the 2019 MRE block model including Measured, Indicated and Inferred category blocks. The final selected Whittle input parameters and cut-off grade parameters used to define the 2019 In-Pit Mineral Resource are defined in Table 14.13.

GMS chose a pit optimization (shell 24) using a revenue factor of 0.78 and a gold price of US$1,250 (effective gold price of US$975). This low revenue factor was chosen due to constraints such as the surrounding lakes and planned project infrastructure, and limited space for waste rock disposal. In addition, the optimization was deepened in the east to ensure that all the Mineral Reserves are encapsulated by the 2019 MRE pit optimization.

Table 14.13: Hardrock Pit Optimization Parameters

Optimization Parameters		Hardrock
Economic Parameters		
Exchange Rate	C$/US$	1.3
Discount Rate	%	5.00%
Gold Price	US$/oz	1,250
Gold Price (Local Currency)	C$/oz	1,625
Transport & Refining Cost	C$/oz	3
Royalty Rate	% NSR	4.40%
Royalty Cost	C$/oz	71.9
Net Gold Value	C$/oz	1,550.15
Recovery & Dilution Factors		
Metallurgical Recovery	%	91.10%
Mining Dilution (Already Diluted)	%	15.90%
Mining Loss	%	1.20%
Processing Costs		
Total Processing Cost Incl. Power	C$/t milled	7.54
Other Ore Based Costs		
Ore Feed Rehandle	C$/t milled	0
IBA Fixed Costs	C$/t milled	0.24
Incremental Ore Haulage	C$/t milled	0
General & Administration Costs	C$/t milled	1.59
Rehabilitation & Closure	C$/t milled	0.38
Sustaining Capital	C$/t milled	0.7
Total Ore Based Cost & Cut-Off Grade		
Total Ore Based Cost	C$/t milled	10.45
Mining		
Total Mining Reference Cost	C$/t mined	1.98
Incremental Bench Cost	C$/10 m bench	0.033

Table 14.14 displays the results of the 2019 MRE for the in situ[1] in-pit portion of the Hardrock Deposit. The Inferred resources represent only a small portion of the global In-Pit resources and are presented in Table 14.19.

Table 14.14: 2019 In-Pit Mineral Resources (Inclusive of Mineral Reserves) at Various Cut-off Grades for the Hardrock Deposit – Measured and Indicated Category

Measured Resource					Indicated Resource				
Zone	**Cut-off g Au/t**	**Tonnes**	**Grade g Au/t**	**Au Oz**	**Zone**	**Cut-off g Au/t**	**Tonnes**	**Grade g Au/t**	**Au Oz**
All Zones	> 0.90	2,429,000	2.37	185,000	All Zones	> 0.90	61,255,000	2.27	4,473,200
	> 0.80	2,673,000	2.23	191,700		> 0.80	67,463,000	2.14	4,642,900
	> 0.70	2,968,000	2.08	198,800		> 0.70	74,166,000	2.01	4,804,400
	> 0.60	3,305,000	1.94	205,800		> 0.60	81,761,000	1.89	4,962,900
	> 0.50	3,727,000	1.78	213,300		> 0.50	91,487,000	1.75	5,134,000
	> 0.40	4,413,000	1.57	223,000		> 0.40	106,223,000	1.57	5,345,100
	> 0.30	**5,690,000**	**1.30**	**237,200**		**> 0.30**	**132,014,000**	**1.33**	**5,631,100**
	> 0.20	7,544,000	1.04	251,900		> 0.20	183,195,000	1.02	6,032,900

Mineral Resource Estimate Notes:

- *The Independent and Qualified Person for the Mineral Resource Estimate, as defined by NI 43-101, is Rejean Sirois, B.Sc., P.Eng. of GMS., and the effective date of the estimate is 04/09/2019;*
- *These Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability;*
- *Mineral Resources are inclusive of Mineral Reserves;*
- *In-Pit results are presented undiluted within a merged surface of the pit optimization shell 24 and the 2019 pit design, using a US$1,250 gold price and a revenue factor 0.78;*
- *The estimate includes 17 gold-bearing zones, and external grade shells to incorporate remaining mineralized material;*
- *In-Pit Resources were compiled at cut-off grades of 0.20, 0.30, 0.40, 0.50, 0.60, 0.70, 0.80 and 0.90 g Au/t; however, the official resource is at a cut-off grade of 0.30 g Au/t;*
- *Density (g/cm^3) data used is on a per zone basis, varying from 2.72 to 3.28 g/cm^3;*
- *A minimum true thickness of 3.0 m was applied during wireframing, using the grade of the adjacent material when assayed, or a value of zero when not assayed;*
- *High grade capping (g Au/t) was undertaken on raw assay data and established on a per zone basis, varying up to 140 g Au/t;*
- *Compositing was done on drill hole intervals within the mineralized zones (2 m lengths);*
- *Resources were estimated using GEOVIA GEMS™ 6.8.2 software from drill hole and surface channel sampling, using a 3 pass ID3 interpolation method in a block model (block size = 10 x 5 x 10 m);*
- *The measured category is defined as blocks within ~15 m of the 2018 and 2019 RCGC drilling;*
- *The indicated category is defined in areas where blocks were interpolated in Pass 1 and 2 (using a minimum of two drill holes) within the 17 principal domains, and external grade shells within the pit optimization;*
- *The inferred category is defined within the areas where blocks were interpolated during Pass 3;*
- *Ounce (troy) = Metric Tonnes x Grade / 31.10348. Calculations used metric units (metres, tonnes and g/t);*
- *The number of metric tonnes was rounded to the nearest thousand and ounces was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in Regulation NI 43-101;*

[1] The term “in situ” is used to represent all the remaining Mineral Resources in place at the time of the 2019 estimate.

- *GMS is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the MRE.*
- *Whittle parameters (all amounts in Canadian dollars): Reference mining cost: $1.98 per tonne, Incremental bench cost ($/10 m bench): $0.033, Milling cost: $7.54/t, NSR Royalty: 4.4%, G&A: $1.59/t, Sustaining capital: $0.70/t, Gold price: $1,625/oz, Milling recovery: 91.1%.*

14.1.14 Underground Mineral Resources

The cut-off grade for 2019 Underground Mineral Resources remains unchanged from the 2016 MRE. The gold selling and processing costs, mining dilution, and processing and mining recoveries were provided by GGM and validated by GMS. The selected underground cut-off grade of 2.0 g Au/t allowed the mineral potential of the deposit to be outlined for the underground mining option, beneath the 2019 MRE pit optimization shell. The estimation of the underground cut-off grade was based on the parameters presented in Table 14.15.

Table 14.15: Input Parameters used for the Underground Cut-off Grade (UCoG) Estimation - Hardrock Deposit

Input Parameter	Value
Exchange Rate	USD 1.00:CAD 1.18
Gold Price (CAD$/oz)	1,715
Gold Selling Costs (CAD$/oz)	4
Royalty (%)	3
Net Gold Price (CAD$/oz)	**1,660**
Mining Costs (CAD$/t)	71.97
Milling Costs (CAD$/t)	8.03
Total Costs	**80**
Processing Recovery (%)	90
Mining Dilution (%)	20
Marginal Cut-off Grade (g Au/t)	**2.01**

The 2019 Underground MRE presented herein uses a rounded value of 2.00 g Au/t for the lower cut-off grade. The Underground MRE is defined by blocks that are located beneath and adjacent to the 2019 MII pit optimization, but not located within.

Table 14.16 displays the results of the MRE for the in situ[1] underground portion of the Hardrock Deposit.

Table 14.16: 2019 Underground Mineral Resources at Various Cut-off Grades for the Hardrock Deposit – Indicated and Inferred Category

Indicated Resource				
Zone	**Cut-off g Au/t**	**Tonnes**	**Grade g Au/t**	**Au Oz**
All Zones	> 4.50	2,397,000	7.26	559,600
	> 4.00	2,988,000	6.66	640,200
	> 3.50	3,840,000	6.01	742,200
	> 3.00	5,086,000	5.33	871,600
	> 2.50	6,820,000	4.67	1,024,200
	> 2.00	**9,792,000**	**3.93**	**1,237,400**
	> 1.50	14,616,000	3.20	1,505,900

Inferred Resource				
Zone	**Cut-off g Au/t**	**Tonnes**	**Grade g Au/t**	**Au Oz**
All Zones	> 4.50	5,039,000	7.69	1,246,600
	> 4.00	6,790,000	6.80	1,485,000
	> 3.50	9,181,000	6.00	1,772,000
	> 3.00	12,345,000	5.30	2,102,000
	> 2.50	17,189,000	4.57	2,527,500
	> 2.00	**24,593,000**	**3.87**	**3,059,100**
	> 1.50	36,322,000	3.18	3,713,400

Mineral Resource Estimate Notes:

- *The Independent and Qualified Person for the Mineral Resource Estimate, as defined by NI 43-101, is Rejean Sirois, B.Sc., P.Eng. of GMS, and the effective date of the estimate is 04/09/2019;*
- *These Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability;*
- *Mineral Resources are inclusive of Mineral Reserves;*
- *UG Mineral Resources are presented undiluted, and are based on blocks located below and adjacent to the 2019 MRE pit optimization shell;*
- *The estimate includes 17 gold-bearing zones, and grade shells to incorporate remaining mineralized material;*
- *UG Resources were compiled at various cut-off grades, however the official resource is at a cut-off grade of 2.00 g Au/t;*
- *Density (g/cm^3) data used is on a per zone basis, varying from 2.72 to 3.28 g/cm^3;*
- *A minimum true thickness of 3.0 m was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed;*
- *High grade capping (g Au/t) was undertaken on raw assay data and established on a per zone basis, varying up to 140 g Au/t;*
- *Compositing was done on drill hole intervals within the mineralized zones (2 m lengths);*
- *Resources were estimated using GEOVIA GEMS™ 6.8.2 software from drill hole and surface channel sampling, using a 3-pass ID3 interpolation method in a block model (block size = 10 x 5 x 10 m);*
- *No measured category exists in the Underground Mineral Resource;*
- *The indicated category is defined in areas where blocks were interpolated in Pass 1 and 2 (using a minimum of two drill holes) in the 17 principal domains;*
- *The inferred category is defined within the areas where blocks were interpolated during Pass 3, and blocks interpolated in Passes 1 and 2 in the external grade shell domains;*
- *Ounce (troy) = Metric Tonnes x Grade / 31.10348. Calculations used metric units (metres, tonnes and g/t);*
- *The number of metric tonnes was rounded to the nearest thousand and ounces was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in Regulation NI 43-101;*
- *GMS is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the MRE.*

14.1.15 Mineral Resource Sensitivity

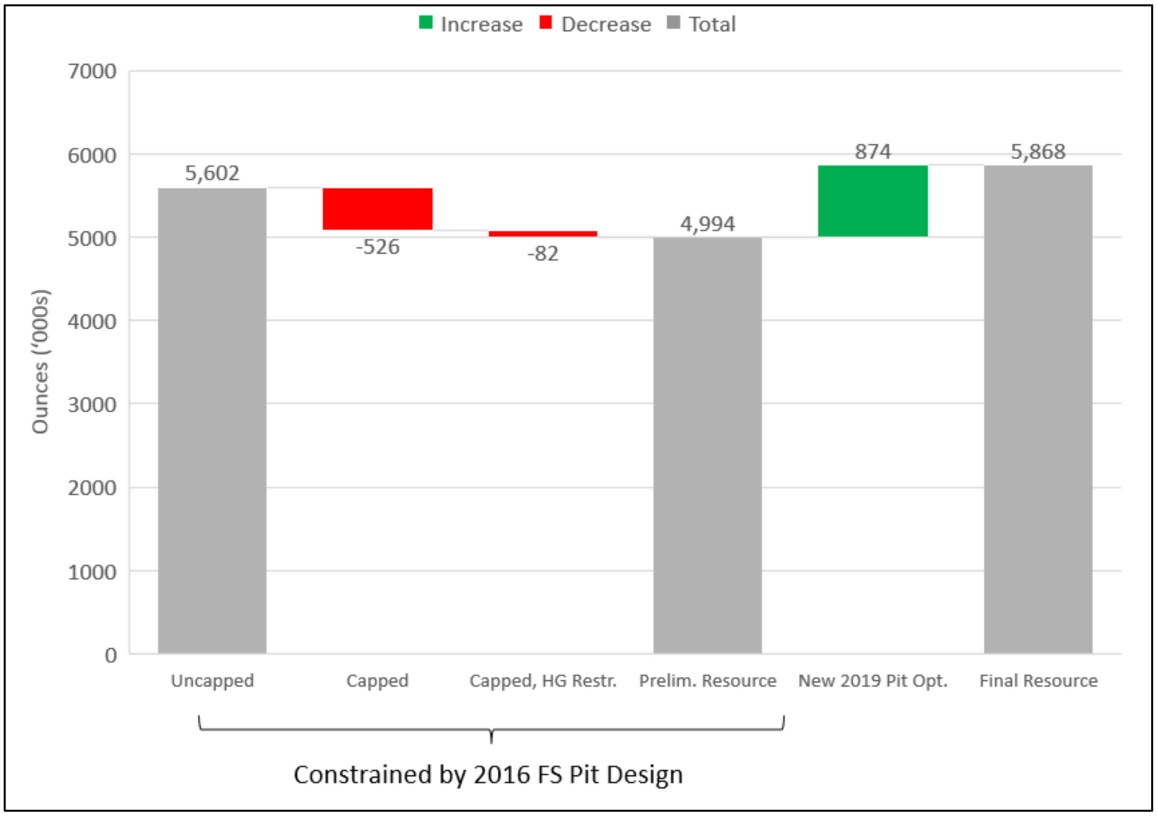
GMS undertook numerous grade estimation runs to assess the sensitivity of the 2019 MRE to various estimation parameters. Estimates were produced using uncapped gold grades, removal of the high-grade restraining, and changing the grade interpolant. In addition, GMS removed the internal sub-domain grade shells from the principal domains to quantify their impact. The results are presented in Table 14.17, and are subdivided into two groups; the seventeen principal wireframes and the external grade shells. Tonnages and grade are reported within the 2019 MRE pit optimization shell for Measured and Indicated resource categories grouped together.

Table 14.17: Summary of Grade Sensitivities Undertaken on the Block Model. Measured and Indicated Categories Combined, Reported within the 2019 MRE Pit Optimization Shell

Sensitivity	Domain	Tonnes (Mt)	Grade (g Au/t)	Ounces (000's)
ID3 Uncapped	17 Principal WF	99.0	1.64	5,231
	External GS	39.0	1.12	1,399
	Total	**138.0**	**1.49**	**6,629**
ID3 Capped, No HG Restraining	17 Principal WF	98.9	1.49	4,729
	External GS	38.9	1.01	1,261
	Total	**137.8**	**1.35**	**5,990**
ID3 Base Case (With HG Restraint)	17 Principal WF	98.9	1.48	4,710
	External GS	38.8	0.93	1,158
	Total	**137.7**	**1.33**	**5,868**
Ordinary Kriging (OK) (HG Restraint)	17 Principal WF	103.0	1.40	4,623
	External GS	38.8	0.93	1,158
	Total	141.7	1.27	5,782
ID3 Base Case, No Internal Grade Shell Sub-domains	17 Principal WF	107.0	1.33	4,595
	External GS	38.8	0.93	1,158
	Total	**145.8**	**1.23**	**5,753**

Capping of gold assays has reduced the overall ounces by 9.6%, with a further 1.8% removed by the HG restraining strategy. Ordinary Kriging (OK) results in slightly lower ounces than the Inverse Distance cubed (ID^3) interpolator (1.4% lower ounces), with lower grade and higher tonnage. This is expected as ordinary kriging generally produces a more smoothed grade estimate than ID^3. The removal of the internal grade shells also slightly reduces the total ounces by 2%, with a 7.5% reduction in grade and a 5.9% increase in tonnage. Figure 14.23 presents the sensitivities in a waterfall chart.

Figure 14.23: Waterfall Chart of 2019 In-Pit Measured + Indicated Ounces showing Sensitivity to Capping and High-grade Restraining, and Deepening of Pit Optimization Shell



GMS also conducted sensitivity grade estimates with a more conservative capping approach, where all composites pertaining to a domain with a Coefficient of Variation ("CoV") greater than 2.0 were further capped to reduce the CoV below 2.0. This is essentially a second capping applied to the composites. An additional 55 composites were capped in the high-grade sub-domains (> 0.6 g/t grade shell) for the F-Zone, Central-Zone, F2-Zone, and the SP-Zone. The > 0.6 g/t external grade shell (rock code 506) was also capped further. The results are shown in Table 14.18.

Table 14.18: Sensitivity Grade Estimate using more Conservative Capping Approach. Measured and Indicated Categories Combined, Reported within the 2019 MRE Pit Optimization Shell

Sensitivity	Domain	Tonnes (Mt)	Grade (g Au/t	Ounces (000's)
ID^3 Base Case (With HG Restraint)	17 Principal WF	98.9	1.48	4,710
	External GS	38.8	0.93	1,158
	Total	137.7	1.33	5,868
ID^3 CoV < 2 Case	17 Principal WF	98.9	1.47	4,664
	External GS	38.8	0.92	1,148
	Total	**137.7**	**1.31**	**5,813**

The more conservative capping approach resulted in a 1% reduction in overall contained ounces.

Lastly, GMS assessed the impact of retaining the composite residuals on the 2019 MRE. The results indicated that by removing composite residuals < 0.5 m in length increases grade and tonnes by less than 1% and increases overall ounces by 1.3%. GMS considers this effect as negligible and decided to retain the composites residuals during estimation.

14.1.16 Summary of the 2019 Hardrock Mineral Resource

The 2019 MRE update presented in this section includes a compilation of:

- An In-Pit Resource Estimate, within the 2019 Whittle-optimized shell (Table 14.14);
- An underground resource estimate, outside the 2019 Whittle-optimized pit-shell (Table 14.16).

Table 14.19 presents the combined resources by resource category for the Hardrock Deposit.

Table 14.19: Summary of 2019 Mineral Resource Estimate (Inclusive of Open-Pit Mineral Reserves) for the Hardrock Project

	2019 Inclusive Mineral Resources					
	In-Pit > 0.3 g Au/t			Underground > 2.0 g Au/t		
Category	**Tonnage (Mt)**	**Grade (g/t)**	**Ounces (000's)**	**Tonnage (Mt)**	**Grade (g/t)**	**Ounces (000's)**
Measured	5.7	1.30	237	-	-	-
Indicated	132	1.33	5,631	9.8	3.93	1,237
M+I	**137.7**	**1.33**	**5,868**	**9.8**	**3.93**	**1,237**
Inferred	0.9	1.19	36	24.6	3.87	3,059

Mineral Resource Estimate Notes:

- *The Independent and Qualified Person for the Mineral Resource estimate, as defined by NI 43-101, is Rejean Sirois, B.Sc., P.Eng. of GMS., and the effective date of the estimate is 04/09/2019;*
- *These Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability;*
- *Mineral Resources are inclusive of Mineral Reserves;*
- *In-Pit results are presented undiluted within a merged surface of the pit optimization shell 24 and the 2019 pit design, using a US$1,250 gold price and a revenue factor 0.78;*
- *UG Mineral Resources are presented undiluted, and are defined as blocks below and adjacent to the 2019 pit optimization;*
- *The estimate includes 17 gold-bearing zones, and grade shells to incorporate remaining mineralized material;*
- *In-Pit Resources were compiled at cut-off grades of 0.20, 0.30, 0.40, 0.50, 0.60, 0.70, 0.80 and 0.90 g Au/t; however, the official resource is at a cut-off grade of 0.30 g Au/t;*
- *Underground Resources were compiled at cut-off grades of 1.50, 2.00, 2.50, 3.00, 3.50, 4.00 and 4.50 g Au/t; however, the official resource is at a cut-off grade of 2.00 g Au/t;*
- *Density (g/cm^3) data used is on a per zone basis, varying from 2.72 to 3.28 g/cm^3;*
- *A minimum true thickness of 3.0 m was applied during wireframing, using the grade of the adjacent material when assayed, or a value of zero when not assayed;*
- *High grade capping (g Au/t) was undertaken on raw assay data and established on a per zone basis, varying up to 140 g Au/t;*
- *Compositing was done on drill hole intervals within the mineralized zones (2 m lengths);*
- *Resources were estimated using GEOVIA GEMS™ 6.8.2 software from drill hole and surface channel sampling, using a 3-pass ID3 interpolation method in a block model (block size = 10 x 5 x 10 m);*
- *The measured category is defined as blocks within ~15 m of the 2018 and 2019 RCGC drilling;*
- *The indicated category is defined in areas where blocks were interpolated in Passes 1 and 2 (using a minimum of two drill holes) within the 17 principal domains and external grade shells within the resource pit optimization;*
- *The inferred category is defined within the areas where blocks were interpolated during pass 3, and blocks within the underground resource interpolated in Passes 1 and 2 in the external grade shell domains;*
- *Ounce (troy) = Metric Tonnes x Grade / 31.10348. Calculations used metric units (metres, tonnes and g/t);*
- *The number of metric tonnes was rounded to the nearest thousand and ounces was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in Regulation NI 43-101;*
- *GMS is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the MRE.*
- *Whittle parameters (all amounts in Canadian dollars): Reference mining cost: $1.98 per tonne, Incremental bench cost ($/10 m bench): $0.033, Milling cost: $7.54/t, NSR Royalty: 4.4%, G&A: $1.59/t, Sustaining capital: $0.70/t, Gold price: $1,625/oz, Milling recovery: 91.1%.*

14.1.17 Comparison with the Previous Estimate

A comparison of the 2019 MRE (In-Pit and Underground) to the 2016 MRE is presented in Table 14.20.

The overall 2019 Combined In-Pit and Underground Measured and Indicated Resources of 7,106,000 ounces of gold for the Hardrock Project represents an 11% increase in total ounces versus the 2016 estimate (at their respective cut-off).

The 2019 In-Pit Measured and Indicated Resources represent a 21% increase in grade, 26% increase in ounces and 4% increase in tonnes versus In-Pit Measured and Indicated Resources reported in the 2016 FS.

The principal factors contributing to the increase in the current MRE are as follows:

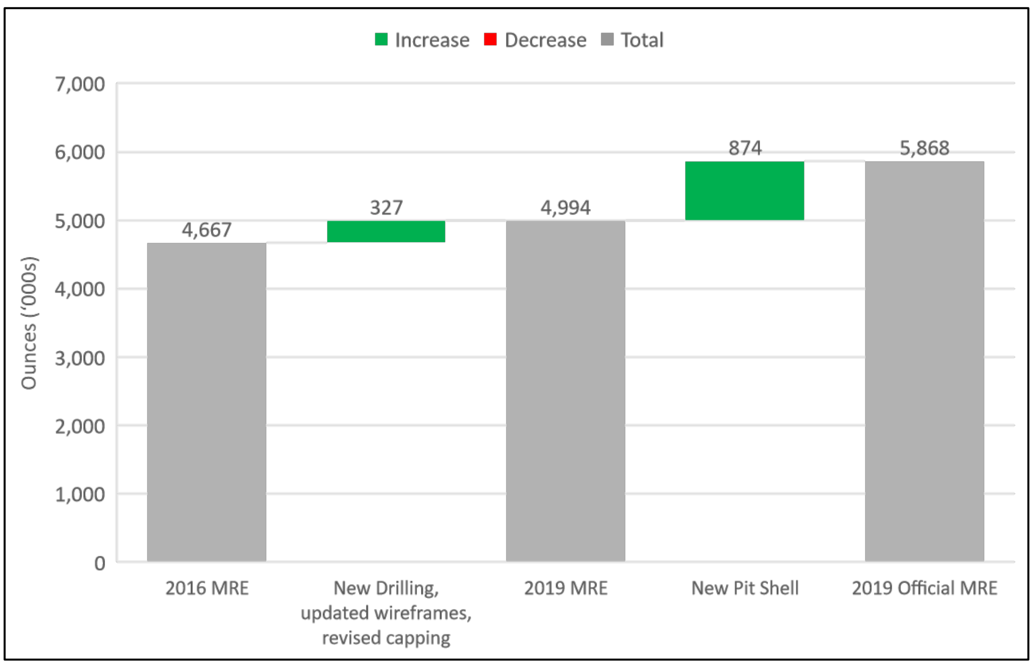
1. The 2019 MRE is constrained by a deeper pit optimization, which incorporates significantly more resources compared to the 2016 MRE;
2. The reduction of internal dilution within the seventeen principal domains has resulted in a 24% increase in average grade of assays within these domains (shown in Table 14.2). This has resulted in a higher grade for the 2019 Mineral Resource;
3. Grade capping was revisited in 2019 (due to the refined wireframes), and new capping thresholds were chosen. They are usually less restrictive as the capping chosen in 2016;
4. RC grade-control drilling and validation diamond drilling conducted in 2018 and 2019 confirmed grade continuity, and generally intersected higher-grades than expected in the 2016 block model. Despite the de-risking drilling programs undertaken in 2018 and 2019 cover a small portion of the entire resource, both programs imply that higher grade than 2016 block model should be encountered during the extraction of the resources.

The same gold price was used during the pit optimization process in the 2019 and 2016 estimates, therefore has not contributed to the upgraded profile of the new mineral resource. The waterfall diagram in Figure 14.24 summarizes changes in the 2019 MRE update. It is important to note that there is only a 7% increase in Measured and Indicated ounces (327 KOz) when reporting the 2019 MRE block model in the 2016 pit optimization shell. The majority of the increase in ounces between the 2016 and 2019 MRE's is a result of the deepening of the pit optimization shell (874 KOz).

Table 14.20: Summary of Changes in 2019 Mineral Resource Estimate (Measured and Indicated) vs. 2016 Estimate

		In-Pit Restrained (> 0.3 g Au/t)			Underground Constrained (> 2.0 g Au/t)		
		Tonnes (Mt)	Grade (g Au/t)	Ounces (000's)	Tonnes (Mt)	Grade (g Au/t)	Ounces (000's)
Measured Indicated Resources (M+I)	2016 MRE	131.9	1.10	4,667	13.7	3.91	1,720
	2019 MRE	137.7	1.33	5,868	9.8	3.93	1,237
	2019 vs 2016 % Change	**4%**	**21%**	**26%**	**-28%**	**1%**	**-28%**
Inferred Resources	2016 MRE	0.2	0.87	5	21.5	3.57	2,470
	2019 MRE	0.9	1.19	36	24.6	3.87	3,059
	2019 vs 2016 % Change	**452%**	**36%**	**648%**	**14%**	**8%**	**24%**

Figure 14.24: In-Pit Constrained Measured and Indicated Mineral Resources Waterfall Chart (0.3 g/t Lower Cut-off) of Ounces



The total underground Mineral Resource remains relatively unchanged, apart from some reclassifications as described below:

- 3.9 Mt reduction in Indicated Mineral Resources;
  - These blocks were previously estimated within a poorly constrained lithological domain and have been reclassified into the Inferred Category;
- 3.1 Mt increase in Inferred Mineral Resources;
  - All underground blocks within the external grade shell domains (rock codes 500, 501 and 506) outside of the principal wireframes are reclassified as Inferred;

14.1.18 2019 In-Pit Constrained Mineral Resources (Exclusive of Mineral Reserves)

Premier Gold and Centerra Gold publish their respective Mineral Resources exclusive of Mineral Reserves (i.e. Mineral Resources are in addition to Mineral Reserves). Table 14.21 presents the 2019 In-Pit Mineral Resources exclusive of Mineral Reserves for Indicated and Inferred categories. When reporting Mineral Resources as exclusive of Mineral Reserves, no remaining Measured resources exist at the Hardrock Project.

Table 14.21: 2019 In-Pit Mineral Resources (Exclusive of Mineral Reserves) at Various Cut-off Grades for the Hardrock Deposit – Indicated and Inferred Categories

Indicated Resource				
Zone	**Cut-off g Au/t**	**Tonnes**	**Grade g Au/t**	**Au Oz**
All Zones	> 0.90	2,671,000	2.07	178,100
	> 0.80	2,935,000	1.96	185,400
	> 0.70	3,219,000	1.86	192,200
	> 0.60	3,559,000	1.74	199,300
	> 0.50	4,010,000	1.61	207,200
	> 0.40	4,767,000	1.42	218,100
	> 0.30	**5,972,000**	**1.21**	**231,400**
	> 0.20	8,297,000	0.94	249,700

Inferred Resource				
Zone	**Cut-off g Au/t**	**Tonnes**	**Grade g Au/t**	**Au Oz**
All Zones	> 0.90	210,000	1.58	10,700
	> 0.80	230,000	1.52	11,200
	> 0.70	240,000	1.48	11,500
	> 0.60	253,000	1.44	11,700
	> 0.50	262,000	1.41	11,900
	> 0.40	307,000	1.27	12,500
	> 0.30	**356,000**	**1.14**	**13,100**
	> 0.20	438,000	0.97	13,700

Mineral Resource Estimate Notes:

- *The Independent and Qualified Person for the Mineral Resource Estimate, as defined by NI 43-101, is Rejean Sirois, B.Sc., P.Eng. of GMS., and the effective date of the estimate is 04/09/2019;*
- *These Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability;*
- *Mineral Resources are exclusive of Mineral Reserves;*
- *In-Pit results are presented undiluted within a merged surface of the pit optimization shell 24 and the 2019 pit design, using a US$1,250 gold price and a revenue factor 0.78;*
- *The estimate includes 17 gold-bearing zones, and external grade shells to incorporate remaining mineralized material;*
- *In-Pit Resources were compiled at cut-off grades of 0.20, 0.30, 0.40, 0.50, 0.60, 0.70, 0.80 and 0.90 g Au/t; however, the official resource is at a cut-off grade of 0.30 g Au/t;*
- *Density (g/cm^3) data used is on a per zone basis, varying from 2.72 to 3.28 g/cm^3;*
- *A minimum true thickness of 3.0 m was applied during wireframing, using the grade of the adjacent material when assayed, or a value of zero when not assayed;*
- *High grade capping (g Au/t) was undertaken on raw assay data and established on a per zone basis, varying up to 140 g Au/t;*
- *Compositing was done on drill hole intervals within the mineralized zones (2 m lengths);*
- *Resources were estimated using GEOVIA GEMS™ 6.8.2 software from drill hole and surface channel sampling, using a 3-pass ID3 interpolation method in a block model (block size = 10 x 5 x 10 m);*
- *The inferred category is only defined within the areas where blocks were interpolated during Pass 3;*
- *The indicated category is only defined in areas where blocks were interpolated in Passes 1 and 2 (using a minimum of two drill holes);*
- *Ounce (troy) = Metric Tonnes x Grade / 31.10348. Calculations used metric units (metres, tonnes and g/t);*
- *The number of metric tonnes was rounded to the nearest thousand and ounces was rounded to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in Regulation NI 43-101;*
- *GMS is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the MRE.*
- *Whittle parameters (all amounts in Canadian dollars): Reference mining cost: $1.98/t, Incremental bench cost ($/10 m bench): $0.033, Milling cost: $7.54/t, NSR Royalty: 4.4%, G&A: $1.59/t, Sustaining capital: $0.70/t, Gold price: $1,625/oz, Milling recovery: 91.1%.*

14.1.19 Underground Mineral Resources (Exclusive of Mineral Reserves)

Since no Mineral Reserves are estimated for the underground portion of the Hardrock Deposit, the 2019 Underground Mineral Resources exclusive of Mineral Reserves are identical to Table 14.16.

14.2 Brookbank, Kailey and Key Lake Deposits Mineral Resource Estimates

The Mineral Resource estimate for the Brookbank, Kailey and Key Lake Deposits were prepared by James Purchase, P. Geo and Director of Geology and Resources at GMS using Leapfrog EDGE™ version 5.1. The MRE is based on a drilling database for the three deposits provided to GMS on June 26th, 2020. The effective date of the MRE for the three deposits is September 3rd, 2020.

The previous MRE for the Brookbank, Kailey and Key Lake Deposits was released on January 30th, 2013 by MICON International (effective date of the MRE was December 14th, 2012). Since the last MRE, there has been a significant advancement in the geological understating of the three deposits, and considerable work has been undertaken by GGM on data verification and increasing the confidence in the drilling database.

All data relating to this MRE is expressed in NAD83, UTM Zone 16N.

No Mineral Reserves have been calculated at the Brookbank, Kailey and Key Lake Deposits, so all Mineral Resources reported in this section can be considered as inclusive of Mineral Reserves.

14.2.1 Brookbank Deposit

The Brookbank Deposit is located around 77 km west of the Hardrock Deposit, and is a greenfield deposit that has not been subject to any mine development or production in the past. In addition to the Brookbank Main deposit, there are two smaller deposit named “Foxear” and “Cherbourg” which form part of the same mineralized system along strike.

14.2.1.1 Drill Hole Database

The Brookbank Deposit (main zone) has been tested by diamond drilling over a strike length of 1,150 m and down to a vertical depth of 1,100 m. The Brookbank drill hole database includes 688 drill holes totalling 187,901 m, of which 592 drill holes and 152,750 m were drilled within the Brookbank, Foxear and Cherbourg Deposits.

The drill holes are on a grid varying from 25 m (close to surface) to 200 m at depth. The main components of the database are the collar, survey, assay and lithology tables which were validated as described in Subsection 12.2.

14.2.1.2 Topography

Topography was supplied by GGM in the form of a wireframe with a spatial resolution of 10 m. The topography matched well with the drill hole collars, with no major discrepancies identified.

14.2.1.3 Geological Modelling Approach

The Brookbank gold deposits comprise the Brookbank Main, the Cherbourg and the Foxear Deposits (Figure 14.25) that occur at three different localities within the 6.5 km long Brookbank shear zone. The deposits are located at and / or near the contacts between mafic volcanics and meta-sediments. The deposits have both structural and lithological controls; however, a second order, sub-vertical plunge and continuity of mineralization is also apparent within the shear zone. Gold mineralization occurs within multiple quartz-carbonate stringers, veinlets and/or stockworks that give rise to broad zones of mineralization varying in width from 1 to 2 m at a depth of about 700 m to up to 20 to 50 m wide at or close to surface.

The Brookbank main deposit has been modelled as two discrete, continuous zones names the footwall “FW” and hanging wall “HW” veins (Figure 14.26). A minimum true width of 1.5 m and a 2 g Au/t lower cut-off was applied during modelling. In addition, the wider altered shear zone has been modelled, alongside the footwall conglomerate and hanging wall metavolcanics lithologies. No overburden surface was modelled as the overburden is generally thin across the deposit (< 2 m thickness).

The Norander-era drilling (1940’s, N-Series holes) were used in the interpretation but excluded from the estimation as their collar locations could not be validated with sufficient confidence. In addition, numerous wedge holes have been drilled at Brookbank (often for metallurgical sampling), and those with no assays were excluded from the estimation.

Figure 14.25: Modelled Solids of the Brookbank Deposits, looking NW

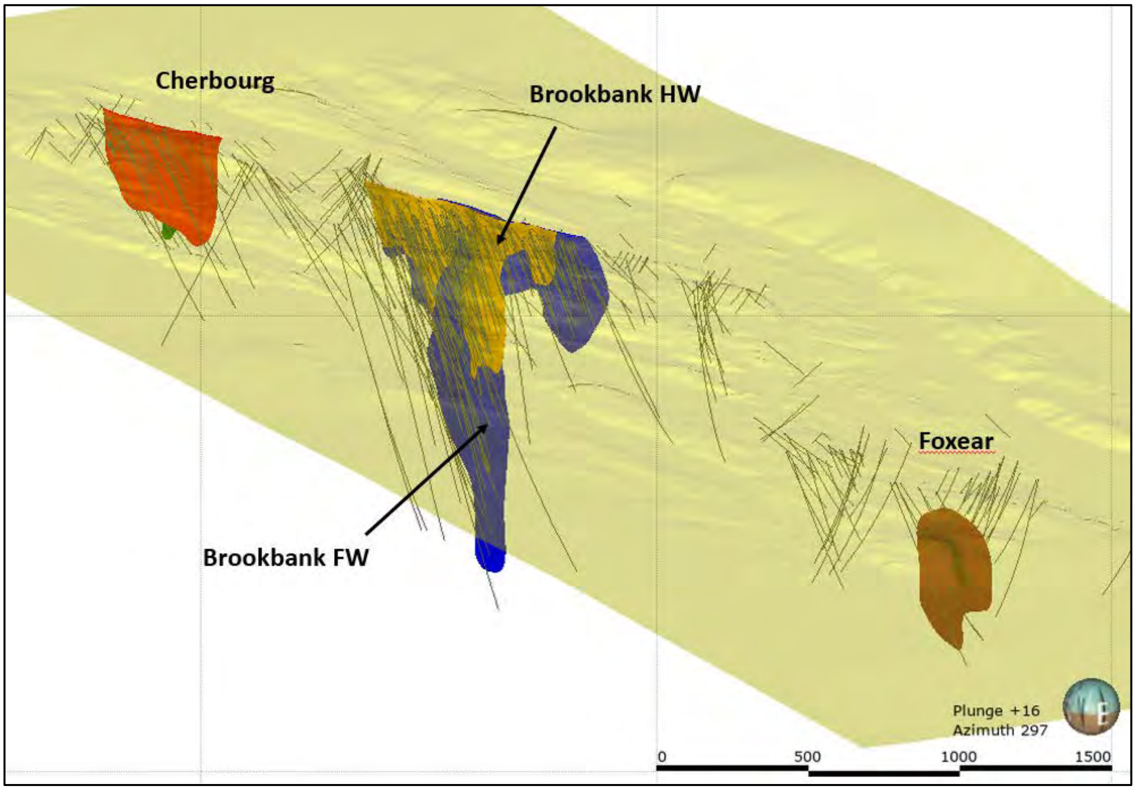
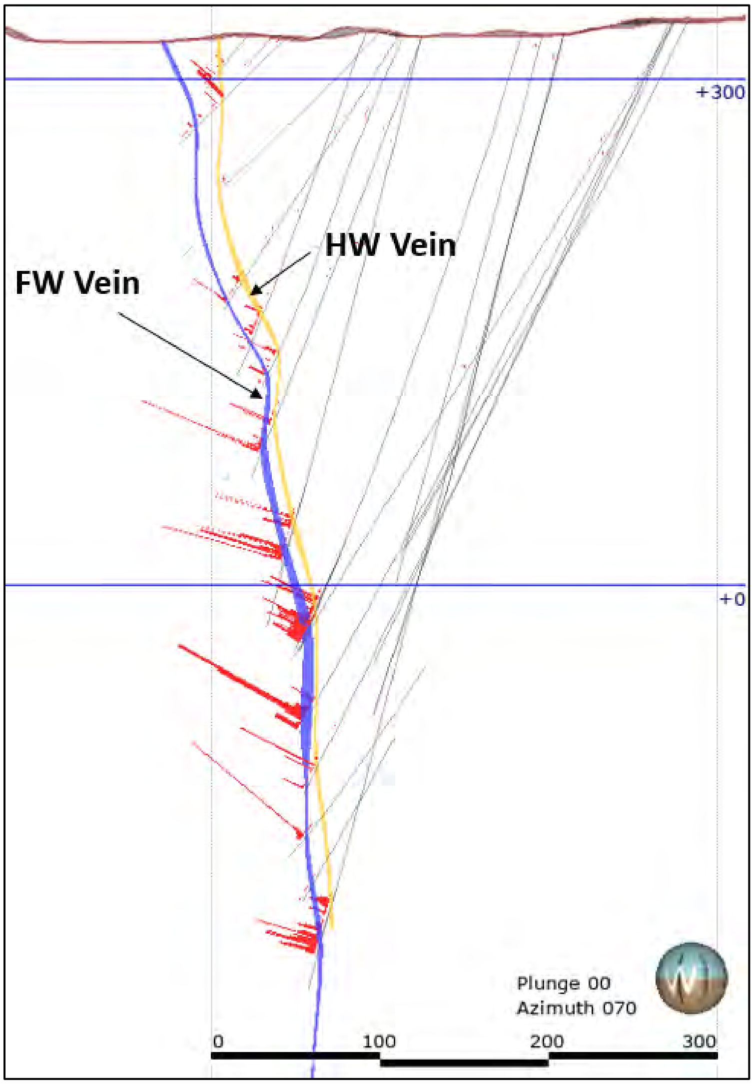


Figure 14.26: Typical section (looking east) showing the Footwall (blue) and Hanging Wall (orange) Veins. Bar Charts on Drilling Traces show Gold Grades



14.2.1.4 Assay Capping and Compositing

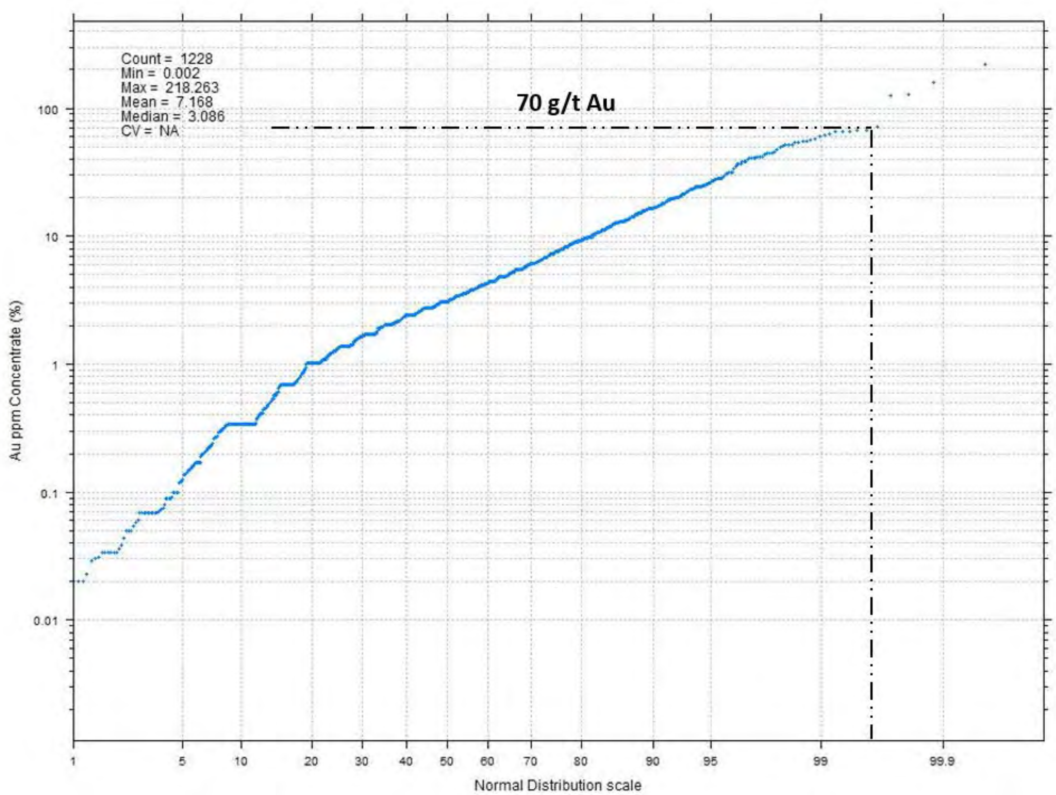
Grade capping levels were determined using probability plots of the various domains, and applied to the assay intervals. Decile analysis was also undertaken as a check to ensure that no more than 10% of the metal is contained in the last percentile. Length-weighted assay statistics and capping levels are shown in Table 14.22. An example of the probability plot for the FW vein domain is shown in Figure 14.27.

Table 14.22: Length-weighted Assays Statistics showing Grade Capping Levels and Metal Loss Factors

Domain	Number of Assays	Max (g Au/t)	Uncut Mean (g Au/t)	High Grade Capping	Cut Mean (g Au/t)	# Samples Cut	% Samples Capped	% Loss Metal Factor
FW	1,261	218.3	6.86	70	6.58	5	0.40%	4.0%
HW	563	141.7	4.04	40	3.77	5	0.89%	6.5%
Alteration	4,426	67.9	0.36	10	0.34	9	0.20%	4.7%
Conglomerate	2,690	42.8	0.16	10	0.14	9	0.33%	6.9%
Metavolcanic	15,949	54.5	0.11	20	0.10	8	0.05%	4.3%
Cherbourg Main	122	32.6	3.54	20	3.14	3	2.46%	11.2%
Cherbourg FW	7	19.4	7.37	None	7.37	0	0.00%	0.0%
Foxear	104	13.6	2.65	None	2.65	0	0.00%	0.0%
Total	**25,122**							**4.7%**

*Notes: % Loss metal factors calculated from length * grade and does not consider the spatial location of the outliers*

Figure 14.27: Example of a Probability Plot for the FW Vein



Core sampling was undertaken at 1.0 m intervals and were broken down on visual lithological and alteration contacts. Considering this, GMS has applied a 1 m compositing run-length split by domain, with any residuals less than 0.5 m added to the last composite. A minimum coverage of 50% was required to create a composite. Missing intervals were replaced with a grade of 0.001 g Au/t.

Length-weighted composite statistics of drilling used in the estimation is shown in Table 14.23.

Table 14.23: Length-weighted Composite Statistics of Capped Gold Grades by Domain

Domain	Number of 1 m Comps	Min (g Au/t)	Max (g Au/t)	Mean (g Au/t)	Median (g Au/t)	St Dev.	Variance	Coeff. Of Variation
FW	1,098	0.001	68.71	6.79	3.55	9.61	92.40	1.42
HW	538	0.001	40.00	3.83	2.06	5.32	28.28	1.39
Alteration	5,133	0.001	9.60	0.30	0.07	0.67	0.45	2.22
Conglomerate	17,427	0.001	9.30	0.02	0.00	0.24	0.06	9.69
Metavolcanic	133,187	0.001	20.00	0.01	0.00	0.19	0.04	14.73
Cherbourg Main	102	0.001	20.00	2.81	1.06	4.32	18.66	1.54
Cherbourg FW	6	2.66	19.37	7.37	5.61	6.32	39.94	0.86
Foxear	97	0.001	13.63	2.43	2.19	2.16	4.65	0.89

14.2.1.5 Variography

Experimental variograms were constructed for the two principal veins (FW and HW) and for the alteration zones using the capped gold composite intervals. Nugget sills were estimated from downhole variograms at a 1 m lag spacing. Pair-wise variograms were interpreted as they showed the most coherent structure.

GMS was able to interpret variograms for the FW and HW veins, and the alteration zone. The conglomerate and metavolcanics domains did not yield reliable variograms, and there was insufficient data to model variograms for the Cherbourg and Foxear Deposits. Variogram parameters are shown in Table 14.24.

Table 14.24: Variogram Parameters for the Brookbank Deposit

Zone	Axis	Nugget (C0)	Sill 1 (C1)	Range 1 (R1)	Sill 2 (C2)	Range 2 (R2)	Dip	Dip Azimuth	Pitch
FW	Major	0.2	0.42	13	0.49	70	82	160	98
	Semi-major			13		60			
	Minor			4		8			
HW	Major	0.3	0.42	7	0.4	40	82	160	98
	Semi-major			7		40			
	Minor			4		8			
Alteration	Major	0.25	0.63	25	0.2	100	82	160	109
	Semi-major			30		90			
	Minor			4		8			

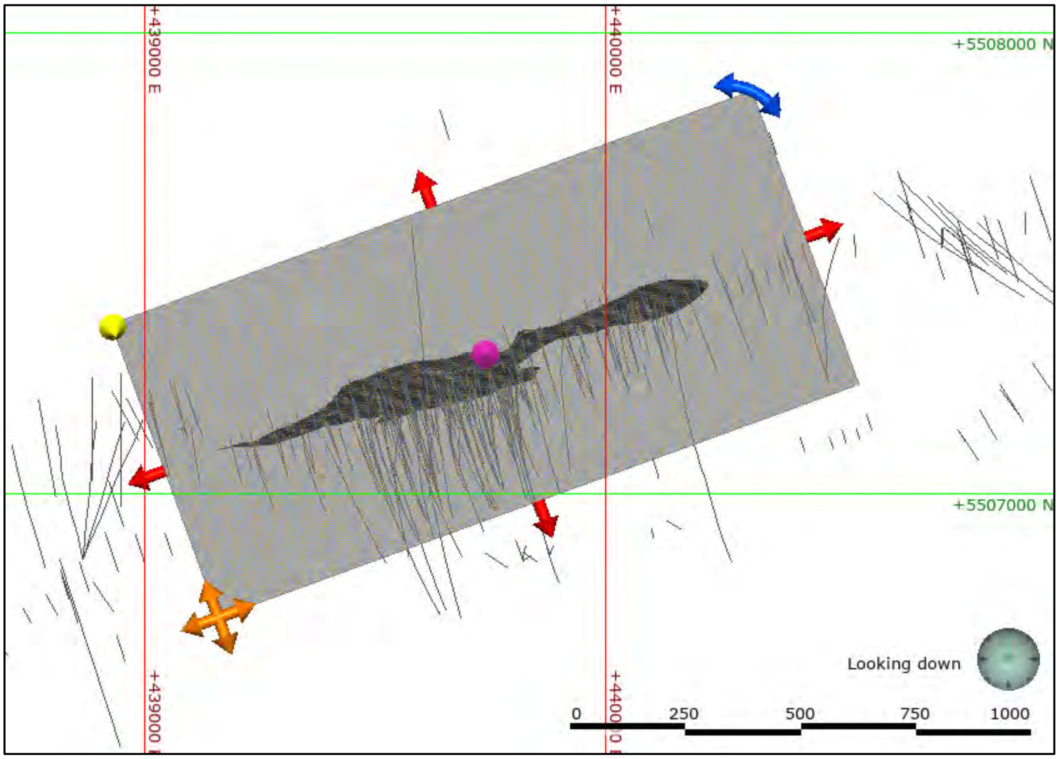
14.2.1.6 Block Modelling

The block model definition is presented in Table 14.25. The upper limit was defined by the surface topography. The parent block size was based primarily on drill hole spacing, envisaged selective mining unit (“SMU”) and geometry of the deposit. The block model was sub-blocked using the domain wireframes. A volume check of the block model versus the mineralization envelope revealed a good representation of the volume of the solid. Figure 14.28 shows a plan view of the Brookbank block model location.

Table 14.25: Brookbank Main Deposit Block Model Attributes

Item	X	Y	Z
Origin Coordinates	439,160.00	5,506,730.00	450
Block Extents (m)	1,480	678	1,600
Number of Parent	148	226	160
Parent Block Size (m)	10	3	10
Sub-Block Size (m)	2.5	1	2.5
Rotation		20 Degrees Anti-clockwise	

Figure 14.28: Plan View of Brookbank Block Model, Wireframes and Drill Traces



14.2.1.7 Bulk Density Data

Bulk Density data was supplied by GGM in the form of a excel spreadsheet containing bulk density readings by lithology and deposit. In total, 196 measurements were taken using the Archimedes method of measuring the weight of the core sampling in water and in air. Table 14.26 presents the bulk density data available for the Brookbank Deposit.

Table 14.26: Statistical Summary Of Bulk Density Data for the Brookbank Deposit

Deposit	Lithology	No. of Meas.	Mean Density (g/cc)	St. Dev Density (g/cc)
Brookbank	Basalt	18	2.83	0.086
	Conglomerate	6	2.83	0.047
	Gabbro	3	2.79	0.067
	Greywacke	3	2.73	0.070
	Tuff	3	2.78	0.038

As the key lithologies at Brookbank are Basalt and Conglomerate, GMS applied a consistent bulk density of 2.83 g/cc for all rock types. No significant overburden is present at Brookbank, and was not incorporated into the block model at this time.

14.2.1.8 Search Ellipsoids and High-grade Restraining

Due to the undulating nature of the veins, GMS decided to use dynamic anisotropy to locally adjust the search ellipse orientations according to the local dip and dip direction of the vein wireframe. A surface was built using the mid-points of the vein, and was used as an input to determine the rotation angles of the search ellipse.

The search ellipse configurations were defined using variography and drill spacing as a guide combined with the geometry of the deposit. A three-pass estimation procedure was used for the interpolation. For all passes, the maximum number of samples per drill hole was set to control the number of drill holes in the interpolation.

For Pass 1, a minimum number of samples of 7, a maximum number of samples of 16, and a maximum samples per drill hole of 3 was applied. This ensured that a minimum of 3 drill holes were required to estimate blocks in the first pass.

For Pass 2, a minimum number of samples of 4, a maximum number of samples of 16, and a maximum samples per drill hole of 3 was applied. This ensured that a minimum of 2 drill holes were required to estimate blocks in the second pass.

For Pass 3, a minimum number of samples of 2, a maximum number of samples of 16, and a maximum samples per drill hole of 3 was applied. This ensured that a minimum of 1 drill holes were required to estimate blocks in the third pass.

In regard to sequencing, Pass 1 took precedence over Pass 2, and Pass 2 took precedence over Pass 3.

In order to control the influence of isolated high-grade composites during grade estimation, GMS applied high-grade restraining (also know as “high-grade restraint” or “transition”). This method involves applying a second, smaller internal search ellipse to restrict the influence of high-grade composites above a user-defined value (a “threshold”). Thresholds were chosen from probability plots of the gold composites for each domain.

The search parameters adopted for grade interpolation are summarized in Table 14.27.

Table 14.27: Summary of Search Parameters - Brookbank Deposit

Domain	Pass	X	Y	Z	Min. S	Max. S	Max. S/DH	High-grade Restraining			
								X	Y	Z	Threshold (g/t)
FW	1	60	45	15	7	16	3	None Applied			
	2	80	60	20	4	16	3	20	15	5	40
	3	100	75	30	2	16	3	20	15	5	20
HW	1	60	45	15	7	16	3	None Applied			
	2	80	60	20	4	16	3	20	15	5	20
	3	100	75	30	2	16	3	20	15	5	10
Alteration	1	60	45	15	7	16	3	None Applied			
	2	80	60	20	4	16	3	20	15	5	5
	3	100	75	30	2	16	3	20	15	5	5
Conglomerate	1	100	75	30	2	16	3	20	15	5	5
Metavolcanics	1	100	75	30	2	16	3	20	15	5	7.5

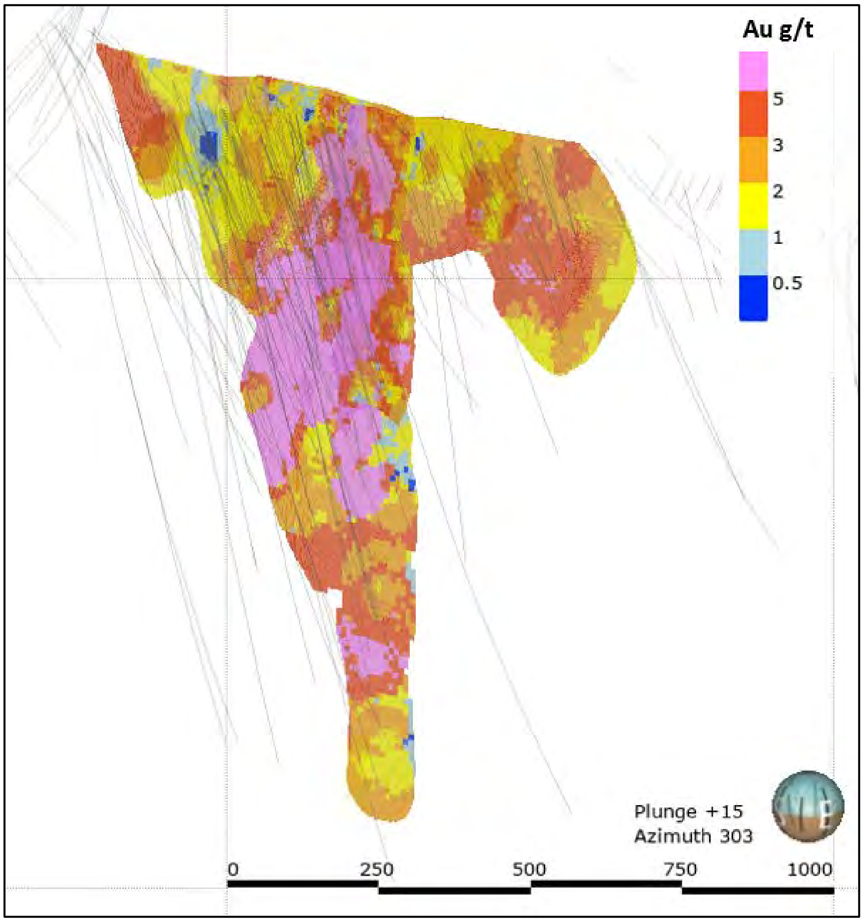
Notes: Min. S = minimum samples; Max. S = maximum samples; S/DH = samples/drill hole

14.2.1.9 Grade Interpolation

OK was the preferred estimator for the FW and alteration domains. Variograms in these domains showed clear structure, and grade smearing was controlled in later estimation passes by using high-grade restraining. See Figure 14.29 for the gold distribution of the FW domain

ID2 was used for the HW domain, and the conglomerate and metavolcanics domains. ID2 is the preferred estimator in these domains due to the lack of interpretable variograms, and the observed reduced grade smearing when compared to OK.

Figure 14.29: Brookbank Gold Grade Distribution in the FW Domain, looking NW



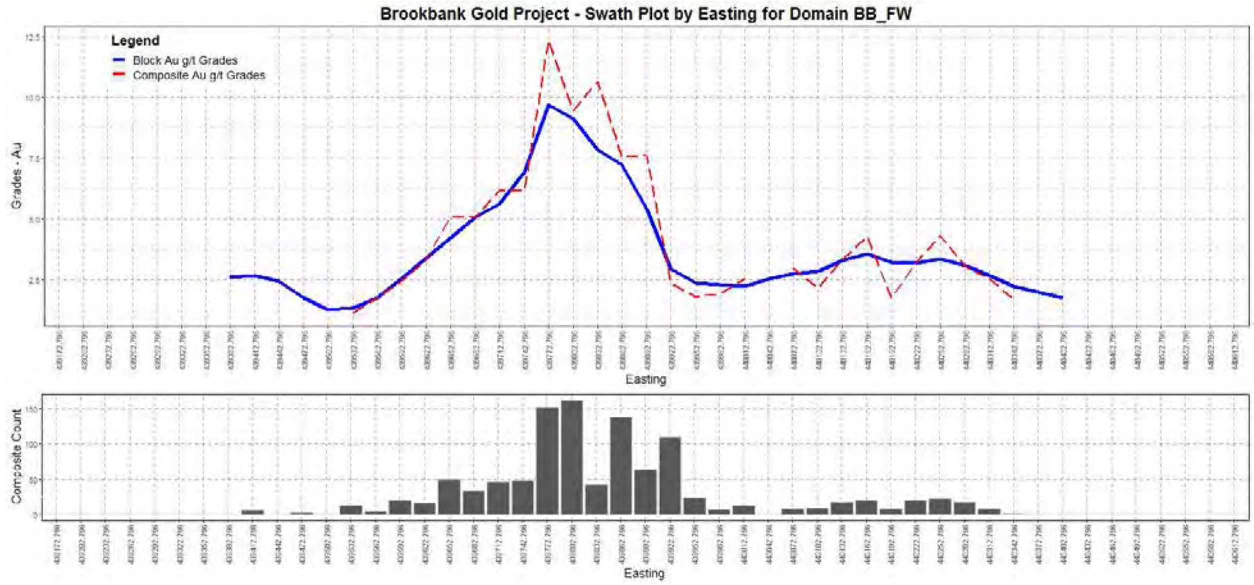
14.2.1.10 Block Model Validation

The block model was validated by visual inspection in plan and section to ensure that block grade estimates reflect the grades seen in intersecting drill holes. In addition, global statistical comparisons were made comparing declustered composites and block grades (Table 14.28), and local validations such as swath plots were used to ensure grade smearing was reduced to a minimum (Figure 14.30).

Table 14.28: Global Statistical Comparison between Blocks and Declustered Composites for all Estimation Passes at Brookbank

Domain	Composites			Blocks		Difference Mean (%)
	No. Comps	Mean	Mean Decl.	No. Blocks	Mean	
FW	1,097	6.71	4.50	187,477	4.74	5.3%
HW	538	3.82	3.76	85,765	3.52	-6.4%
Alteration	5,133	0.30	0.32	670,392	0.33	3.1%

Figure 14.30: Swath Plot Comparing Block Gold Grades (blue) with Capped Composite Gold Grades (red dotted) for the FW Domain, by Easting



GMS found that the global mean grade was comparable to the declustered composites for each domain, and fell within the +-10% acceptable range for this style of deposit. Swath plots showed good local reproduction of composite grades, with block grade slightly conservative within the central portion of the footwall domain. The conglomerate and metavolcanics domains were not validated as they will remain unclassified in the MRE.

14.2.1.11 Determination of Mineral Resources (Open Pit Shell vs. Underground) – All Deposits

The resource block model was examined for open pit and underground economic potential at various cut-off grades. To do this, the block model was subjected to an analysis using a conventional Lerchs-Grossmann algorithm within Whittle™, to define a series of potentially economic open pit shells. All Indicated and Inferred Blocks were considered during pit optimization.

In order to run the Whittle economic pit optimization, GMS adopted certain economic parameters such as operating costs, commodity prices and foreign exchange rates. The metallurgical recovery for Brookbank were derived from the 2009 Technical Report by RPA, where test work indicated recoveries between 93.8% and 96.5%. GMS discounted this to 92%, which was used as a parameter during pit optimization. For Kailey and Key Lake, no metallurgical data was available therefore a 90% metallurgical recovery was assumed. All other parameters were assumed from prior experience with the Hardrock Deposit. In addition, GMS assumed that ore from these three deposits would be treated at the Hardrock plant, therefore an incremental haulage charge was applied per km. Table 14.29 and Table 14.30 and shows the various parameters/assumptions used in the open pit and underground analysis as well as the gold cut-off grades used for reporting the MRE.

Table 14.29: Economic Parameters used in the Open Pit Analysis

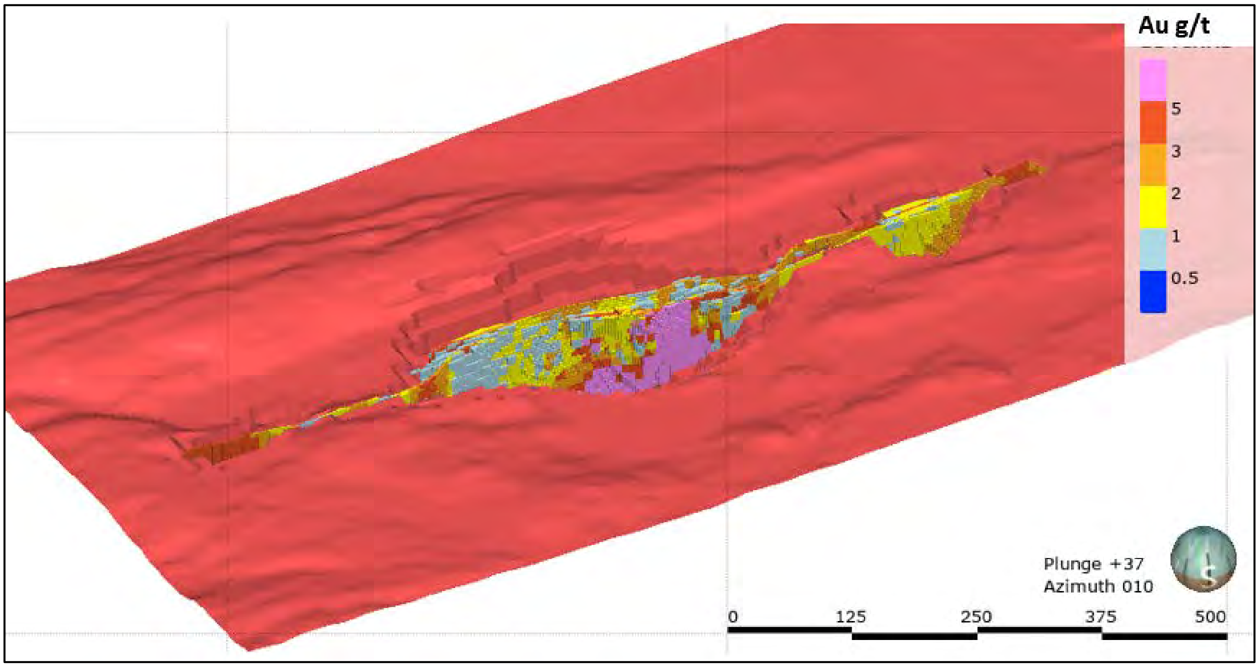
Item	Unit	Brookbank	Kailey	Key Lake
Open Pit Mining Cost	CAD/All Material Tonne	4.00	4.00	4.00
Open Pit Dilution + MiningLoss	%	19.0	12.0	19.0
Processing Cost	CAD/Ore Tonne	7.54	7.54	7.54
G&A Cost	CAD/Ore Tonne	1.59	1.59	1.59
Gold Price	USD/Troy Ounce	1,500.00	1,500.00	1,500.00
Incremental Ore Haulage	CAD/Ore Tonne	17.90	1.70	4.51
Mill Recovery	Percent	92	90	90
Exchange Rate	USD to CAD	1.30	1.30	1.30
Open Pit Calculated Gold Cut-off Grade	**g Au/t**	**0.60**	**0.40**	**0.40**

Table 14.30: Economic Parameters used in the Underground Analysis

Item	Unit	Brookbank
Exchange Rate	C$/US$	1.30
Discount Rate	%	6%
Gold Price	US$/oz	1500
	C$/oz	1950
Transport & Refining Cost	C$/oz	3.00
Royalty Rate	% NSR	3%
Net Gold Value	C$/oz	1888.5
Metallurgical Recovery	%	92%
Mining Dilution	%	20%
Mining Recovery	%	90%
Processing Cost	C$/t milled	8.00
Surface Ore Haulage	C$/t milled	17.90
Diamond Drilling	C$/t milled	1.50
Stope Preparation	C$/t milled	20.00
Mining	C$/t milled	25.00
Services - Surface	C$/t milled	3.00
Service - Mine	C$/t milled	8.00
Service - Mechanical	C$/t milled	3.00
Electrical	C$/t milled	2.00
Technical Services	C$/t milled	2.50
G&A	C$/t milled	10.00
Underground Calculated Gold Cut-off Grade	**g Au/t**	**2.40**

After completing the Whittle pit optimization, the results were re-imported back into Leapfrog where the block model was flagged for the material in the economic pit-shell, with the material outside of the shell being flagged as potential underground material. The resulting pit is shown in Figure 14.31.

Figure 14.31: Brookbank Project Pit Optimization - US$1,500 Pit Shell, MII Blocks



14.2.1.12 Resource Categorization

GMS classified resource blocks in the block model based largely upon the drilling density and the passes criteria, while also accounting for variography results and deposit geometry. The resource categories are shown on Figure 14.32. At this stage, there are no Measured Mineral Resources for the Brookbank Project.

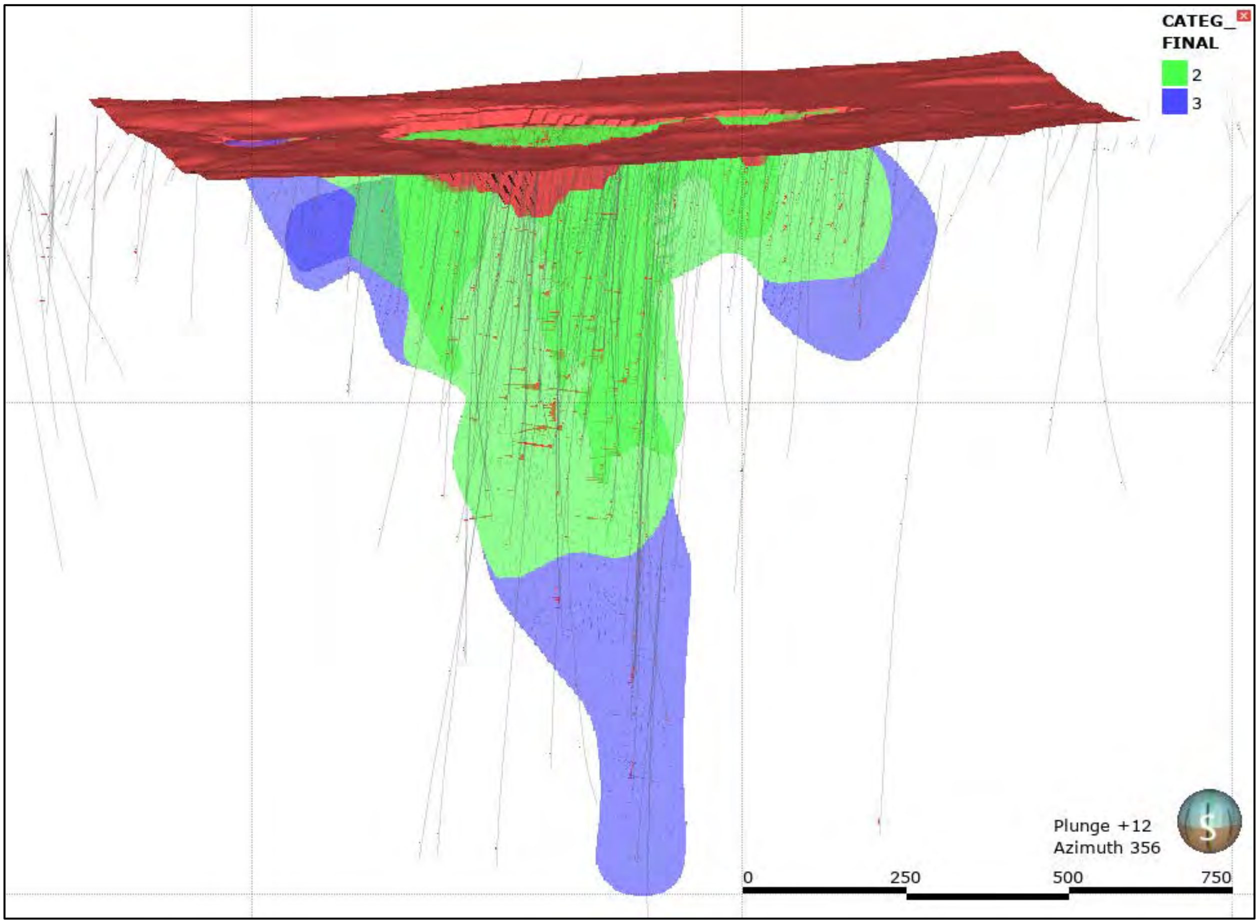
The Indicated Mineral Resource category was assigned to coherent portions of the deposit covered by a 40 m x 40 m drill spacing, and estimated predominantly in Pass 1, including islands of Pass 2 encompassed within. Good visual evidence of adequate sample/drill hole coverage was also considered.

The Inferred Mineral Resource category was assigned to areas outside of the 40 m x 40 m drill spacing, and blocks estimated predominantly in Passes 2 and 3. These areas have limited drill hole information and often include extrapolation of grades towards the boundaries of the wireframe.

The conglomerate and metavolcanics domain were not classified as mineral resources as they contained insignificant tonnage.

The Foxear and Cherbourg Deposits were deemed too small to meet the requirements for Reasonable Prospects for Eventual Economic Extraction ("RPEEE"), however they do represent good near-term targets for expansion drilling.

Figure 14.32: Brookbank Mineral Resource Classification



14.2.1.13 Mineral Resource Statement

The Mineral Resources are summarized in Table 14.31 at cut-off grades of 0.6 g Au/t and 2.4 g Au/t for open pit and underground resources, respectively. The cut-off grades adopted offer the deposit reasonable prospects for eventual economic extraction on the assumptions summarized in Table 14.29.

The estimated Mineral Resources conform to the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves, as required by NI 43-101 – Standards of Disclosure from Mineral Projects.

Table 14.31: Summary of the Brookbank Mineral Resource

Deposit	Cut-off Category	Mineral Resource Category	Tonnes (Mt)	Gold Grade (g Au/t)	Gold Ounces (000's)
Brookbank Project	Open Pit	Measured (M)	-	-	-
		Indicated (I)	1.147	2.24	83
		Subtotal M & I	**1.147**	**2.24**	**83**
		Inferred	0.045	2.07	3
	Underground	Measured (M)	-	-	-
		Indicated (I)	2.281	7.06	517
		Subtotal M & I	**2.281**	**7.06**	**517**
		Inferred	0.706	3.38	77

Notes:

1. *Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability;*
2. *There are no mineral reserves at Brookbank;*
3. *The independent and qualified person for the Brookbank MRE is Mr. James Purchase, P.Geo of GMS.*
4. *The effective date of the Mineral Resource is September 3rd, 2020*
5. *Open pit mineral resources are constrained within an optimized pit shell using a gold price of US$1,500, a CAD:USD exchange rate of 1.3 and a metallurgical recovery of 92%. An incremental ore haulage cost of $17.90 per t milled is also assumed for Brookbank*
6. *Underground mineral resource are reported below the pit optimization, and are constrained by a cut-off grade calculated using the same parameters as the open-pit resource, but with an underground mining cost of $65 per tonne.*
7. *Mineral Resources are quoted at an open-pit lower cut-off of 0.6 g/t, and an underground cut-off of 2.4 g/t*

Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing or other relevant issues.

14.2.1.14 Mineral Resource Sensitivity

The block model was reported at varying cut-offs for the open-pit and underground components of the mineral resource to understand the sensitivity of the tonnes, grade and ounces to changes in the economic cut-off. The results are presented in Table 14.32 and Table 14.33.

Table 14.32: Brookbank Open-Pit Mineral Resource Sensitivity

Cut-off Grade (g Au/t)	Indicated Category			Inferred Category		
	Tonnes (million)	Gold Grade (g Au/t)	Gold Ounces (000's)	Tonnes (million)	Gold Grade (g Au/t)	Gold Ounces (000's)
2.0	0.481	3.81	59	0.020	3.67	2
1.5	0.611	3.37	66	0.020	3.63	2
1.0	0.814	2.83	74	0.027	3.01	3
0.8	0.936	2.59	78	0.032	2.64	3
0.6	**1.147**	**2.24**	**83**	**0.045**	**2.07**	**3**
0.4	1.606	1.74	90	0.076	1.43	3

Table 14.33: Brookbank Underground Mineral Resource Sensitivity

Cut-off Grade (g Au/t)	Indicated Category			Inferred Category		
	Tonnes (million)	Gold Grade (g Au/t)	Gold Ounces (000's)	Tonnes (million)	Gold Grade (g Au/t)	Gold Ounces (000's)
4.0	1.425	9.44	432	0.104	5.39	18
3.5	1.619	8.76	456	0.217	4.49	31
3.0	1.880	7.99	483	0.377	3.97	48
2.4	**2.281**	**7.06**	**517**	**0.706**	**3.38**	**77**
2.0	2.577	6.50	538	0.888	3.14	90

14.2.1.15 QP Commentary

When compared to the 2012 MRE by Micon International, the 2019 MRE by GMS for the Brookbank Deposit has resulted in a reduction in M&I open-pit ounces (83 KOz vs. 171 KOz), and an increase in underground ounces (517 KOz vs. 429 KOz). This is primarily due to a shallower pit optimization in 2019 (a result of an improved understanding of the economic parameters) and an increased cut-off grade to report the open-pit resource. The underground ounces have also grown due to the shallowing of the pit optimization, but also partially due to the refined geological interpretation removing internal waste.

14.2.2 Key Lake

14.2.2.1 Drill hole Database

The Key Lake Deposit has been tested by diamond drilling over a strike length of 2,300 m and down to a vertical depth of 300 m. The Key Lake drill hole database includes 312 drill holes totalling 63,919 for which 23,112 m was assayed.

The drill holes are generally on a 50 m grid spacing in the central and eastern portions of the deposit, with some areas infilled to 25 m. The western portion of the deposit is drilled at 100 m spacing. The main components of the database are the collar, survey, assay and lithology tables which were validated as described in Subsection 12.2.

14.2.2.2 Topography

Topography was supplied by GGM in the form of a wireframe with a spatial resolution of 10 m. The topography matched well with the drill hole collars, with no major discrepancies identified.

14.2.2.3 Geological Modelling Approach

The Key Lake gold deposit is hosted with a package of sediments (greywacke and arkose), iron formations and narrow porphyry dykes. Mineralization generally follows the regional foliation, aligned with the porphyry dyke that acts as a marker horizon. The deposit shows both structural and lithological controls, with mineralization hosted predominantly in the arkosic units and felsic dykes. Only small amount of mineralization is found in the iron formations. Gold mineralization occurs within multiple quartz-carbonate stringers and fine sulphide disseminations that results in sub-vertical, erratic zones that are difficult to interpret.

The Key Lake Deposit has been modelled as twelve discrete, continuous zones named KL-1 to KL-12 (Figure 14.33). The zones are generally wide therefore no minimum mining thickness was considered during modelling (Figure 14.34). An overburden surface was also modelled from the lithology logging, and varies between 10 m and 20 m thick above the deposit.

The Dome-era drilling were used in the interpretation but excluded from the estimation as the assays and drill hole locations could not be verified. Five additional holes were also ignored from the Goldstone-era drilling due to unusual drill traces and erroneous surveys. These holes are KL-10-007B, KL-10-014, KL-10-051, KL-11-099 and KL-11-123.

Figure 14.33: Modelled Solids of the Key Lake Deposit, Looking NW

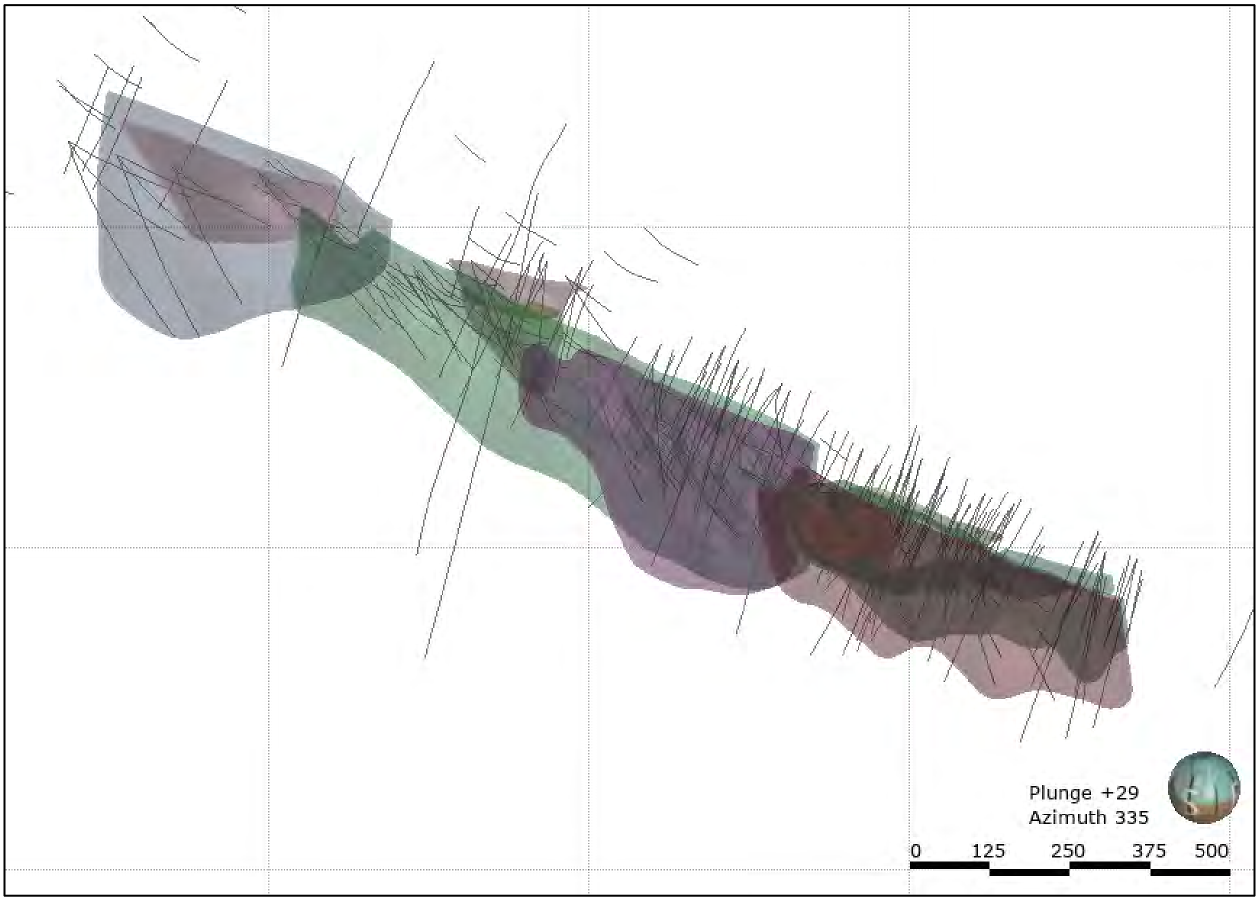
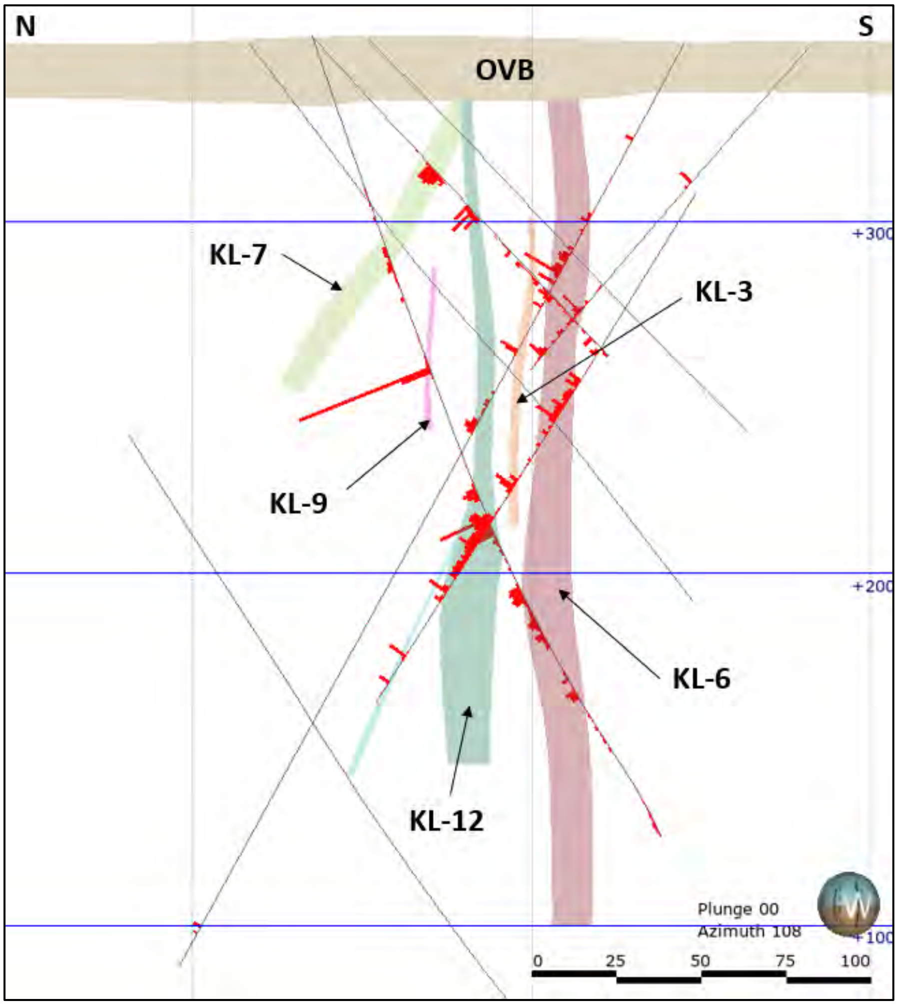


Figure 14.34: Typical Section (looking south-east) showing the Mineralization Wireframes and Overburden. Bar Charts on Drilling Traces show Gold Grades



14.2.2.4 Assay Capping and Compositing

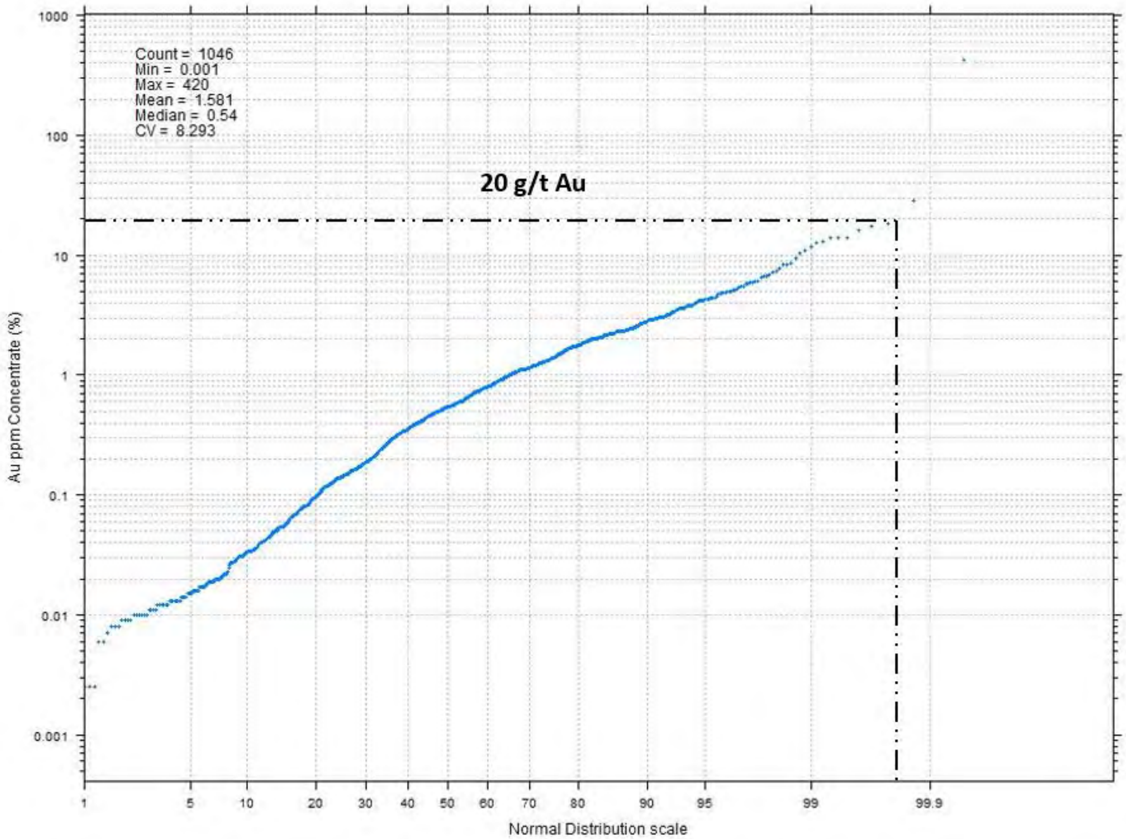
Grade capping levels were determined using probability plots of the various domains, and applied to the assay intervals. Overall, roughly 12% of the metal was removed by capping. Length-weighted assay statistics and capping levels are shown in Table 14.34. An example of the probability plot for the KL-5 domain is shown in Figure 14.35.

Table 14.34: Length-Weighted Assays Statistics showing Grade Capping Levels and Metal Loss Factors

Domain	Number of Assays	Max (g Au/t)	Uncut Mean (g Au/t	High Grade Capping (g Au/t)	Cut Mean (g Au/t)	# Samples Cut	% Samples Capped	% Loss Metal Factor
KL-1	157	252	2.36	25	1.64	3	1.9%	-30%
KL-2	49	38.3	1.44	20	1.26	2	4.1%	-13%
KL-3	87	46.8	1.67	15	1.26	2	2.3%	-25%
KL-4	819	229	1.53	15	1.27	5	0.6%	-17%
KL-5	1,046	420	1.24	20	1.10	2	0.2%	-12%
KL-6	895	121	1.10	20	0.98	1	0.1%	-10%
KL-7	168	12.4	1.35	none	1.35	0	0.0%	0%
KL-8	113	38.6	1.12	15	1.05	1	0.9%	-6%
KL-9	44	20.65	1.51	none	1.51	0	0.0%	0%
KL-10	59	8.82	2.26	none	2.26	0	0.0%	0%
KL-11	54	7.83	1.18	none	1.18	0	0.0%	0%
KL-12	506	7.62	0.89	none	0.89	0	0.0%	0%
Total	**3,997**							**-12%**

*Note: % Loss metal factors calculated from length * grade and does not consider the spatial location of the outliers*

Figure 14.35: Example of a Probability Plot for the KL-5 Domain



Core sampling was undertaken at 1.0 m and 1.5 m intervals and were broken down on visual lithological and alteration contacts. Considering the scale of the deposit and its suitability for open-pit mining, GMS has applied a 2 m compositing run-length split by domain, with any residuals less than 0.5 m added to the last composite. A minimum coverage of 50% was required to create a composite. Missing intervals were replaced with a grade of 0.001 g Au/t.

Length-weighted composite statistics of drilling used in the estimation is shown in Table 14.35. The coefficient of variations are generally low for this style of deposit (less than 2.0)

Table 14.35: Length-weighted 2 m Composite Statistics of Capped Gold Grades by Domain

Domain	Number of 2 m Comps	Min (g Au/t)	Max (g Au/t)	Mean (g Au/t)	Median (g Au/t)	St Dev.	Variance	Coeff. Of Variation
KL-1	88	0.010	12.51	1.64	0.82	2.55	6.51	1.55
KL-2	28	0.047	8.49	1.24	0.63	1.81	3.28	1.46
KL-3	53	0.015	7.59	1.23	0.70	1.49	2.23	1.22
KL-4	407	0.001	10.00	1.32	0.96	1.43	2.05	1.09
KL-5	612	0.001	19.10	1.02	0.53	1.48	2.18	1.45
KL-6	471	0.001	10.17	0.97	0.73	1.07	1.13	1.10
KL-7	90	0.001	7.57	1.27	0.79	1.29	1.66	1.01
KL-8	60	0.001	5.28	1.01	0.71	1.10	1.21	1.09
KL-9	18	0.001	12.64	2.10	0.33	3.43	11.78	1.63
KL-10	39	0.001	7.45	1.96	1.79	1.82	3.32	0.93
KL-11	38	0.001	5.72	0.90	0.48	1.31	1.71	1.45
KL-12	259	0.001	5.12	0.92	0.67	0.87	0.76	0.95

14.2.2.5 Variography

Experimental variograms were constructed for the better populated domains (KL-5, KL-5, KL-6 and KL-12) using the capped gold composite intervals. Nugget sills were estimated from downhole variograms at a 2 m lag spacing. Various experimental variogram types were used, and the normal-score transformed variograms showed the most coherent structure.

Nugget variances were moderate to high, and were interpreted at 30 – 50% of the total sill. The major axis was interpreted to be aligned along strike dipping shallowly to the WNW (dip = 20 degrees, dip direction = 290 Azimuth), with the semi-major axis dipping steeply to the ESE (dip = 70, dip direction = 110). The major axis showed maximum ranges in the order of 60 – 70m, with semi-major ranges slightly less in the order of 50 – 60 m, although the variograms were difficult to interpret.

Due to the amount of internal waste present inside the wireframes, and the difficulties obtaining interpretable variograms, Inverse Distance Cubed ("ID3") will be used as an interpolator and the results of the variography will be used to guide the dimensions of the search ellipses.

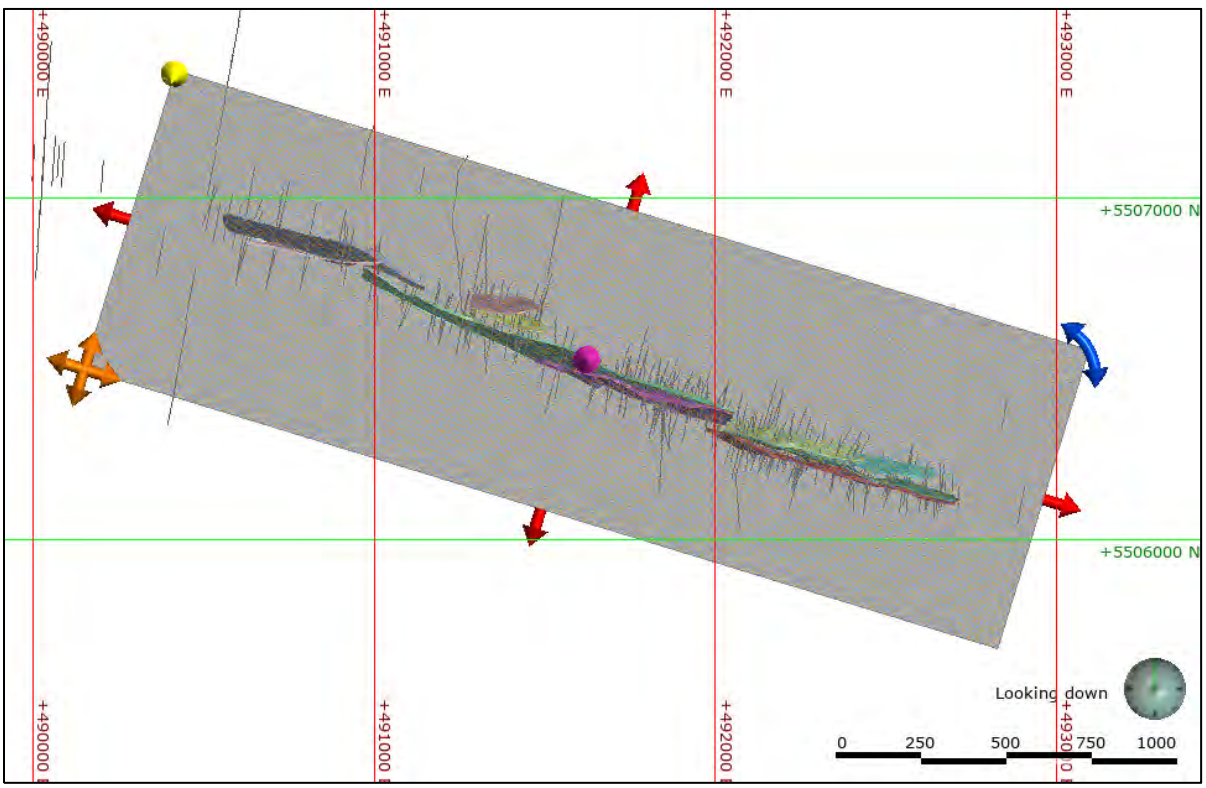
14.2.2.6 Block Modelling

The block model definition is presented in Table 14.36. The upper limit was defined by the surface topography. The parent block size was based primarily on drill hole spacing, envisaged selective mining unit ("SMU") and geometry of the deposit. The block model was sub-blocked using the domain wireframes. A volume check of the block model versus the mineralization envelope revealed a good representation of the volume of the solid. Figure 14.36 shows a plan view of the Key Lake block model location.

Table 14.36: Key Lake Deposit Block Model Attributes

Item	X	Y	Z
Origin Coordinates	490,150.00	5,506,500.00	600
Block Extents (m)	2,800	915	800
Number of Parent	280	183	160
Parent Block Size (m)	10	5	5
Sub-Block Size (m)	2.5	1.25	1.25
Rotation		17 Degrees Clockwise	

Figure 14.36: Plan View of Key Lake Block Model, Wireframes and Drill Traces



14.2.2.7 Bulk Density Data

Bulk Density data was supplied by GGM in the form of a excel spreadsheet containing bulk density readings by lithology and deposit. In total, 66 measurements were taken using the Archimedes method of measuring the weight of the core sampling in water and in air. Table 14.37 presents the bulk density data available for the Key Lake Deposit.

Table 14.37: Statistical Summary of Bulk Density Data for the Key Lake Deposit

Deposit	Lithology	No. of Meas.	Mean Density (g/cc)	St. Dev Density (g/cc)
Key Lake	Arkose	11	2.77	0.09
	Greywacke	50	2.74	0.09
	Iron Formation	2	3.19	0.12
	Porphyry	3	2.78	0.05

GMS applied the bulk density values described above into the block model using the lithology model developed in Leapfrog GEO™ provided by GGM. Overburden was assumed to be 2.0 g/cc.

14.2.2.8 Search Ellipsoids

Due to the undulating nature of the veins, GMS decided to use dynamic anisotropy to locally adjust the search ellipse orientations according to the local dip and dip direction of the vein wireframe. A surface was built using the mid-points of the vein, and was used as an input to determine the rotation angles of the search ellipse.

The search ellipse configurations were defined using variography and drill spacing as a guide combined with the geometry of the deposit. A three-pass estimation procedure was used for the interpolation. For all passes, the maximum number of samples per drill hole was set to control the number of drill holes in the interpolation.

For Pass 1, a minimum number of samples of 7, a maximum number of samples of 16, and a maximum samples per drill hole of 3 was applied. This ensured that a minimum of 3 drill holes were required to estimate blocks in the first pass.

For Pass 2, a minimum number of samples of 4, a maximum number of samples of 16, and a maximum samples per drill hole of 3 was applied. This ensured that a minimum of 2 drill holes were required to estimate blocks in the second pass.

For Pass 3, a minimum number of samples of 1, a maximum number of samples of 16, and a maximum samples per drill hole of 3 was applied. This ensured that a minimum of 1 drill holes were required to estimate blocks in the third pass.

In regard to sequencing, Pass 1 took precedence over Pass 2, and Pass 2 took precedence over Pass 3.

GMS applied high-grade restraining for the blocks outside of the 12 modelled domains, and only for the third estimation pass. Thresholds were chosen from probability plots. The search parameters adopted for grade interpolation are summarized in Table 14.38.

Table 14.38: Summary of Search Parameters – Key Lake Deposit

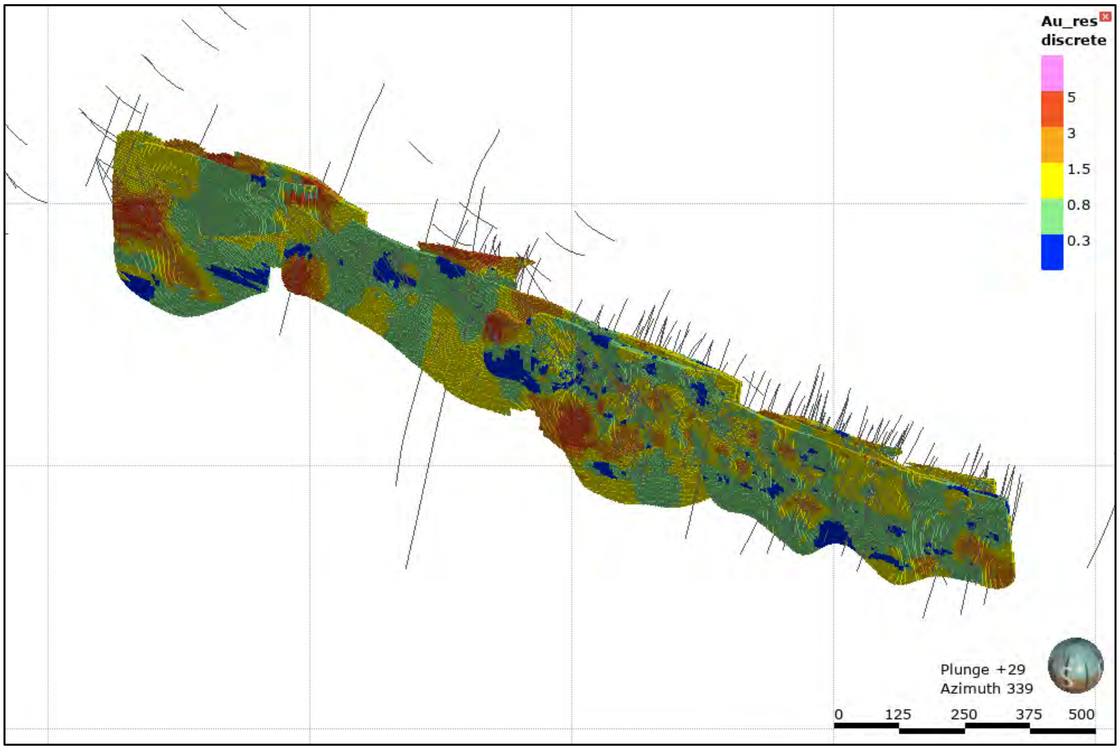
Domain	Pass	X	Y	Z	Min. S	Max. S	Max. S/DH	High-grade Restraining			
								X	Y	Z	Threshold (g/t)
KL-1 to KL-12	1	45	45	15	7	16	3	None Applied			
	2	60	60	25	4	16	3				
	3	100	100	35	1	16	3				
Outside	1	45	45	15	7	16	3	None Applied			
	2	60	60	25	4	16	3				
	3	100	100	35	1	16	3	25	25	10	5

Notes: Min. S = minimum samples; Max. S = maximum samples; S/DH = samples/drill hole

14.2.2.9 Grade Interpolation

Inverse Distance Cubed ("ID3") was the preferred estimator for the Key Lake Deposit. Variograms showed poor structure and were difficult to interpret. In addition, the inclusion of significant internal dilution in the wireframes was unavoidable during modelling, therefore there was a requirement to minimized grade smearing. Block grades are shown in Figure 14.37.

Figure 14.37: Key Lake Gold Grade Distribution in Resource Block Model, looking NW



14.2.2.10 Block Model Validation

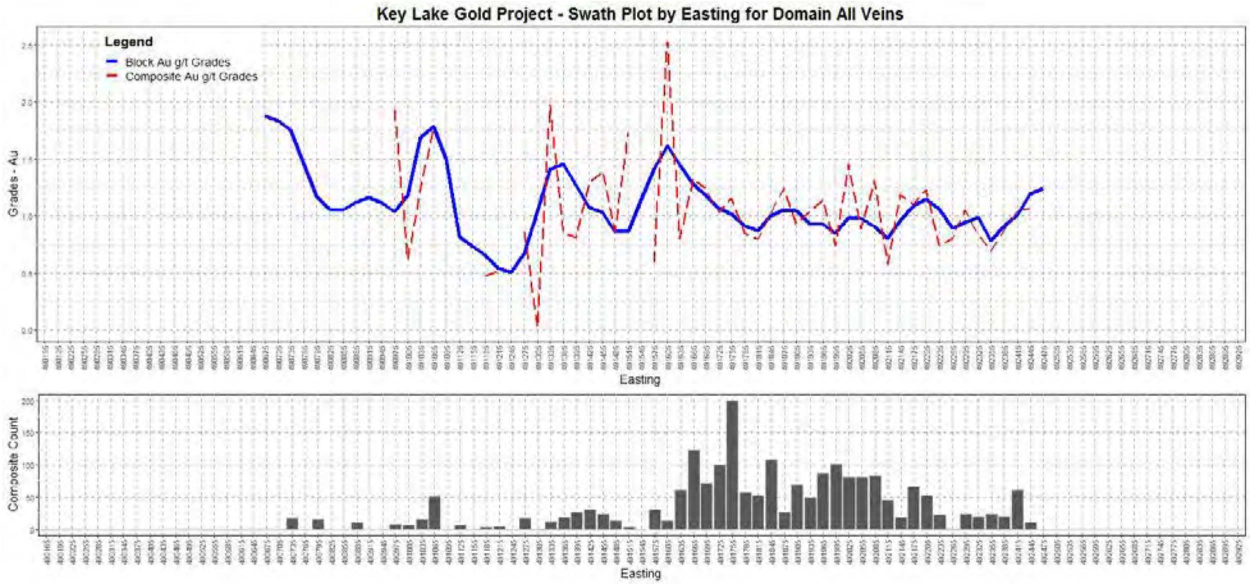
The block model was validated by visual inspection in plan and section to ensure that block grade estimates reflect the grades seen in intersecting drill holes. In addition, global statistical comparisons were made comparing declustered composites and block grades (Table 14.39), and local validations such as swath plots were used to ensure grade smearing was reduced to a minimum (Figure 14.38).

Table 14.39: Global Statistical Comparison between Blocks and Declustered Composites for all Estimation Passes at Key Lake

Domain	Composites			Blocks		Difference Mean (%)
	No. Comps	Mean	Mean Decl.	No. Blocks	Mean	
KL-1	85	1.58	1.71	202,856	1.49	-13%
KL-2	29	1.12	0.95	53,984	0.93	-2%
KL-3	46	1.14	1.45	12,456	1.19	-18%
KL-4	380	1.24	1.24	310,443	1.17	-6%
KL-5	594	1.00	1.02	210,295	1.02	0%
KL-6	454	0.94	0.91	207,606	0.90	-2%
KL-7	90	1.24	1.33	40,375	1.26	-5%
KL-8	57	0.92	0.84	48,224	0.96	14%
KL-9	17	2.02	2.51	4,008	2.04	-19%
KL-10	32	2.02	2.37	23,870	2.27	-4%
KL-11	36	0.84	0.74	22,028	1.01	37%
KL-12	240	0.87	0.91	123,825	0.91	0%
All 12 Domains	**2060**	**1.08**	**1.10**	**1,259,970**	**1.11**	**1%**

The comparison between the mean grades of declustered composites and blocks vary domain-by-domain as expected, and GMS believes these results are acceptable and mostly fall within the +-10% margin of error. The swath plots show good local accuracy of the gold estimate for the twelve domains.

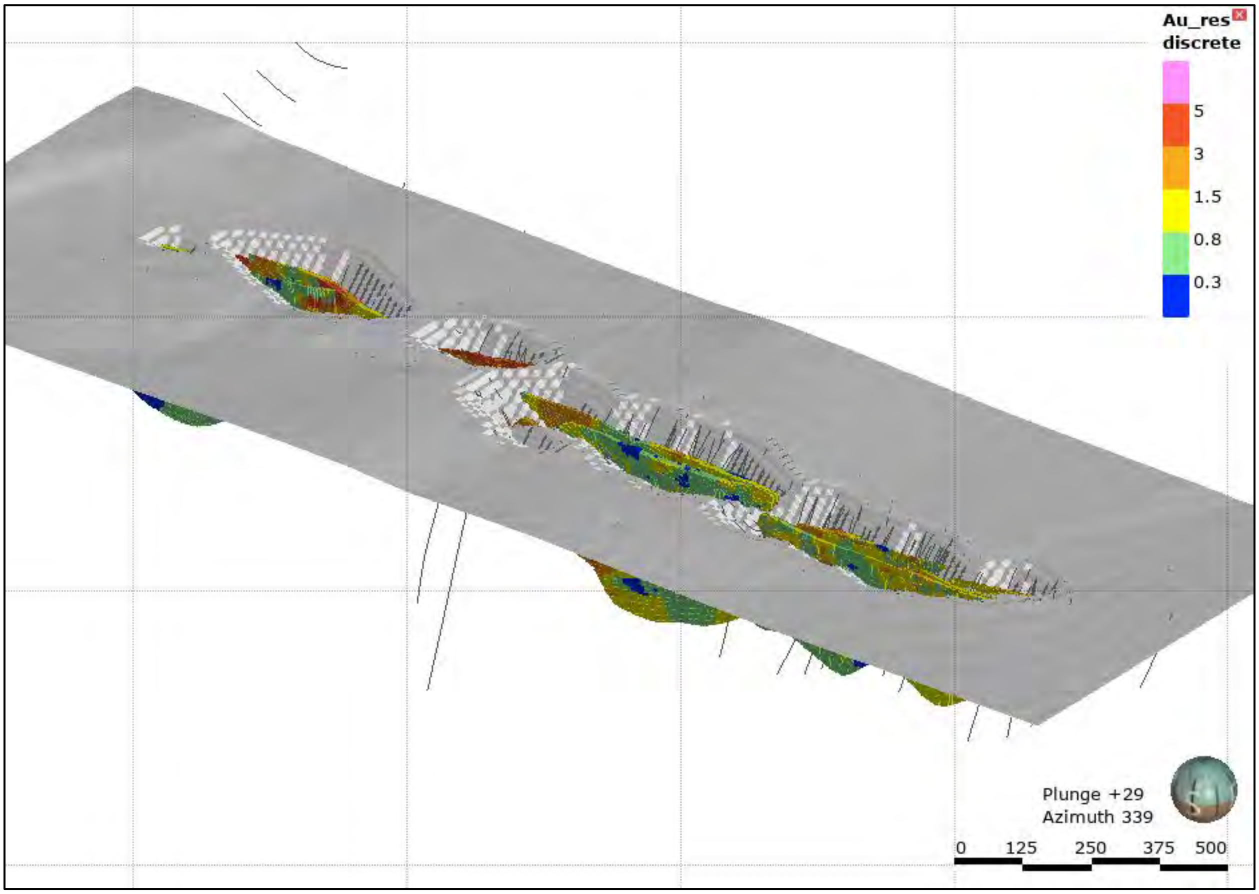
Figure 14.38: Swath Plot Comparing Block Gold Grades (blue) with Capped Composite Gold Grades (red dotted) for the 12 Domains Grouped Together, by Easting



14.2.2.11 Determination of Mineral Resources (Open Pit Shell vs. Underground)

The resource block model was examined for open pit and underground economic potential at various cut-off grades. To do this, the block model was subjected to an analysis using a conventional Lerchs-Grossmann algorithm within Whittle™, to define a series of potentially economic open pit shells. All Indicated and Inferred Blocks were considered during pit optimization. The parameters used during the pit optimization process are showed in Table 14.29, and the chosen pit is shown in Figure 14.39.

Figure 14.39: Key Lake Deposit Pit Optimization - US$1,500 Pit Shell, MII Blocks



14.2.2.12 Underground Voids

There has been limited past production at the Key Lake Deposit, which was formerly known as the Jellicoe Mine. GMS was supplied an underground void model for the shaft and underground drifts, which was incorporated into the block model and were assigned a density of zero. No stopes were modelled, however production at the Jellicoe mine was very limited with production records indicating 14,722 tonnes were mined for 5,675 ounces of gold produced.

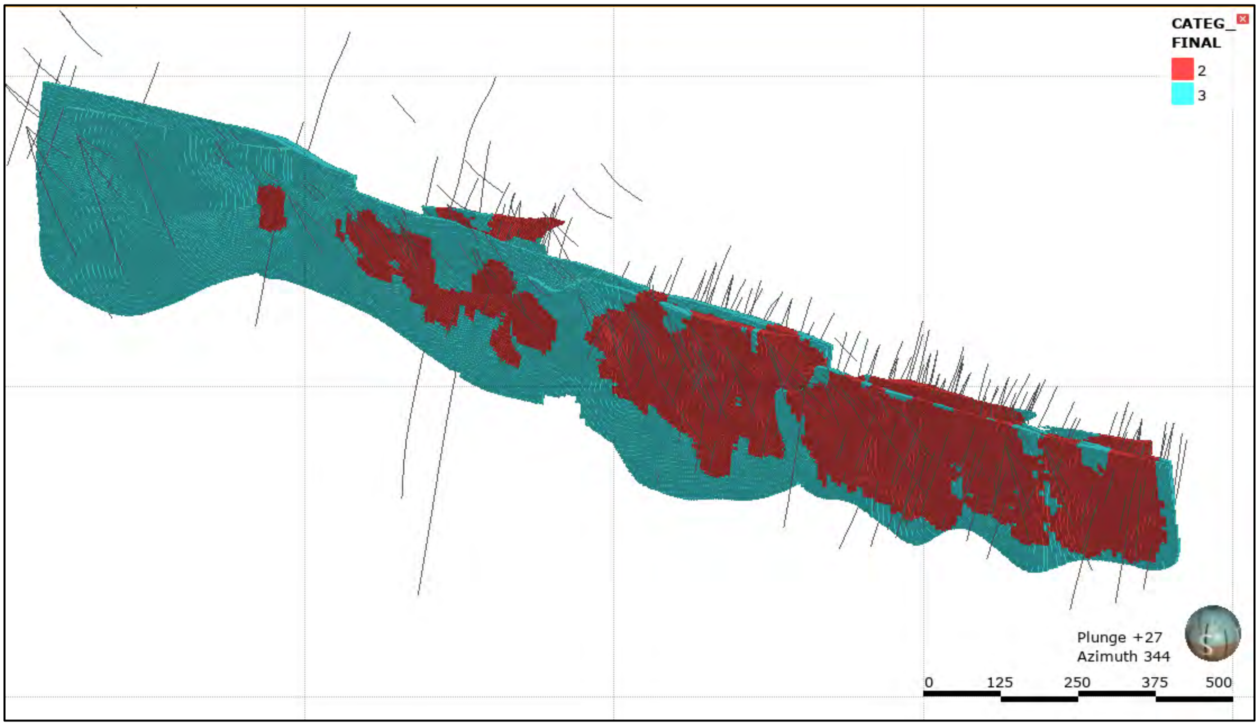
14.2.2.13 Resource Categorization

The Key Lake block model was classified based largely upon estimation pass and distance to nearest composites. The resource categories are shown in Figure 14.40. At this stage, there are no Measured Mineral Resources for the Key Lake Deposit.

The Indicated Mineral Resource category was assigned to coherent portions of the deposit estimated in Pass 1 and Pass 2, with a distance to closest composite less than 35 m.

The Inferred Mineral Resource category was assigned to blocks estimated in Pass 1 and Pass 2 with a distance to closest composite greater than 35 m, and blocks estimated in Pass 3 . In addition, all blocks in domains KL-1 and KL-2 were downgraded to Inferred category due to a wider drill spacing, and all blocks outside of the 12 modelled domains and below 0 RL elevation were assigned to Inferred category.

Figure 14.40: Key Lake Deposit Coloured by Resource Category. Red = Indicated, Light Blue = Inferred



14.2.2.14 Mineral Resource Statement

The Mineral Resources are summarized in Table 14.40 at a lower cut-off grade of 0.4 g Au/t for the open pit category. The cut-off grades adopted offer the deposit reasonable prospects for eventual economic extraction using the assumptions summarized in Table 14.29.

The estimated Mineral Resources conform to the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves, as required by NI 43-101 – Standards for Disclosure for Mineral Projects.

Table 14.40: Summary of the 2020 Key Lake Mineral Resource

Deposit	Cut-off Category	Mineral Resource Category	Tonnes (Mt)	Gold Grade (g Au/t	Gold Ounces (000's)
Key Lake	Open Pit	Measured (M)	-	-	-
		Indicated (I)	3.761	1.16	141
		Subtotal M & I	**3.761**	**1.16**	**141**
		Inferred	1.839	1.39	82

Notes:

1. *Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability;*
2. *No mineral reserves are published at Key Lake;*
3. *The independent and qualified person for the Key Lake MRE is Mr. James Purchase, P.Geo of GMS.*
4. *The effective date of the Mineral Resource is September 3rd, 2020*
5. *Open pit mineral resources are constrained within an optimized pit shell using a gold price of US$1,500, a CAD:USD exchange rate of 1.3 and a metallurgical recovery of 90%. An incremental ore haulage cost of $4.51 per t milled is also assumed for Key Lake;*
6. *No underground Mineral Resources are quoted;*
7. *Mineral Resources are quoted at an open-pit lower cut-off of 0.4 g/t.*

Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing or other relevant issues.

14.2.2.15 Mineral Resource Sensitivity

The block model was reported at varying cut-offs to understand the sensitivity of the tonnes, grade and ounces to changes in the economic cut-off. The results are presented in Table 14.41.

Table 14.41: Key Lake Open-Pit Mineral Resource Sensitivity

Cut-off Grade (g Au/t)	Indicated Category			Inferred Category		
	Tonnes (million)	Gold Grade (g Au/t)	Gold Ounces (000's)	Tonnes (million)	Gold Grade (g Au/t)	Gold Ounces (000's)
1.5	0.855	2.18	60	0.613	2.43	48
1.0	1.859	1.66	99	1.027	1.94	64
0.8	2.418	1.49	116	1.322	1.71	73
0.6	3.007	1.33	129	1.525	1.58	77
0.4	**3.761**	**1.16**	**141**	**1.839**	**1.39**	**82**
0.2	5.250	0.91	154	2.418	1.12	87

14.2.2.16 QP Commentary

The 2020 MRE includes some changes compared to the previous MRE released in 2012 by Micon International:

- An underground void model has been incorporated into the block model;
- The geological wireframes have been refined to better honour the lithology and structural setting of Key Lake, placing more emphasis on the porphyry dykes and arkosic units;
- Additional data verification and compilation has been undertaken since 2012 which included a comprehensive review of assay certificates, QAQC and general data robustness. A result of this review was the exclusion of the Dome-era drilling from this MRE.

The grade of the M&I portion of the deposit has remained similar (1.16 g/t in 2020 vs.1.17 g/t in 2020), however the ounces have increased (141 KOz in 2020 vs. 97 KOz in 2012) due to a deeper pit optimization used to constrain the resource. GMS has not reported an underground resource at Key Lake as we do not believe that continuity of mineralization has been demonstrated to support an underground resource, and that more drilling is required at depth to better understand the higher-grade intercepts to the west of the deposit.

14.2.3 Kailey Deposit

The Kailey Deposit is located 3 km NW of the Hardrock Project, and was originally an underground mine named the Little Long Lac Mine.

14.2.3.1 Drill hole Database

The Kailey Deposit has been tested by diamond drilling over a strike length of 800 m and down to a vertical depth of 500 m. The Kailey drill hole database includes 82 drill holes totalling 34,599 m for which 27,892 m was assayed.

The drill holes are on an irregular grid spacing of around 50 m to 70 m, and at depth the spacing becomes 80 m to 100 m. Historical drilling data exists on the project digitized from hard copies, however no significant assays are available and all the historical data is unverifiable.

14.2.3.2 Topography

No detailed topography was available, so GMS constructed a topography using the drill collars. The area around the historical Little Long Lac Mine is generally flat.

14.2.3.3 Geological Modelling Approach

The Kailey Deposit is hosted with a sedimentary sequence of greywacke and massive arkose units. Mineralization is constrained to steeply dipping, ENE-striking axial planes of an antiformal-synformal feature, with fold noses steeply plunging to the WSW. The deposit shows both structural and lithological controls, with mineralization hosted predominantly within a massive arkosic unit. Only minor amounts of mineralization is found in the greywacke. High-grade gold mineralization occurs within three discrete narrow, high-grade veins that were the target of past production activities. Wide, lower-grade Fe-carbonate and sericite alteration haloes centralized on the axial planes of the folds are also present. Mineralization is associated with fine sulphide disseminations that are difficult to observed in drill core.

The Kailey Deposit has been modelled as three, wide (20 – 30m), continuous zones named Kailey, Main and No.9 (Figure 14.41). The zones are generally wide therefore no mining thickness was considered during modelling (Figure 14.34). An overburden surface was also modelled from the lithology logging, and varies between 5 m and 20 m thick above the deposit.

Only Premier-era drilling was used during the interpretation and estimation. In addition, drill hole PLL08013 was excluded due to uncertainties surrounding the downhole surveys.

Figure 14.41: Isometric View looking NW of the Three Domains at the Kailey Deposit. Also shown are Underground Workings

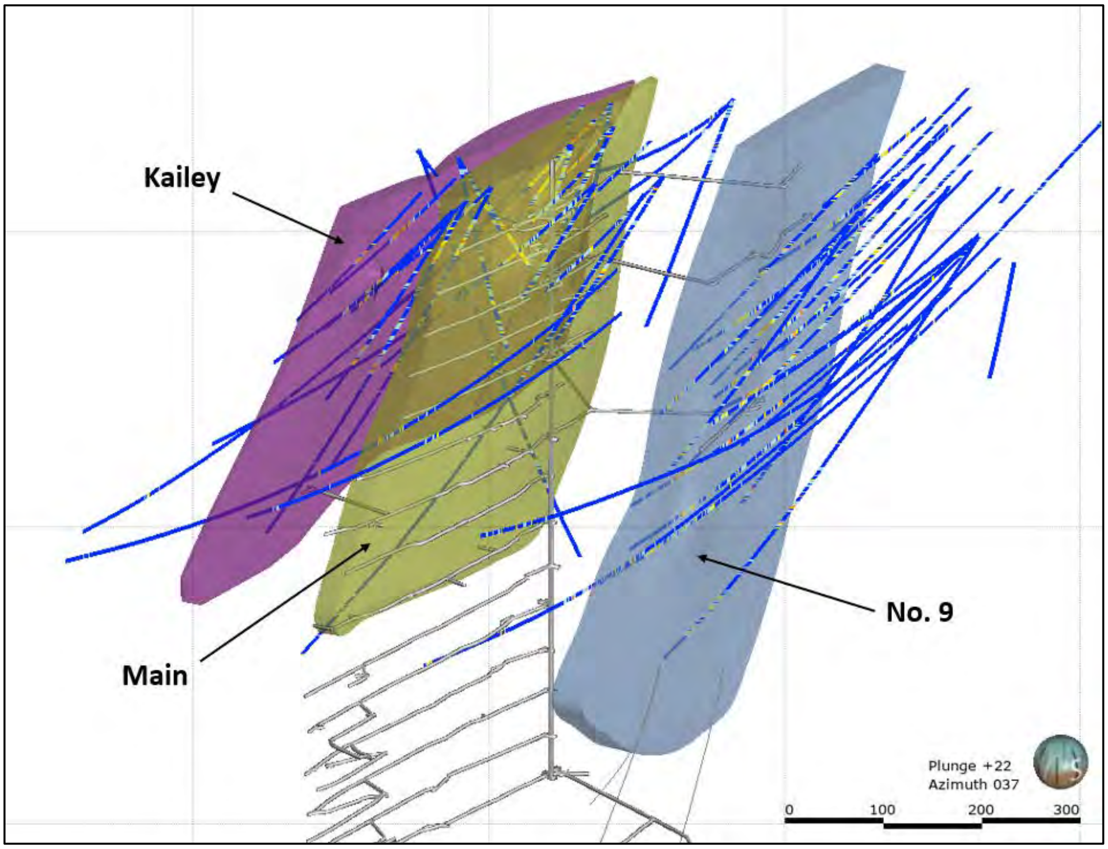
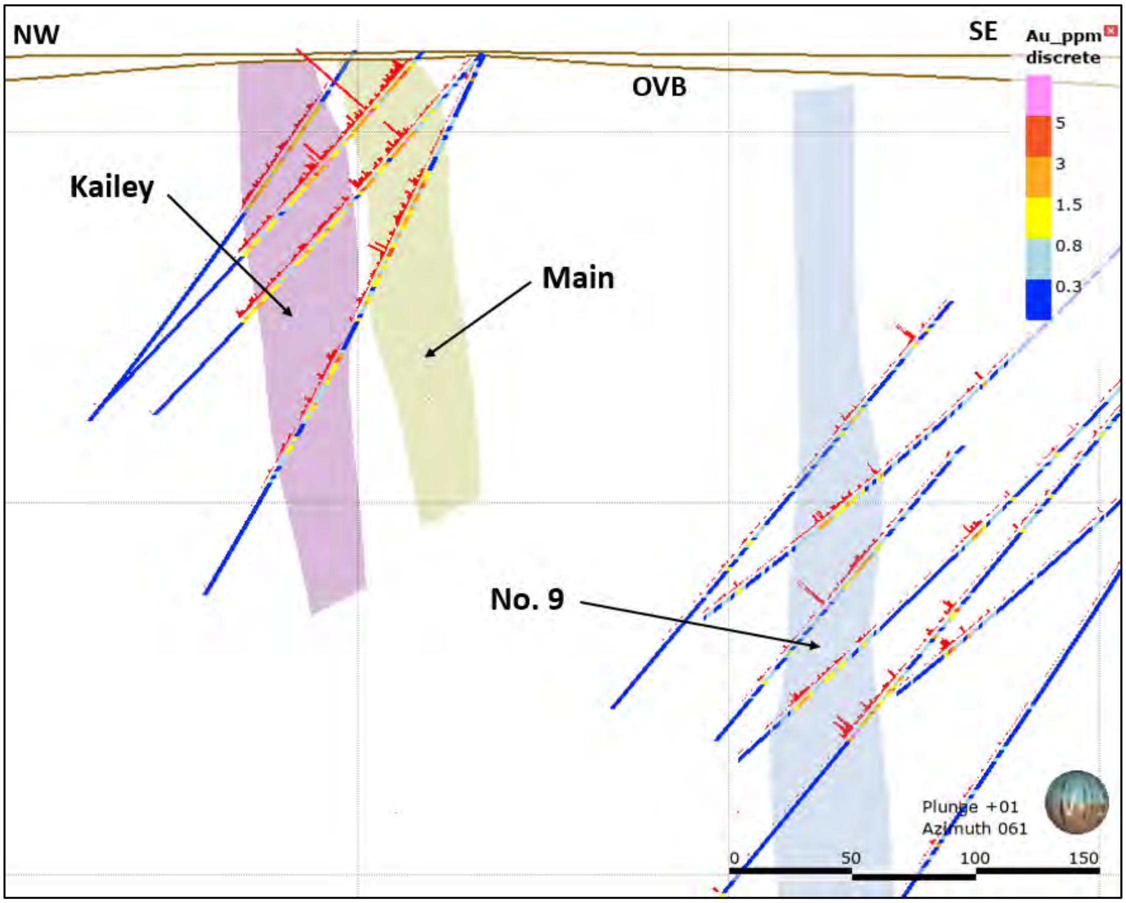


Figure 14.42: Typical Section showing Kailey and Main Domains Near Surface, and No. 9 Domain at Depth. Drill Holes Coloured by Gold



14.2.3.4 Assay Capping and Compositing

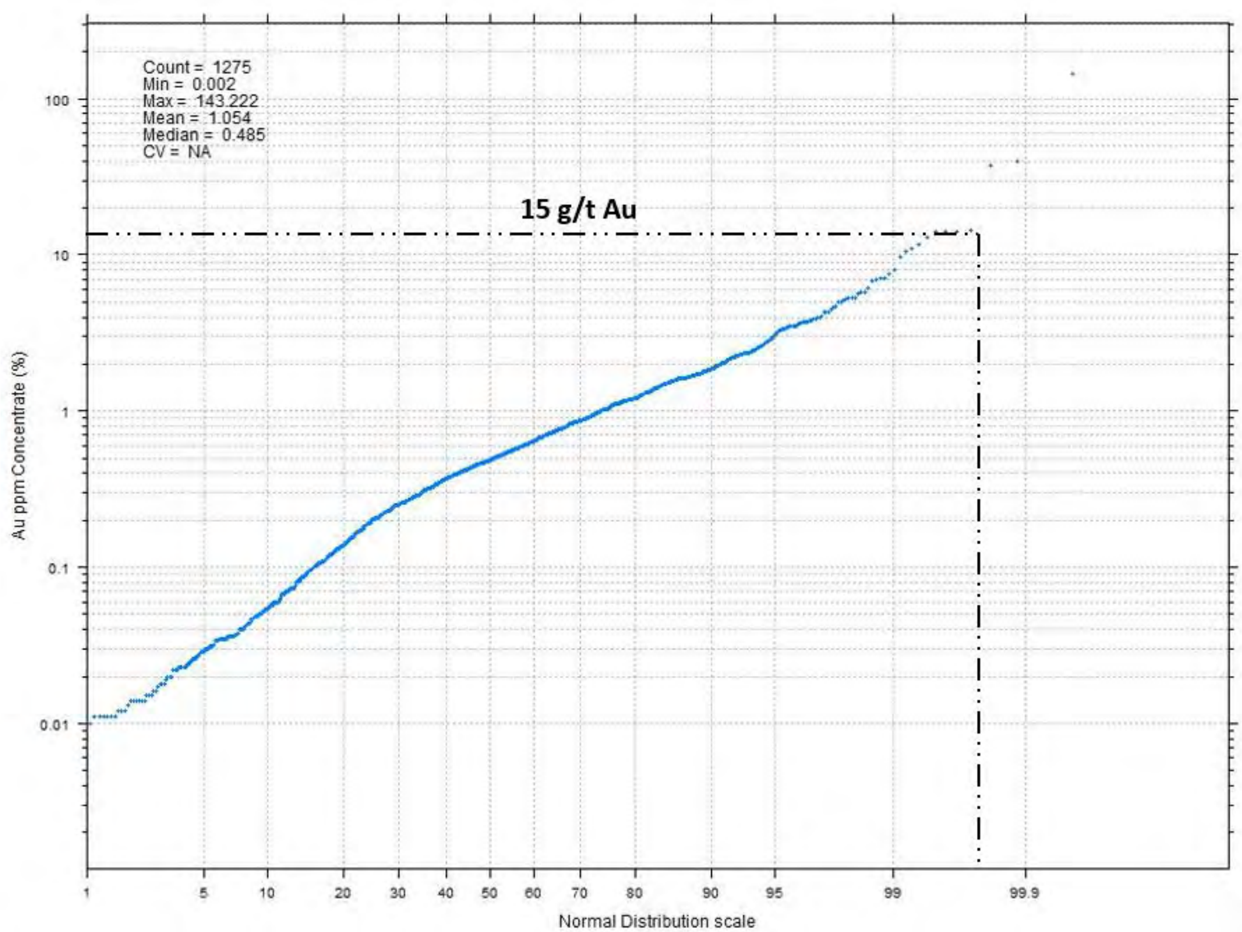
Grade capping levels were determined using probability plots of the various domains, and applied to the assay intervals. Overall, roughly 5% of the metal was removed by capping. Length-weighted assay statistics and capping levels are shown in Table 14.42. An example of the probability plot for the KL-5 domain is shown in Figure 14.43.

Table 14.42: Length-Weighted Assays Statistics Showing Grade Capping Levels and Metal Loss Factors

Domain	Number of Assays	Max (g Au/t)	Uncut Mean (g Au/t)	High Grade Capping (g Au/t)	Cut Mean (g Au/t)	# Samples Cut	% Samples Capped	% Loss Metal Factor
Kailey	1,080	72.5	1.03	10	0.97	2	0.2%	5.8%
Main	1,275	143.2	0.98	15	0.89	3	0.2%	9.0%
No. 9	1,219	20.9	0.76	15	0.76	5	0.4%	0.8%
Outside	23,929	211.4	0.13	15	0.12	9	0.0%	4.8%
Total	**27,503**							**5.2%**

*Notes: % Loss metal factors calculated from length * grade and does not consider the spatial location of the outliers*

Figure 14.43: Example of a Probability Plot for the Main Domain



Core sampling was undertaken at 1.0 m and 1.5 m intervals and were broken down on visual lithological and alteration contacts. Considering the scale of the deposit and its suitability for open-pit mining, GMS has applied a 2 m compositing run-length split by domain, with any residuals less than 0.5 m added to the last composite. A minimum coverage of 50% was required to create a composite. Missing intervals were replaced with a grade of 0.001 g Au/t. Breakthroughs into underground voids were omitted from the compositing.

Length-weighted composite statistics of drilling used in the estimation is shown in Table 14.43. The coefficient of variations are generally low for this style of deposit.

Table 14.43: Length-weighted 2 m Composite Statistics of Capped Gold Grades by Domain

Domain	Number of 2 m Comps	Min (g/t Au)	Max (g/t Au)	Mean (g Au/t)	Median (g Au/t)	St Dev.	Variance	Coeff. Of Variation
Kailey	541	0.001	6.18	0.96	0.79	0.85	0.72	0.88
Main	638	0.003	8.94	0.90	0.56	1.15	1.33	1.29
No 9 Vein	540	0.003	9.78	0.76	0.51	1.00	0.99	1.30
Outside	9702	0.001	7.78	0.12	0.02	0.35	0.12	2.82

14.2.3.5 Variography

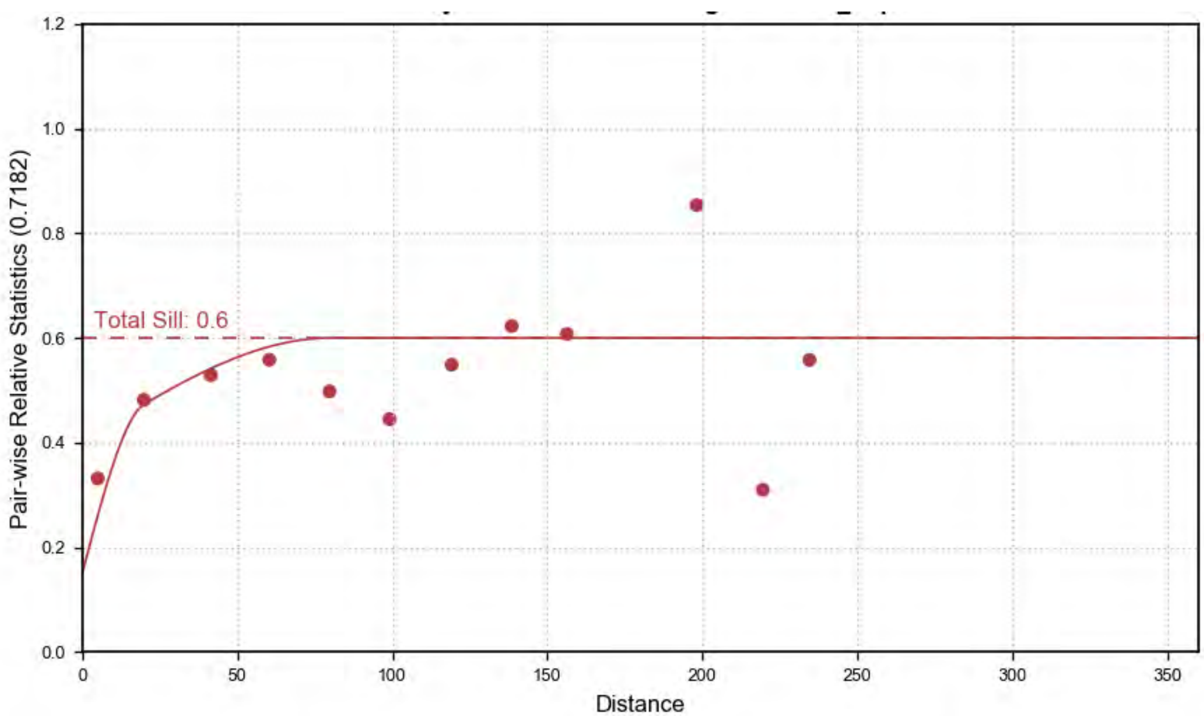
Experimental variograms were constructed for the three domains (Kailey, Main and No. 9) using the capped gold composite intervals. Nugget sills were estimated from downhole variograms at a 2 m lag spacing. Pair-wise variograms were interpreted as they showed the most coherent structure.

GMS was able to interpret variograms for all three domains. Variograms for the composites outside of the three domains (the “Outside” domain) showed poor structure. Variogram parameters are shown in Table 14.44, and an example is shown in Figure 14.44.

Table 14.44: Variogram Parameters for the Kailey Deposit

Zone	Axis	Nugget (C0)	Sill 1 (C1)	Range 1 (R1)	Sill 2 (C2)	Range 2 (R2)	Dip	Dip Azimuth	Pitch
Kailey	Major	0.15	0.25	20	0.2	80	85	170	90
	Semi-			20		80			
	Minor			5		10			
Main	Major	0.25	0.17	30	0.2	90	90	162	117
	Semi-			25		70			
	Minor			5		10			
No. 9	Major	0.20	0.30	30	0.25	90	90	155	128
	Semi-			30		90			
	Minor			5		10			

Figure 14.44: Example Variogram for the Major Axis of the Kailey Domains



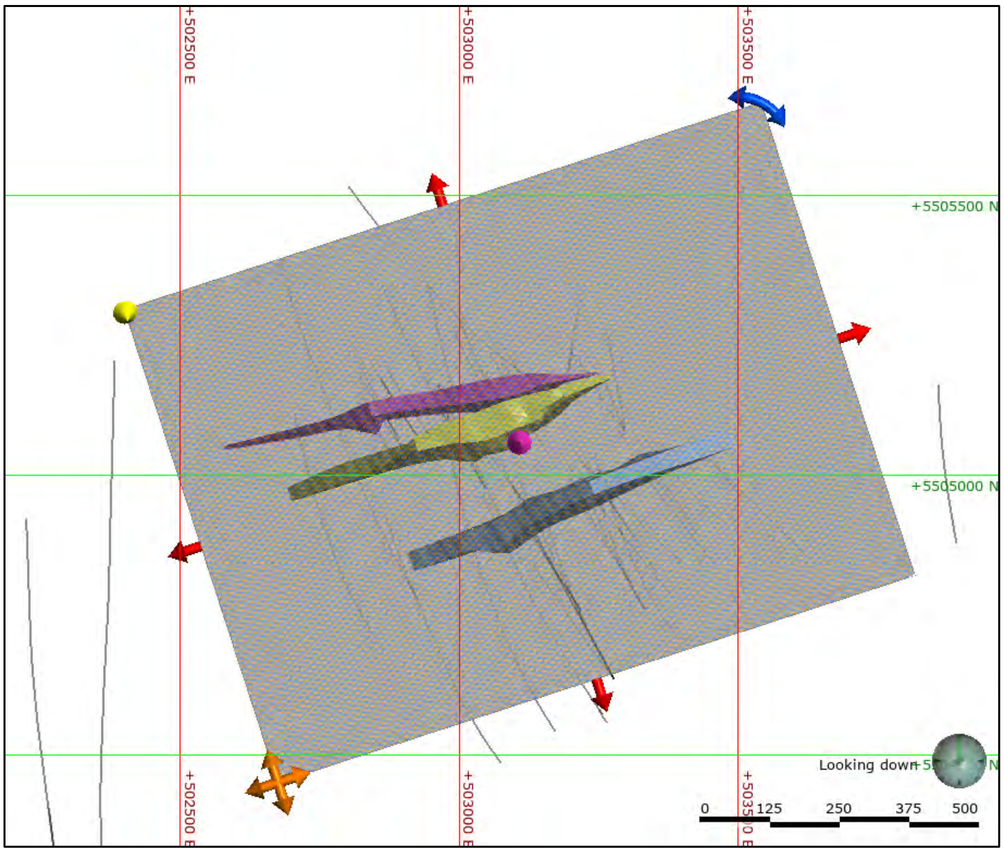
14.2.3.6 Block Modelling

The block model definition is presented in Table 14.45. The upper limit was defined by the surface topography. The parent block size was based primarily on drill hole spacing, envisaged selective mining unit ("SMU") and geometry of the deposit. The block model was sub-blocked using the domain wireframes. A volume check of the block model versus the mineralization envelope revealed a good representation of the volume of the solid. Figure 14.45 shows a plan view of the Kailey block model location.

Table 14.45: Kailey Deposit Block Model Attributes

Item	X	Y	Z
Origin Coordinates	502,675.00	5,504,450.00	385
Block Extents (m)	1,200	890	690
Number of Parent	120	178	138
Parent Block Size (m)	10	5	5
Sub-Block Size (m)	2.5	1	1.25
Rotation		18 Degrees Anti-clockwise	

Figure 14.45: Plan view of Kailey Block Model, Wireframes and Drill Traces



14.2.3.7 Bulk Density Data

Bulk Density data was supplied by GGM in the form of a excel spreadsheet containing bulk density readings by lithology and deposit. In total, 91 measurements were taken using the Archimedes method of measuring the weight of the core sampling in water and in air. Table 14.46 presents the bulk density data available for the Kailey Deposit.

Table 14.46: Statistical Summary of Bulk Density Data for the Key Lake Deposit

Deposit	Lithology	No. of Meas.	Mean Density (g/cc)	St. Dev Density (g/cc)
Key Lake	Arkose	15	2.72	0.10
	Greywacke	55	2.75	0.13
	Iron Formation	4	3.46	0.31
	Gabbro	17	2.82	0.14

GMS applied the bulk density values described above into the block model using the lithology model developed in Leapfrog GEO™ provided by GGM. Overburden was assumed to be 2.0 g/cc.

14.2.3.8 Search Ellipsoids

GMS decided to use dynamic anisotropy to locally adjust the search ellipse orientations according to the local dip and dip direction of the vein wireframe. A surface was built using the mid-points of the vein, and was used as an input to determine the rotation angles of the search ellipse.

The search ellipse configurations were defined using variography and drill spacing as a guide combined with the geometry of the deposit. A four-pass estimation procedure was used for the interpolation. For all passes, the maximum number of samples per drill hole was set to control the number of drill holes in the interpolation.

For Pass 1, a minimum number of samples of 7, a maximum number of samples of 16, and a maximum samples per drill hole of 3 was applied. This ensured that a minimum of 3 drill holes were required to estimate blocks in the first pass.

For Pass 2, a minimum number of samples of 4, a maximum number of samples of 16, and a maximum samples per drill hole of 3 was applied. This ensured that a minimum of 2 drill holes were required to estimate blocks in the second pass.

For Pass 3, a minimum number of samples of 3, a maximum number of samples of 16, and a maximum samples per drill hole of 3 was applied. This ensured that a minimum of 1 drill holes were required to estimate blocks in the third pass.

For Pass 4, a minimum number of samples of 1, a maximum number of samples of 16, and a maximum samples per drill hole of 3 was applied. This pass was designed to ensure all blocks were filled.

In regard to sequencing, Pass 1 took precedence over Pass 2, Pass 2 took precedence over Pass 3, and Pass 3 took precedence over Pass 4. No high-grade restraining was used at Kailey. The search parameters adopted for grade interpolation are summarized in Table 14.47.

Table 14.47: Summary of Search Parameters – Kailey Deposit

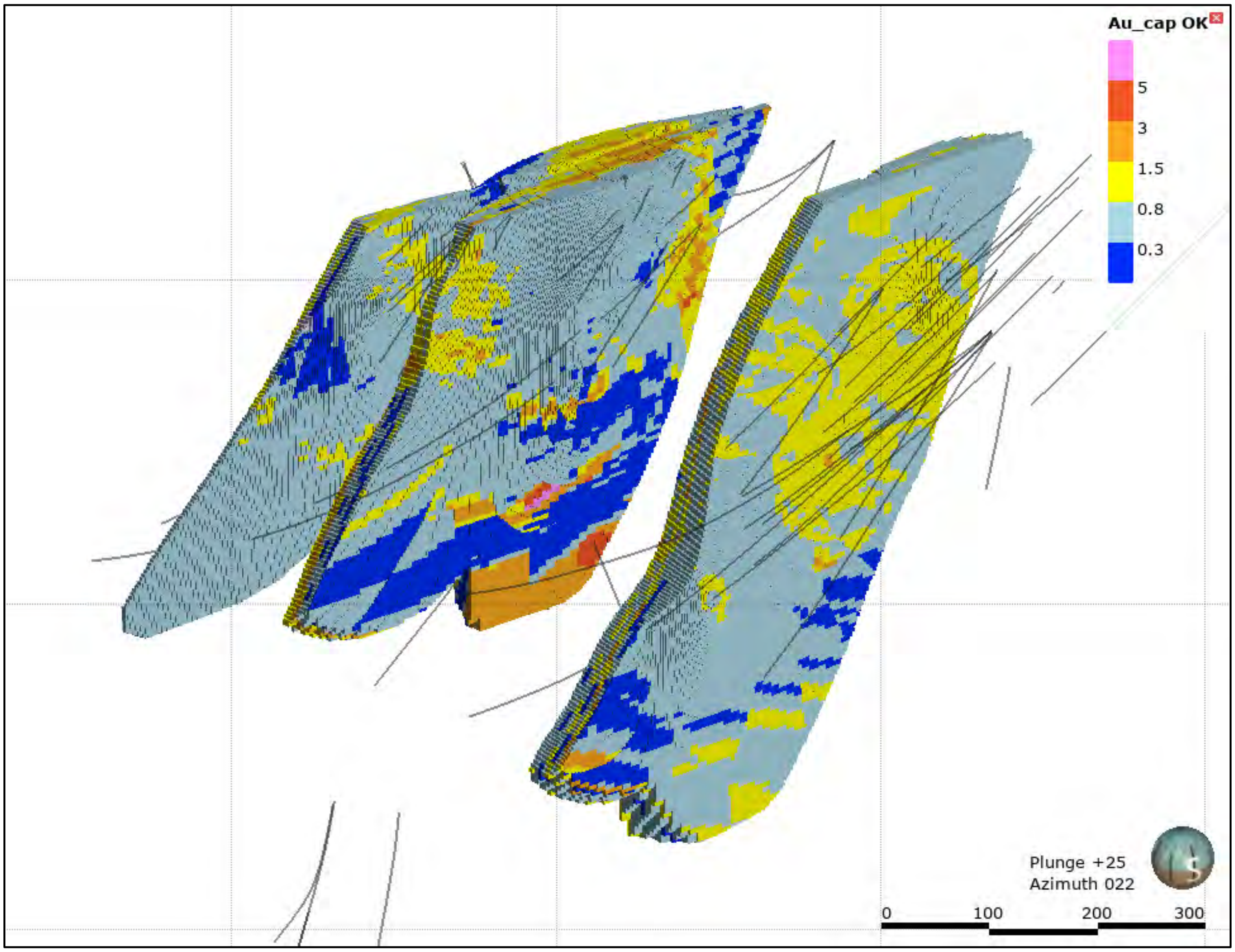
Domain	Pass	X	Y	Z	Min. S	Max. S	Max. S/DH	High-grade Restraining			
								X	Y	Z	Threshold (g/t)
Main	1	60	45	15	7	16	3	None Applied			
	2	80	60	25	4	16	3				
	3	100	75	35	3	16	3				
	4	150	150	50	1	16	3				
Kailey, No. 9 and Outside	1	60	60	15	7	16	3				
	2	80	80	25	4	16	3				
	3	100	100	35	3	16	3				
	4	150	150	50	1	16	3				

Notes: Min. S = minimum samples; Max. S = maximum samples; S/DH = samples/drill hole

14.2.3.9 Grade Interpolation

OK was the preferred estimator for the Kailey, No.9 and the Main domains. Variograms showed good structure and were readily interpretable. All blocks outside of these three domains were estimated using ID2. Estimated blocks are shown in Figure 14.46.

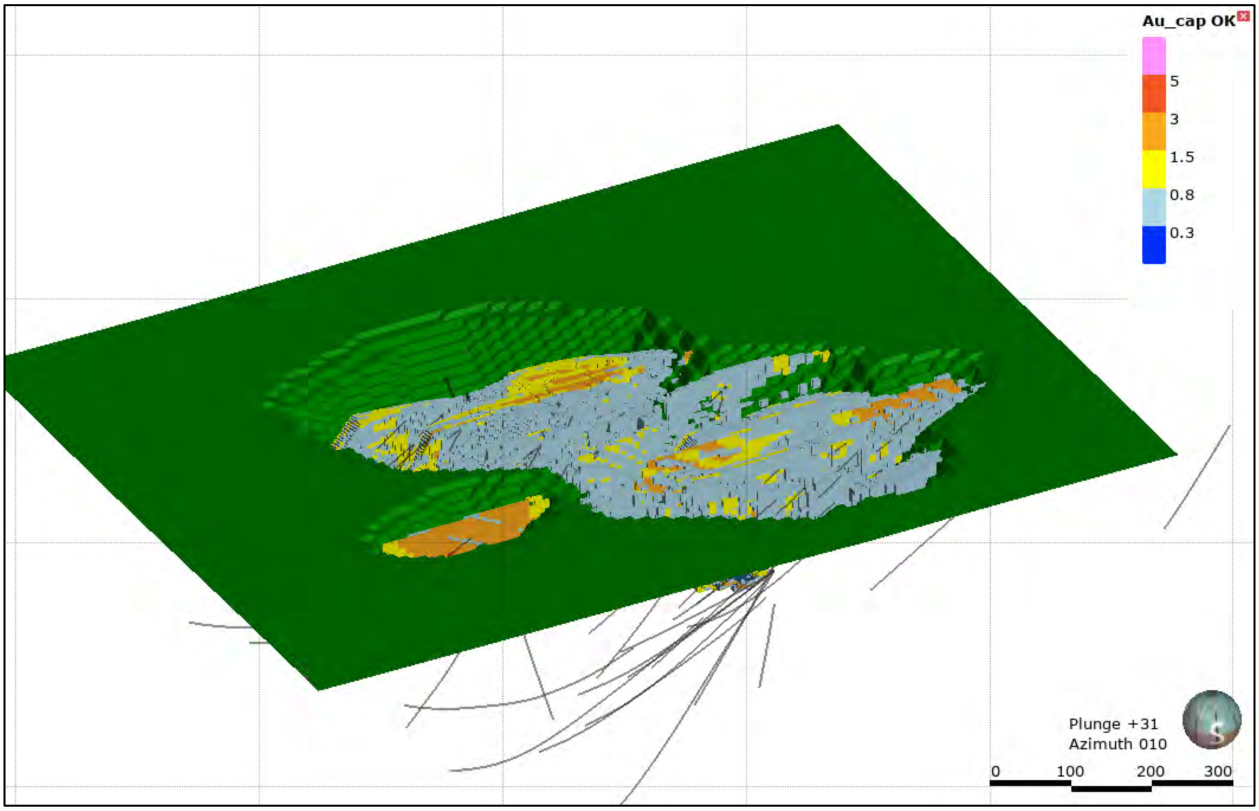
Figure 14.46: Kailey Gold Grade Distribution in Resource Block Model, looking NW



14.2.3.10 Determination of Mineral Resources (Open Pit Shell vs. Underground)

The resource block model was examined for open pit and underground economic potential at various cut-off grades. To do this, the block model was subjected to an analysis using a conventional Lerchs-Grossmann algorithm within Whittle™, to define a series of potentially economic open pit shells. All Indicated and Inferred Blocks were considered during pit optimization. The parameters used during the pit optimization process are showed in Table 14.29, and the chosen pit is shown in Figure 14.47.

Figure 14.47: Key Lake Deposit Pit Optimization - US$1,500 Pit Shell, MII Blocks

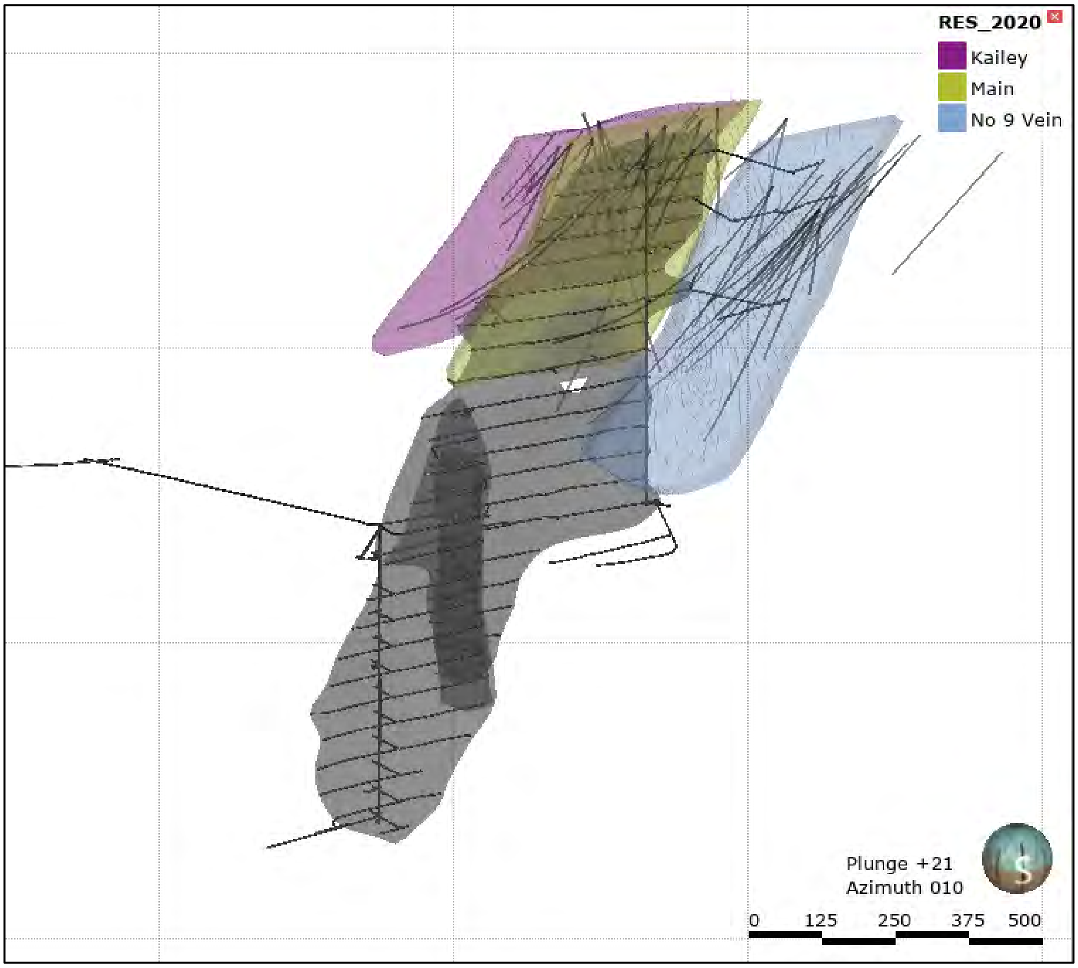


14.2.3.11 Underground Voids

There has been significant past production at the Kailey Deposit, which was formerly known as the Little Long Lac Mine and produced 1.78 Mt at an average grade of 10.6 g Au/t for 607 KOz over an 18-year mine life. GMS was supplied an underground void model for the shaft, underground drifts, and an 3D wireframe representing the mined main vein. These were incorporated into the block model and were assigned a density of zero.

The workings extend to a depth of 1,200 m and the mine operated between 1934 and 1952. Figure 14.48 shows the underground workings in relation to the three modelled domains.

Figure 14.48: Underground Void Model (grey) and Three Modelled Domains at Kailey



14.2.3.12 Resource Categorization

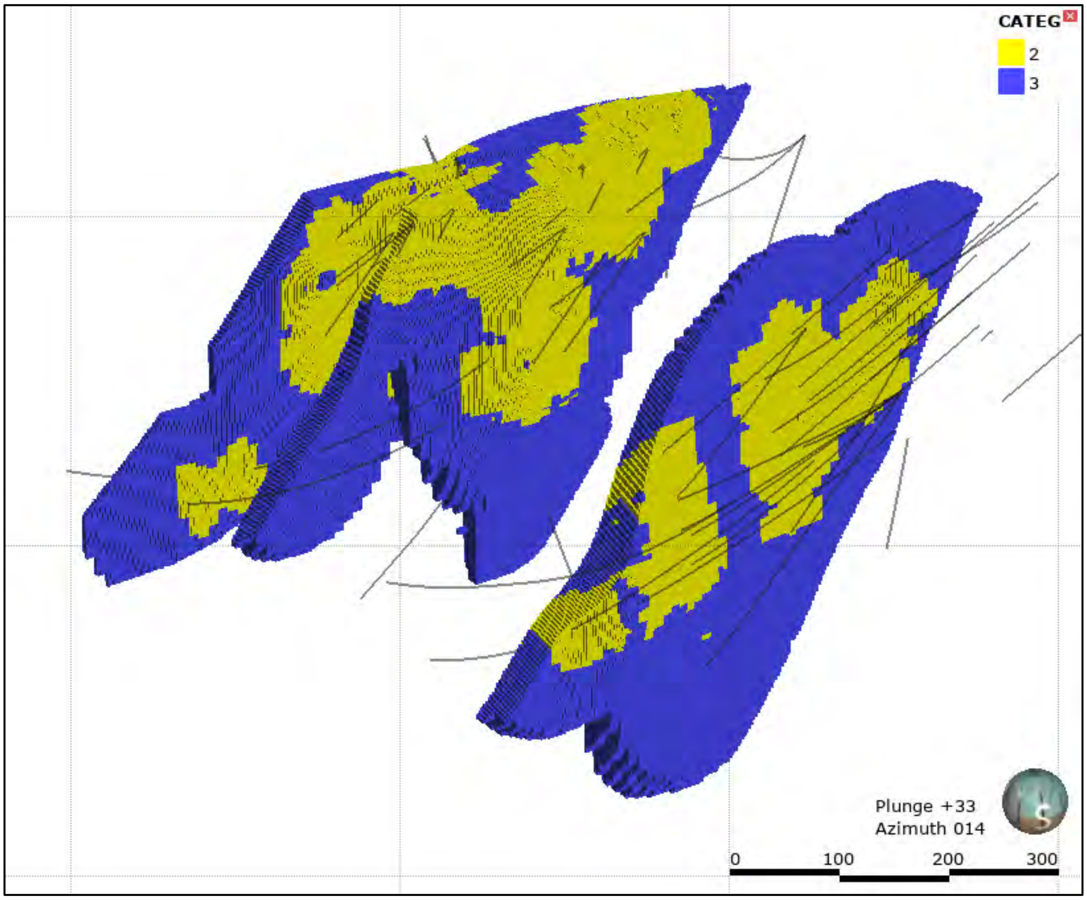
The Kailey block model was classified based largely upon estimation pass and distance to nearest composites. The resource categories are shown in Figure 14.49. There are no Measured Mineral Resources for the Kailey Deposit, with the relatively wide drill spacing being the limiting factor.

The Indicated Mineral Resource category was assigned to coherent portions of the deposit estimated in Pass 1 and Pass 2, with a distance to closest composite less than 40 m.

The Inferred Mineral Resource category was assigned to blocks estimated in Pass 1 and Pass 2 with a distance to closest composite greater than 40 m, and blocks estimated in Pass 3. In addition, all blocks outside the three principal domains and below the pit optimization were unclassified.

All blocks in Pass 4 were unclassified.

Figure 14.49: Kailey Deposit Coloured by Resource Category. Yellow = Indicated, Blue = Inferred



14.2.3.13 Mineral Resource Statement

The Mineral Resources are summarized in Table 14.48 at a lower cut-off grade of 0.4 g Au/t for the open pit category. The cut-off grades adopted offer the deposit reasonable prospects for eventual economic extraction using the assumptions summarized in Table 14.29.

The estimated Mineral Resources conform to the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves, as required by NI 43-101 – Standards of Disclosure for Mineral Projects.

Table 14.48: Summary of Kailey Mineral Resource

Deposit	Cut-off Category	Mineral Resource Category	Tonnes (million)	Gold Grade (g Au/t)	Gold Ounces (000's)
Kailey	Open Pit	Measured (M)	-	-	-
		Indicated (I)	11.276	0.96	348
		Subtotal M & I	**11.276**	**0.96**	**348**
		Inferred	4.858	0.87	136

Notes:

1. *Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability;*
2. *No mineral reserves are published at Kailey;*
3. *The independent and qualified person for the Kailey MRE is Mr. James Purchase, P.Geo of GMS.*
4. *The effective date of the Mineral Resource is September 3rd 2020*
5. *Open pit mineral resources are constrained within an optimized pit shell using a gold price of US$1,500, a CAD:USD exchange rate of 1.3 and a metallurgical recovery of 90%. An incremental ore haulage cost of $1.70 per t milled is also assumed for Kailey;*
6. *No underground Mineral Resources are quoted;*
7. *Mineral Resources are quoted at an open-pit lower cut-off of 0.4 g/t.*

Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing or other relevant issues.

14.2.3.14 Mineral Resource Sensitivity

The block model was reported at varying cut-offs to understand the sensitivity of the tonnes, grade and ounces to changes in the economic cut-off. The results are presented in Table 14.49.

Table 14.49: Kailey Open-Pit Mineral Resource Sensitivity

Cut-off Grade (g Au/t)	Indicated Category			Inferred Category		
	Tonnes (million)	Gold Grade (g Au/t)	Gold Ounces (000's)	Tonnes (million)	Gold Grade (g Au/t)	Gold Ounces (000's)
1.5	1.395	1.89	85	0.510	2.14	35
1.0	4.437	1.42	203	1.188	1.59	61
0.8	6.268	1.27	256	1.871	1.34	80
0.6	8.306	1.13	302	3.198	1.07	110
0.4	**11.276**	**0.96**	**348**	**4.858**	**0.87**	**136**
0.2	18.117	0.71	411	10.017	0.57	183

14.2.3.15 QP Commentary

Various improvements have been incorporated into the 2020 Kailey MRE due to a significant effort by GGM to advance the geological understanding of the deposit, and an improved confidence in the drilling database:

- An underground void model has been incorporated into the block model;
- The geological model has changed significantly to honour the ENE strike of the three main domains, compared to the single domain trending ESE that was previously used in the 2012 MRE by Micon International Ltd;
- An improved base of overburden surface was also used.

The grade of the M&I portion of the deposit has remained similar (0.96 g/t in 2020 vs.0.95 g/t in 2012), however the ounces have increased (348 KOz in 2020 vs. 265 KOz in 2012) due to a deeper pit optimization used to constrain the resource. The new, more discrete geological interpretation has also likely increased the overall grade of the deposit, by reducing the mixing of different data populations.

GMS has not reported an underground resource at Kailey as there is insufficient drilling at depth to model a high-grade internal domain within the three low-grade domains. In addition, GMS believes the drill spacing is too wide to support Measured Resources.

There remains significant upside to expand mineral resources inside the current pit shell, as the No. 9 domain remains open up-dip and undrilled towards the surface. This represents an excellent exploration target. In addition, a denser drilling pattern would likely enable the modelling of discrete, high-grade zones which could result in the declaration of an underground mineral resource. Discrete veins were mined at Little Long Lac in the past at an average grade of ≈ 10 g Au/t, and there is good potential to discover new high-grade veins in the axial planes of newly identified folds.

15. MINERAL RESERVE ESTIMATES

15.1 Summary

The Mineral Reserve for the Hardrock Project is estimated at 135.3 Mt an average grade of 1.27 g Au/t for 5.54 M ounces of gold as summarized in Table 15.1. The Mineral Reserve Estimate ("MRE") was prepared by G Mining Services Inc. ("GMS"). The resource block model was also generated by GMS and is dated June 2019.

The mine design and MRE have been completed to a level appropriate for feasibility studies. The MRE 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 Mineral Resources, and do not include any Inferred Mineral Resources. There is only 4% of measured mineral resources in the reserve, but this ore is located in the first phases and will be mined in the first five years of the mine life. The Inferred Mineral Resources contained within the mine design are classified as waste.

Table 15.1: Hardrock Open Pit Mineral Reserve Estimate

Category	Diluted Ore Tonnage (kt)	Gold Grade (g Au/t)	Contained Gold (koz Au)
Proven	5,623	1.28	232
Probable	129,700	1.27	5,307
Total P&P	**135,323**	**1.27**	**5,539**

Notes:

1. *CIM definitions were followed for Mineral Reserves;*
2. *Effective date of the estimate is August 8, 2019;*
3. *Mineral Reserves are estimated at a cut-off grade of 0.35 g Au/t;*
4. *Mineral Reserves are estimated using a long-term gold price of USD 1,250/oz and an exchange rate of CAD/USD 1.30;*
5. *A minimum mining width of 5 m was used;*
6. *Bulk density of ore is variable but averages 2.78 t/m^{3};*
7. *The average strip ratio is 5.10:1;*
8. *Dilution factor is 17.2%;*
9. *Numbers may not add due to rounding.*

15.2 Resource Block Model

The block model consists of six folders with block percent attributes for intact rock mass, overburden, organics, tailings, historical underground openings and backfill. The historical underground openings have been modelled and depleted in the block model with backfill densities assigned for stopes backfilled with sand or rock. Some tailings overlay the pit footprint and have been modelled to allow for their tracking and management in the material movement plan.

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 Report, Measured and Indicated Mineral Resource blocks were considered for optimization purposes and for mineable resource calculations. However, sensitivities were run using the complete resource.

15.3.1 Pit Slope Geotechnical Assessment

Golder was mandated to produce a feasibility level pit slope design study to support the mine designs. The conclusions of this study have been used as an input to the pit optimization and design process.

The Golder scope included reviewing geotechnical field investigations carried out by mine design engineering, carrying out follow-up field investigations and providing feasibility level slope designs for the open pit.

It has been assessed that the open pit will be developed in a good to very good rock mass where rock mass failure is not a concern. Historical underground long wall mining has proven the quality of the rock mass. The mineralization is found in upright sub-vertical axial planes that trend roughly east-west. The fold axes are shallowly west-plunging.

Potential instability will involve structural controls, the most significant being the foliation control on the bench face angle and the potential control of flat sets on the bench crest back-break angles. No major faults have been identified that will adversely daylight on the final pit walls. The locations of the underground workings and whether they are filled or unfilled are well understood. Risks to safety and pit access due to these stopes must be mitigated through design and planning at all stages of the Project.

While there are localized differences in the orientations of the discontinuity populations, they do not justify distinctly different slope designs. The slope configuration options are presented in Table 15.2. Double benching will have to be done with vertical pre-split, no sub-grade drilling and well controlled blasting practices are required.

The final pit was designed using a double benching configuration to a final height of 20 m. The pit slope profile is based on recommendations by Golder as presented in Table 15.2. The slope profile is based on vertical batter angles with a 10 m catch bench width for an inter-ramp angle of 63.4 degrees. A 16 m geotechnical berm is introduced every 100 m, where ramp segments do not pass in the slope to reduce the vertical stack height.

At the bedrock-overburden contact, the overburden is sloped at a 2H:1V angle. The overburden slopes will be comprised of fluvial or glacial cohesionless or cohesive material of sufficient strength. On the east side of the pit the overburden thickness averages 15 m with a maximum depth of 25 m. On the north side, the average depth is approximately 10 m with a maximum of 30 m when including the historical MacLeod Mine tailings.

As reported by Golder, the rock mass is assumed to have a very low permeability. It is also unknown at which rate the historical underground workings were filled with water. The water table is observed to be close to surface in fenced glory holes. For slope stability assessments, it has been assumed that slopes will be partially saturated with drawdown cones similar to another open pit in the region.

Table 15.2: Hardrock Final Wall Geotechnical Recommendations

Slope Parameters	
Final Bench Height (m)	20.0
Bench Face Angle (º)	90
Avg. Design Catch Bench Width (m)	10.0
Inter-ramp Angle (º)	63.4
Overall Slope Angle (º)	60.8
Geotechnical benches (m)	16.0[1]

Note 1: Geotechnical Catch berm will be of 16.0 m at every 100 m

15.3.2 Mining Dilution and Ore Loss

A mining dilution assessment was made by evaluating the number of contacts for blocks above an economic cut-off grade (“CoG”). The block contacts are then used to estimate a dilution skin around ore blocks to estimate an expected dilution during mining. The dilution skin consists of 0.75 m of material in a north-south direction (across strike) and 1.0 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 diluted grades (Au, As and S) and a new density are calculated by taking into account the in-situ grades and in-situ density of the surrounding blocks.

15.3.3 Pit Optimization Parameters

A summary of the pit optimization parameters is presented in Table 15.3 for a milling rate of 27 kt/d based on a long-term gold price of USD 1,250/oz and an exchange rate of CAD/USD 1.30.

The gold selling cost includes a 3% royalty fee plus a transport and refining cost of CAD 3.00/oz. The cost parameters were estimated based on first principles. The total ore-based cost is estimated at CAD 10.45/t which includes processing, general and administration costs and a sustaining capital provision.

Unit reference mining costs are used for a “reference mining block” usually located near the pit crest or surface and are incremented with depth which corresponds to the additional cycle time and thus hauling cost. The reference mining cost is estimated at CAD 2.05/t with an incremental depth factor of CAD 0.033/t per 10 m bench.

A physical hard boundary was imposed in the optimization process to prevent the pit from encroaching into nearby lakes (Figure 15.1). The hard boundary was established to maintain a 30 m buffer zone between the pit and the lake high water limit which corresponds to the 330 m level contour.

Figure 15.1: Pit Limit Hard Boundary Constraint

Mining Services	**Pit Limit Hard Boundary**	**Author: LP Gignac** **Date: Aug 2019**

The overall slope angles utilized in Whittle are based on the inter-ramp angles recommended in the Golder pit slope study with provisions for ramps and geotechnical berms. The overall slope angle in competent rock is 55 degrees based on a designed inter-ramp angle of 63.4 degrees. The overall slope angle in overburden is 26 degrees.

15.3.4 Cut-Off Grades

The cut-off grade resulting from the optimization parameters is estimated at 0.27 g Au/t which assumes an average metallurgical recovery of 91.1% and an average mining dilution of 17%. In order to increase the average gold grade, a raised COG of 0.35g/t was used to calculate the reserve.

The cut-off grade is the breakeven grade where revenue equals costs to carry the full operation while excluding direct mining costs:

$$COG(g/t) = \frac{Cp + Ca + Cr + Com + Csibc + Cmc}{r * (P - Cs)}$$

Where:

r is the metallurgical recovery (%)

P is the gold price in CAD/oz

Cs is the cost of selling gold (refining and royalties) in CAD/oz

Cp is the total Processing Costs (Fixed and Variable) in CAD/t treated

Ca is Administration and General cost in CAD/t treated

Cr is the cost of rehandle in CAD/t treated

Com is the difference between ore and waste mining cost in CAD/t treated

Csibc is Non-mining sustaining capital in CAD/t treated over life of mine

Cmc is Mine Closure cost incurred during the life of mine in CAD/t treated

Table 15.3: Optimization Parameters

Hardrock Pit Optimization Parameters		
Nominal Milling Rate	t/d	27,000
Plant Throughput	kt/y	9,855
Exchange Rate	CAD/USD	1.30
Diesel Fuel Price Delivered	CAD/litre	0.90
Electricity Cost	CAD/kWh	0.059
Gold Price	USD/oz	1250
Gold Price (local currency)	CAD/oz	1625
Transport and Refining Cost	CAD/oz	3.00
Royalty Rate	%	3.0%
Metallurgical Recovery at Cut-Off Grade	%	91.1%
Total Processing Cost	CAD/t milled	7.54
General and Administration	CAD/t milled	1.59
Rehabilitation and Closure	CAD/t milled	0.38
Sustaining Capital	CAD/t milled	0.70
Total Ore-based Cost	CAD/t milled	**10.45**
Marginal Cut-Off Grade Raised Cut-Off Grade	g Au/t g Au/t	0.265 0.35
Mining Rate	kt/y	70,000
Mining Dilution	%	17.0%
Mining Loss	%	1.2%
Total Mining Reference Cost	CAD/t mined	**2.05**
Incr. Bench Cost (CAD /10 m bench)	CAD/10 m bench	0.033
Overall Slope Angle in Fresh Rock	degrees	55
Overall Slope Angle in Overburden	degrees	26

15.3.5 Open Pit Optimization Results

The Whittle nested shell results are presented in Table 15.4 using only the Measured and Indicated Mineral Resource. The nested shells are generated by using revenue factors to scale up and down from the base case selling price.

Table 15.4: Measured and Indicated Mineral Resource Whittle Shell Results

Pit Shell	Best Case Disc. @ 5% (M$)	Specified Disc. @ 5% (M$)	Worst Case Disc. @ 5% (M$)	Total Tonnage (kt)	Ore Tonnage (kt)	Strip Ratio	Grade g Au/t
10	3,199	3,117	2,942	593,235	116,798	4.08	1.30
11	3,259	3,166	2,977	621,119	121,432	4.11	1.29
12	3,270	3,173	2,982	626,180	122,581	4.11	1.29
13	3,277	3,178	2,985	630,348	123,355	4.11	1.28
14	3,293	3,187	2,991	640,308	125,015	4.12	1.28
15	3,317	3,198	2,997	655,711	127,565	4.14	1.27
16	3,350	3,219	3,002	683,944	130,689	4.23	1.27
17	3,360	3,223	3,003	692,021	131,905	4.25	1.27
18	3,381	3,228	3,000	712,466	134,714	4.29	1.26
19	3,406	3,236	2,994	737,749	138,419	4.33	1.26
20	3,412	3,238	2,991	745,490	139,313	4.35	1.25
21	3,421	3,238	2,983	758,332	140,868	4.38	1.25
22	3,427	3,240	2,981	766,193	142,134	4.39	1.25
23	3,446	3,233	2,953	794,725	146,804	4.41	1.23
24	3,450	3,231	2,946	801,438	147,994	4.42	1.23
25	3,456	3,226	2,930	814,468	149,772	4.44	1.22
26	3,491	3,211	2,852	909,270	157,763	4.76	1.22
27	3,501	3,192	2,803	945,744	160,708	4.88	1.22
28	3,502	3,192	2,801	949,428	161,296	4.9	1.21
29	3,503	3,190	2,796	952,803	161,910	4.9	1.21
30	3,506	3,180	2,771	969,850	163,342	4.9	1.21
31	3,507	3,180	2,770	971,143	163,563	4.9	1.21
32	3,514	3,132	2,671	1,052,309	169,038	5.2	1.21
33	3,518	3,098	2,583	1,111,491	174,130	5.4	1.20
34	3,521	3,030	2,440	1,236,833	179,046	5.9	1.22
35	3,521	3,028	2,435	1,240,699	179,739	5.9	1.22
36	3,521	3,026	2,431	1,243,743	180,175	5.9	1.22
37	3,520	3,023	2,425	1,248,086	180,629	5.9	1.21
38	3,519	3,017	2,411	1,256,373	181,601	5.9	1.21
39	3,519	3,016	2,409	1,258,227	181,889	5.9	1.21
40	3,519	3,015	2,407	1,259,381	182,036	5.9	1.21
41	3,518	3,010	2,399	1,265,471	182,517	5.9	1.21
42	3,518	3,009	2,397	1,266,910	182,605	5.9	1.21
43	3,472	2,771	1,983	1,549,486	191,160	7.1	1.24
44	3,472	2,768	1,978	1,552,853	191,412	7.1	1.24
45	3,465	2,735	1,923	1,592,663	192,790	7.3	1.25

The shell selection is presented in Table 15.5. Pit shell 22 is selected as the optimum final pit shell which corresponds to a USD 925/oz pit shell (Revenue Factor 0.74) which is a conservative selection that minimizes the strip ratio. This shell has a total tonnage of 766 Mt including 142.1 Mt of ore at an average grade of 1.25 g Au/t for 5.70 M in-situ ounces of gold. The average strip ratio is 4.4:1. This is the smallest shell that achieves close to maximum value using a practical phasing approach.

Figure 15.2: Pit by Pit Graph M&I Resource

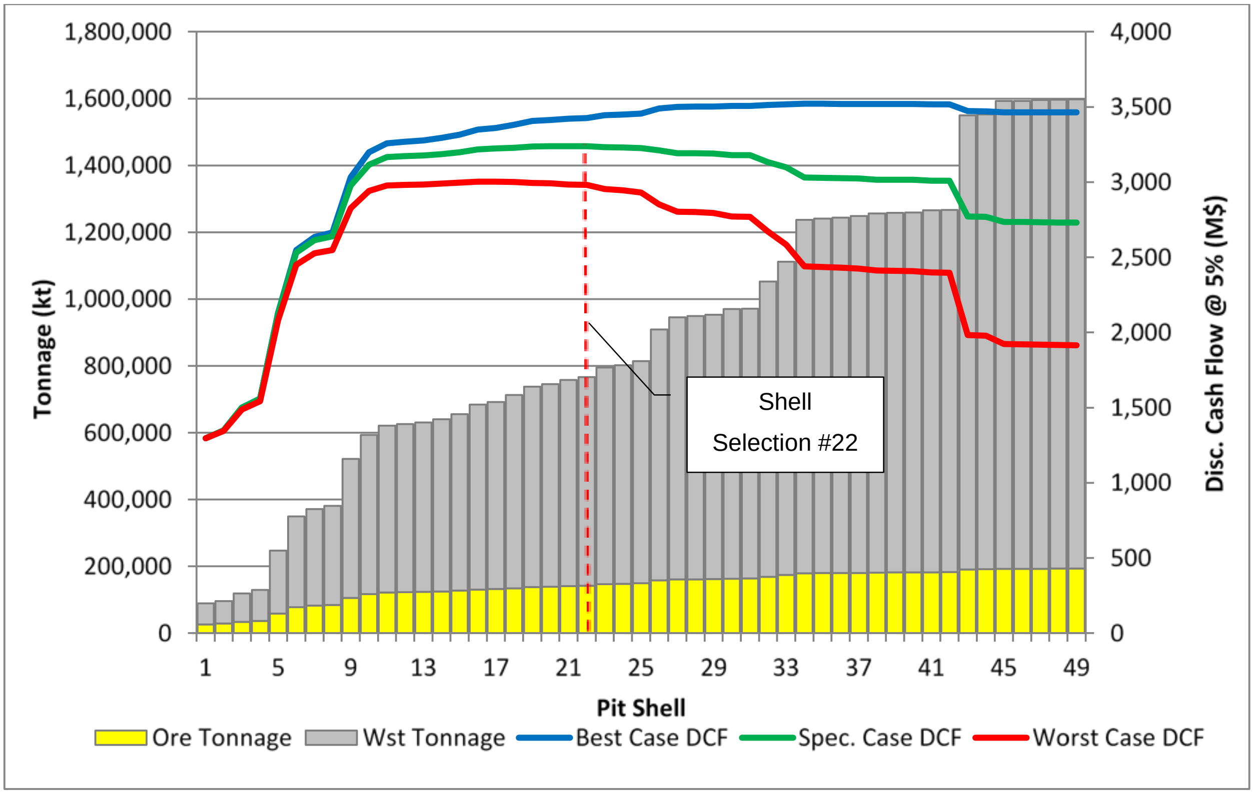


Table 15.5: Measured and Indicated Mineral Resource Pit Shell Selection

Shell Selection	
Shell Number	22
Shell RF	0.74
Shell Price (USD/oz)	925
Total Tonnage (kt)	766,193
Waste Tonnage (kt)	624,059
Strip Ratio (W:O)	4.39
Ore Tonnage (kt)	142,134
Grade (g Au/t)	1.25
In-situ Gold (koz)	5,703

15.4 Mine Design

15.4.1 Underground Voids

The presence of underground stopes was considered when designing the pits mainly for the void in the F-Zone which is 150 m high and 30 m wide. Most of the other underground openings are backfilled with sand fill or rock fill.

Three permanent accesses have been maintained in the West Wall in order to have accesses where the F-Zone stope intersects the pit wall. An access is established above the stope (level 50), near the middle (level -30) and below at the bottom of the pit, in order to do wall maintenance.

The C-Zone underground stope, which is backfilled, was considered by avoiding intersecting it with the final ramp. To achieve this, the ramp circles around the opening at the 240 and 130 levels.

15.4.2 Ramp Design Criteria

The ramps and haul roads are designed for the largest equipment being a 250-t class haul truck with a canopy width of 8.65 m. For double lane traffic, industry best-practice is to design a travelling surface of at least three times the width of the largest vehicle. Ramp gradients are established 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 assure proper drainage of the running surface. The ditch will be 1.2 m wide. To facilitate drainage of the roadway a 2% cross slope on the ramp is planned.

The double lane ramp width is 32.0 m wide and the single lane ramp is 18.5 m wide. Single lane ramps are introduced in the pit bottom when the benches start narrowing and when the mining rates will be significantly reduced.

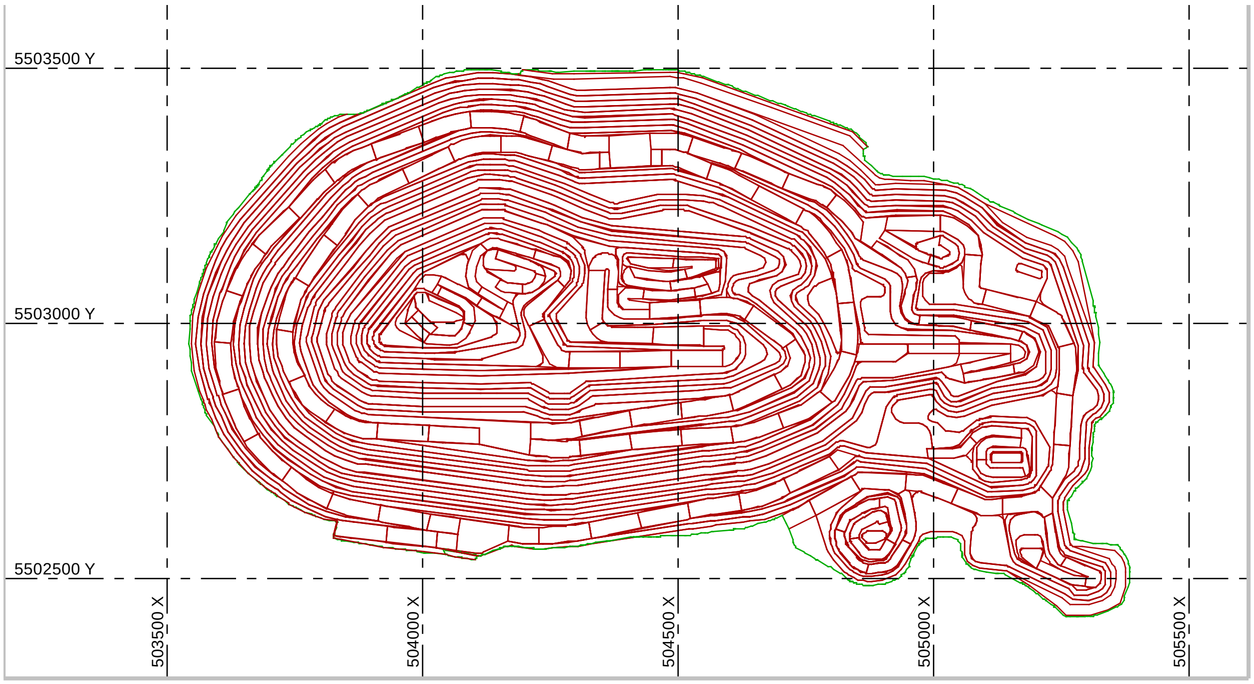
15.4.3 Open Pit Mine Design Results

The final pit design is presented in Figure 15.3. The final pit is 1,800 m along strike and 875 m wide and reaches a depth of 610 m. The final pit design has two exits; one to the east and one to the west, to provide access to the pushbacks and to shorten haul distances to the crusher and waste dumps. The west ramp system connects with the east ramp system at a plateau at a depth of 190 m. The ramp system introduces several switchbacks in several instances to avoid ramps passing through underground openings and to reduce the overall slope angle.

The west wall is steepest to access ore at depth. The ore is located mostly in the F-Zone. The pit is shallower on the east side as the mineralization plunges to the West.

There is a satellite pit to the east that is separated from the main pit. This satellite pit is 100 m deep and is limited to the east by the high-level water mark of the Kenogamisis Lake.

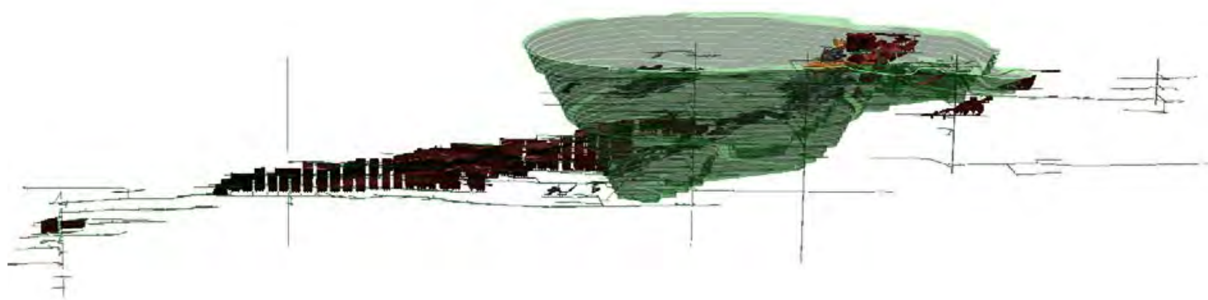
Figure 15.3: Final Pit Design



Mining Services	**Final Pit Design**	**Author: LP Gignac** **Date: Aug 2019**

A 3D view and longitudinal section is presented in Figure 15.4. Several of the underground voids will be entirely mined by the final open pit but certain voids will remain in the wall such as those from the F-Zone at depth.

Figure 15.4: 3D View of Final Open Pit with Historical Underground Voids



Mining Services	3D View of Final Pit with Historical Underground Voids	Author: LP Gignac Date: Aug 2019

15.5 Mineral Reserve Statement

The Mineral Reserve and stripping estimates are based on the final pit design presented in the previous section.

The Proven and Probable Mineral Reserves are inclusive of mining dilution and ore loss. The total ore tonnage before dilution and ore loss is estimated at 117.2 Mt at an average grade of 1.46 g Au/t for 5.49 Moz. Isolated ore blocks are treated as an ore loss and represent 1.72 Mt or 1.5% in terms of ore tonnage. The dilution envelope around the remaining ore blocks results in a dilution tonnage of 19.8 Mt at an average grade of 0.13 g Au/t for 86 koz. The dilution tonnage represents 17.2% of the ore tonnage before dilution and the dilution grade is estimated from the block model and corresponds to the average grade of the dilution skin. Table 15.6 presents a Resource to Reserve reconciliation.

Table 15.6: Resource to Reserve Reconciliation

Resource to Reserve Reconciliation	Tonnage (kt)	Grade (g Au/t)	Contained gold (koz)
Ore before Ore Loss and Dilution	117,201	1.46	5,490
Less: Ore loss (isolated blocks)	1,718	0.65	36
Ore before mining dilution	115,483	1.47	5,454
Add: Mining Dilution	19,840	0.13	86
Proven & Probable Mineral Reserve	135,323	1.27	5,539

The Proven Mineral Reserves total 5.6 Mt at an average grade of 1.28 g Au/t for 232 koz gold. The Probable Mineral Reserves total 129.7 Mt at an average grade of 1.27 g Au/t for 5,307 koz gold. The total Proven and Probable reserve is 135.3 Mt at an average grade of 1.27 g Au/t for 5,539 koz gold. The total tonnage to be mined is estimated at 824.9 Mt for an average strip ratio of 5.10 which includes overburden, historical tailings and underground backfill (Table 15.7).

Table 15.7: Hardrock Open Pit Mineral Reserves and Quantities

Final Pit Quantities		
Proven and Probable Mineral Reserve		
Proven Ore Tonnage	kt	5,623
Gold Grade	g Au/t	1.28
Contained Gold	koz	232
Probable Ore Tonnage	kt	129,700
Gold Grade	g Au/t	1.27
Contained Gold	koz	5,307
Proven & Probable Ore Tonnage	kt	135,323
Gold Grade	g Au/t	1.27
Contained Gold	koz	5,539
Waste Material (including Inferred)		
Overburden	kt	8,857
Historical Tailings	kt	3,164
UG Backfill	kt	1,641
Waste Tonnage	kt	675,935
Total Tonnage	**kt**	**824,920**

Notes:

1. *CIM definitions were followed for Mineral Reserves;*
2. *Effective date of the estimate is August 8, 2019;*
3. *Mineral Reserves are estimated at a cut-off grade of 0.35 g Au/t;*
4. *Mineral Reserves are estimated using a long-term gold price of USD 1,250/oz and an exchange rate of CAD/USD 1.30;*
5. *A minimum mining width of 5 m was used;*
6. *Bulk density of ore is variable but averages 2.78 t/m^{3};*
7. *The average strip ratio is 5.10:1;*
8. *Dilution factor is 17.2%;*
9. *Numbers may not add due to rounding.*

16. MINING METHODS

16.1 Introduction

The Project consists of developing an open pit that will mine through the historical underground workings of the MacLeod-Cockshutt and Hard Rock Mines. Furthermore, the proposed open pit location is bisected by the Trans-Canada Highway 11 and requires a new by-pass and the relocation of various surface infrastructures.

16.2 Mine Designs

16.2.1 Open Pit Phases

Mining of the Hardrock main pit will occur in five main phases preceded by a starter pit. The content of each mining phase is summarized in Table 16.1. The objective of pit phasing is to improve the economics of the Project by feeding the highest grade during the earlier years and/or delaying waste stripping until later years. The starter pit and Phase 1 are designed to initiate mining before the Trans-Canada Highway is relocated. With the mineralization plunging westward, the pit phases progressively expand to the west.

Table 16.1: Pit Phase Design Summary

Phase Design Content		Starter Pit	Phase 1	Phase 2	Phase 2.5	Phase 3	Phase 4	Phase 4.5	Phase 5	Phase 5.5	Total Pit
Total Tonnage	kt	22,225	40,781	37,706	205,006	239,977	208,172	4,044	59,219	7,789	**824,920**
Overburden	kt	1,270	579	1,057	1,187	1,430	716	-	1,864	547	**8,649**
Tailings	kt	-	-	-	2,888	246	30	-	1	-	**3,164**
UG Backfill	kt	27	90	61	728	171	361	-	-	-	**1,438**
Waste Rock	kt	17,193	28,973	32,094	161,067	208,146	172,499	2,925	46,459	6,578	**675,935**
Diluted Ore	kt	3,735	11,138	4,494	38,749	29,969	34,559	1,119	10,896	664	**135,323**
Diluted Grade	g Au/t	1.32	1.22	1.38	1.35	1.13	1.39	1.39	1.00	1.39	**1.27**
In-situ Gold	koz	159	437	199	1,677	1,089	1,549	50	349	30	**5,539**
Strip Ratio	W:O	5.0	2.7	7.4	4.3	7.0	5.0	2.6	4.4	10.7	**5.1**
% of Gold	%	2.9%	7.9%	3.6%	30.3%	19.7%	28.0%	0.9%	6.3%	0.5%	**100.0%**

The phase designs introduce different geotechnical slope profiles for temporary pit walls. The temporary wall slope profile allows for wider catch benches to allow for overbank hazard management on pit walls. Overbank hazard results from muck from one phase spilling down the slope of the previous pit phase, filling the catch benches. This creates an increased rockfall hazard for workers and equipment at the bottom of the previous pit phase. The temporary wall design allows the catch bench to be accessed to remove debris. The maximum double bench inter-ramp angle for temporary walls is 52 degrees. Table 16.2 presents the different slopes. Phase 2 is created with final pit wall parameters to mitigate extended stripping; special caution must be made on maintaining the slope and the work below.

Table 16.2: Pit Phase Design Criteria

Slope Parameters	Temporary Pit Walls	Final Pit Walls
Final Bench Height (m)	20.0	20.0
Bench Face Angle (º)	90	90
Avg. Design Catch Bench Width (m)	15.5	10.0
Inter-ramp Angle (º)	52.2	63.4
Overall Slope Angle (º)	50.1	60.8
Geotechnical Benches (m)	16.0[1]	16.0[1]

Note 1: Geotechnical catch berm will be 16.0 m at every 100 m

The starter pit phase is designed to avoid various surface constraints such as the Trans-Canada Highway 11. Mining during the pre-production period is concentrated in the starter pit which provides for a 50 m buffer with the Trans-Canada Highway 11. The highway will be relocated to the north during the initial construction period. This starter pit phase reduces risk with respect to the timing of the highway relocation.

The Phase 1 design continues the constraint of the 50 m buffer from the Trans-Canada Highway 11 but without the constraint of avoiding historical tailings. Phase 1 pit uses the same ramp from the starter pit and descends significantly deeper. On the west side of Phase 1 is a flattened bench. This area will act as a plateau and dumping grounds for the historical tailings. Moving the tailings will allow the mining to the north. This plateau will be preserved throughout Phase 2 until it is eventually properly removed.

Phase 2 is restricted by the historical high tailings of a buffer of 50 m. The Trans-Canada Highway 11 has been displaced by this point and is removed to allow mining underneath. The plateau is kept with an independent ramp as the material can be transported away in this phase. A ramp on the north wall is added to facilitate the future mining of Phase 2.5 without requiring an additional external ramp.

Phase 2.5 continues with the main ramp from Phase 2, utilizing the internal ramp to mine material not normally accessible. There are no immediate constraints limiting this design and it is driven by profitability.

Phase 3 begins a new temporary ramp for access to the ore body until the depth of the pit is more then Phase 2.5, that the haulage is shifted to the main haulage ramp from 2.5. This explains the seemingly abrupt end to the Phase 3 temporary ramp when the switch is made between ramps.

Phase 4 is the maximum depth of the pit. Phase 4.5 is an internal sub pit that is mined after Phase 4 is complete. This is to ensure that the main haulage lane always has a side wall during production and is less prone to geological faults and sloughing. Phase 4.5 also cuts the main haulage lane into a single lane in a part. This can only be done when the main haulage from the main pit is completed.

Phase 5 is the eastern extension of the pit. There is a new independent ramp that will service all the haulage from the phase. This ramp will connect to and replace the main haulage lane created in Phase 2. Phase 5.5 are two sub pits on the edge of the Phase 5 pit. There are no constraints on when they are mined and are driven by the optimized schedule.

Figure 16.1: Starter Pit Phase Design

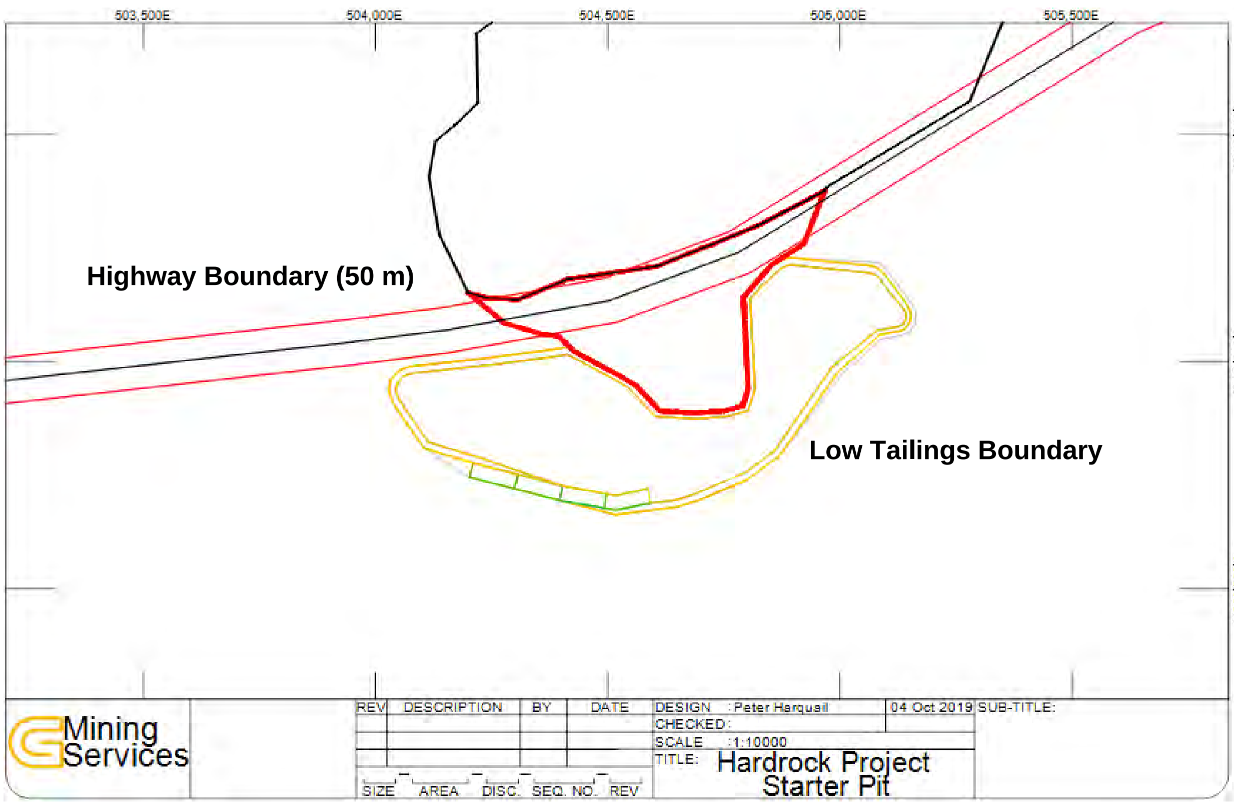


Figure 16.2: Phase 1 Design

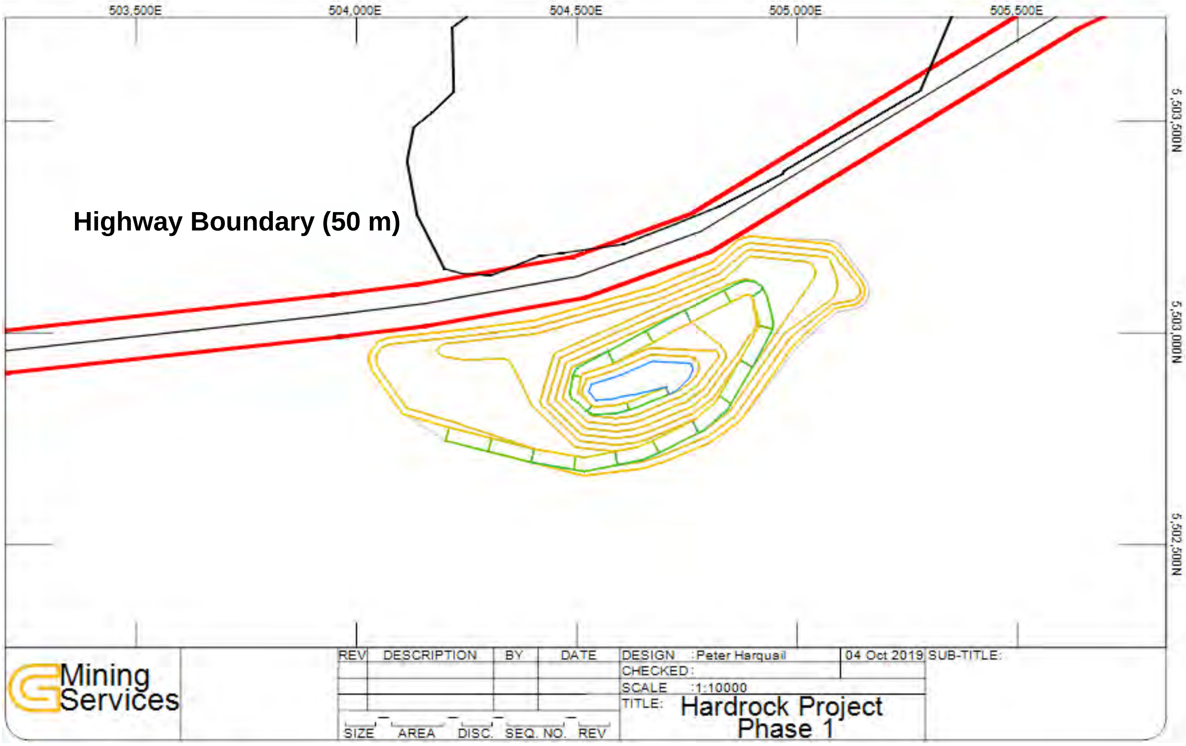


Figure 16.3: Phase 2 Design

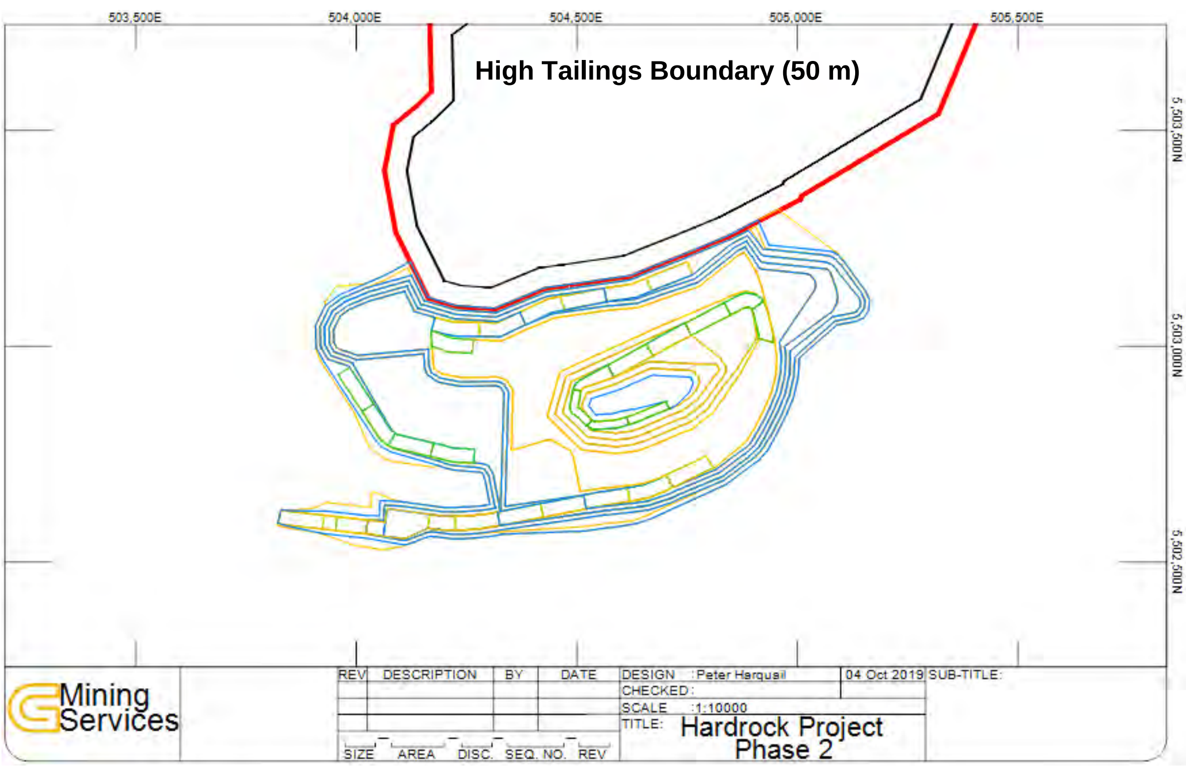


Figure 16.4: Phase 2.5 Design

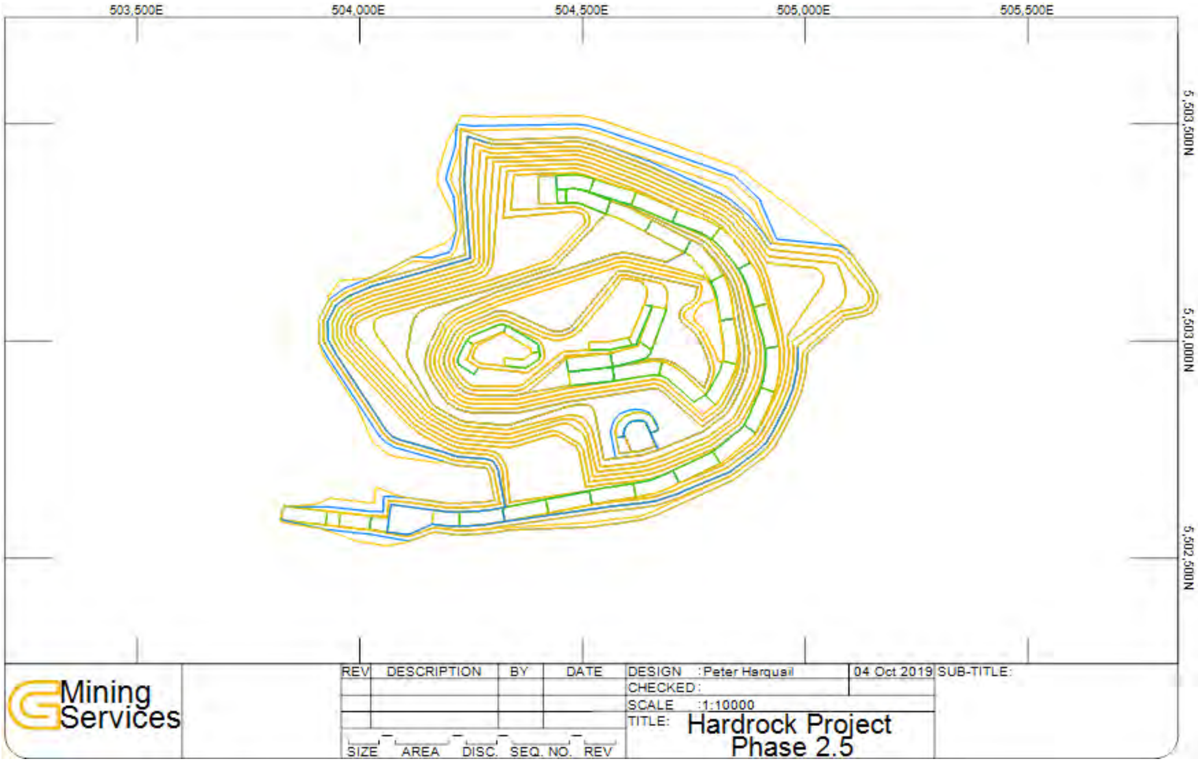


Figure 16.5: Phase 3 Design

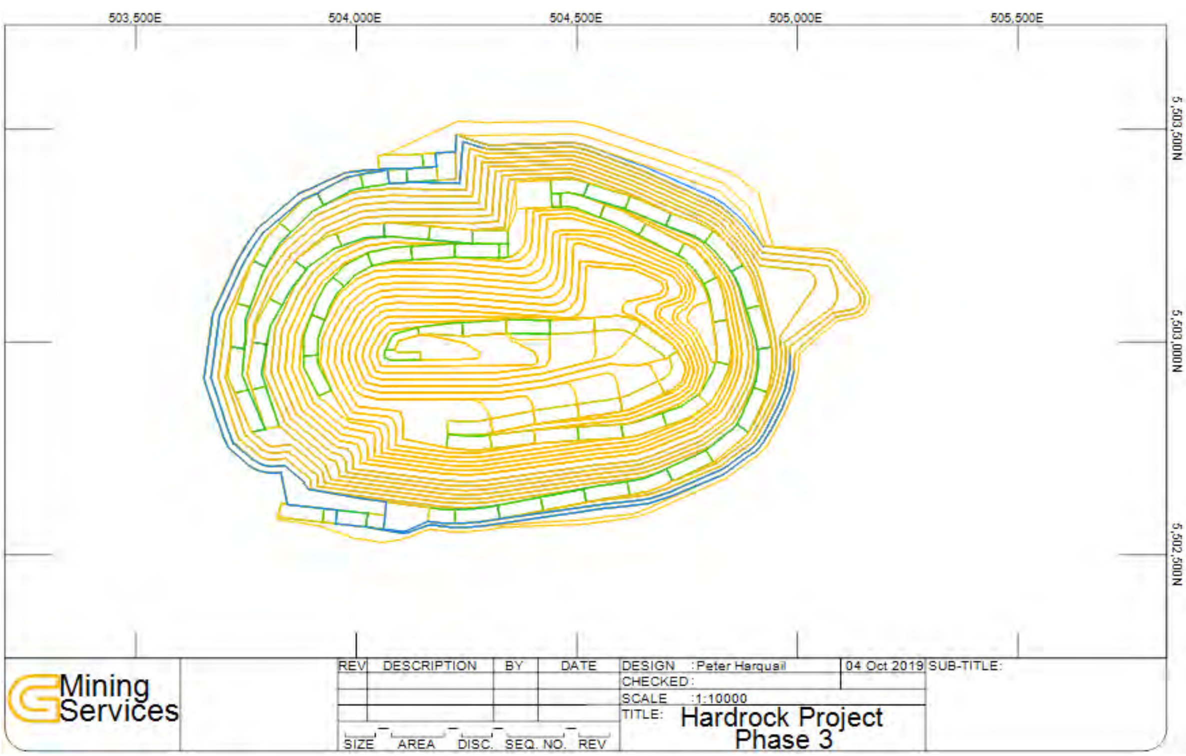


Figure 16.6: Phase 4 Design

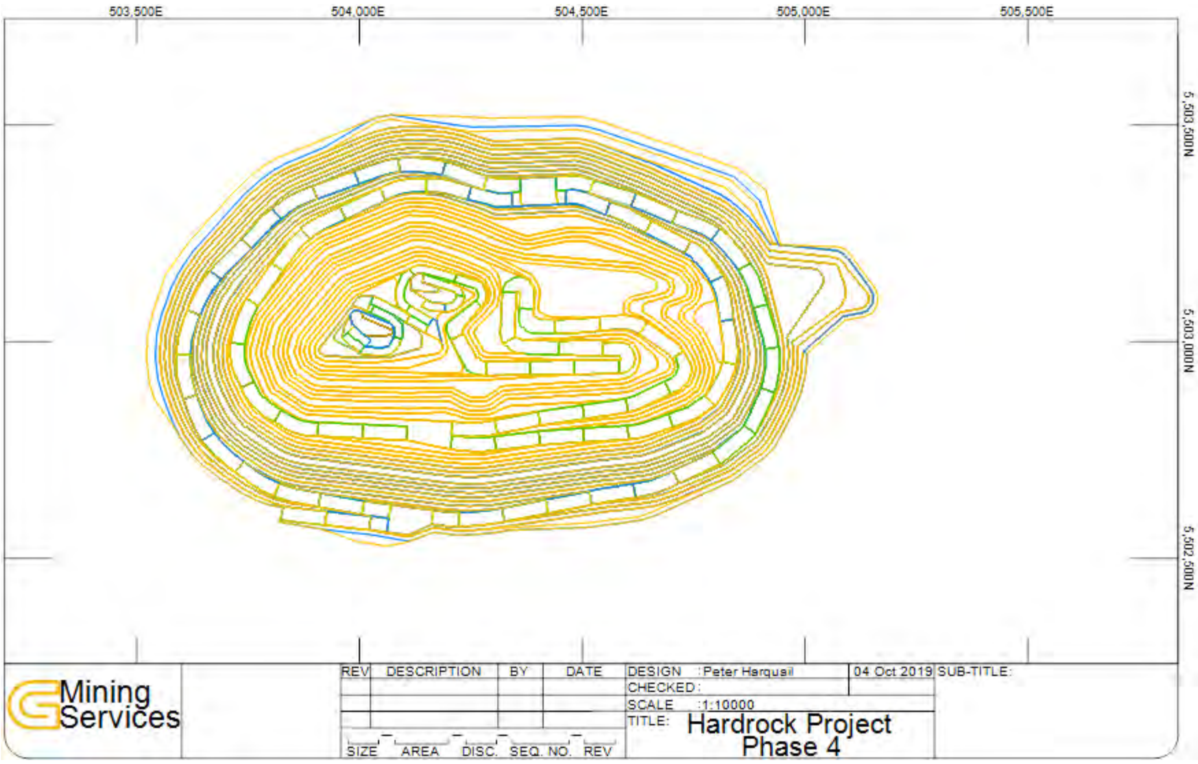


Figure 16.7: Phase 4.5 Design

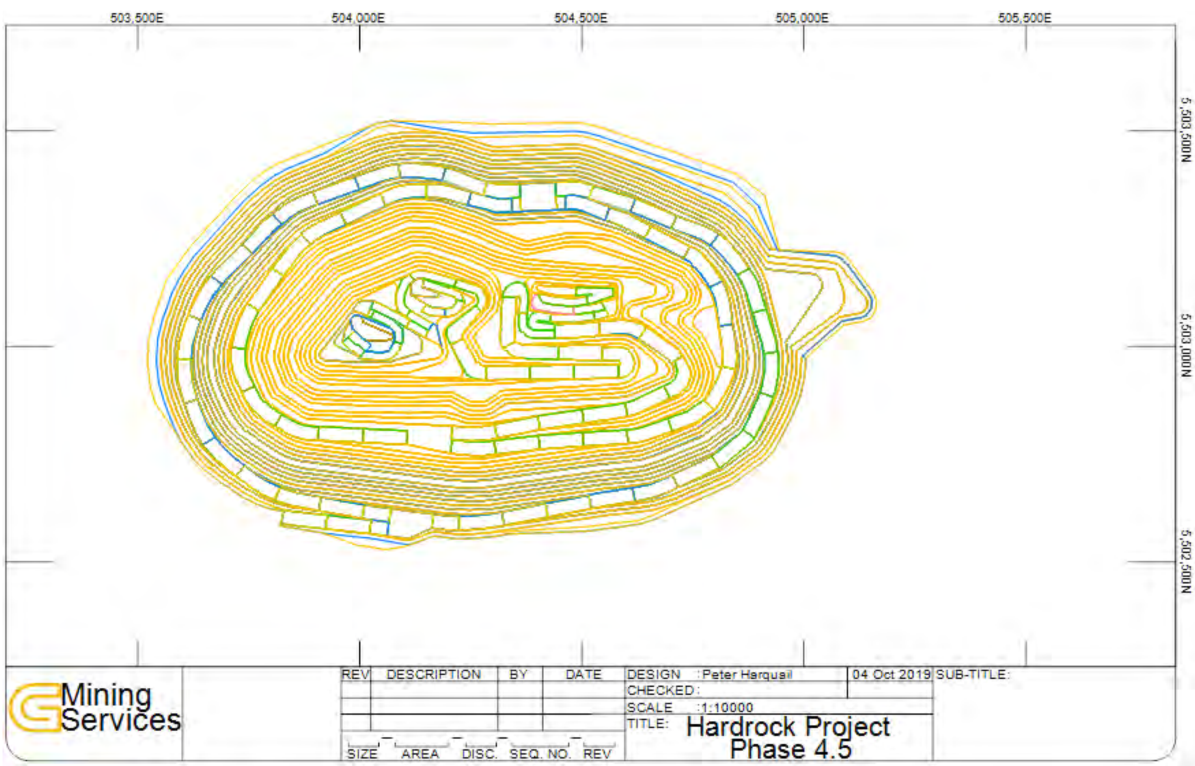


Figure 16.8: Phase 5 Design

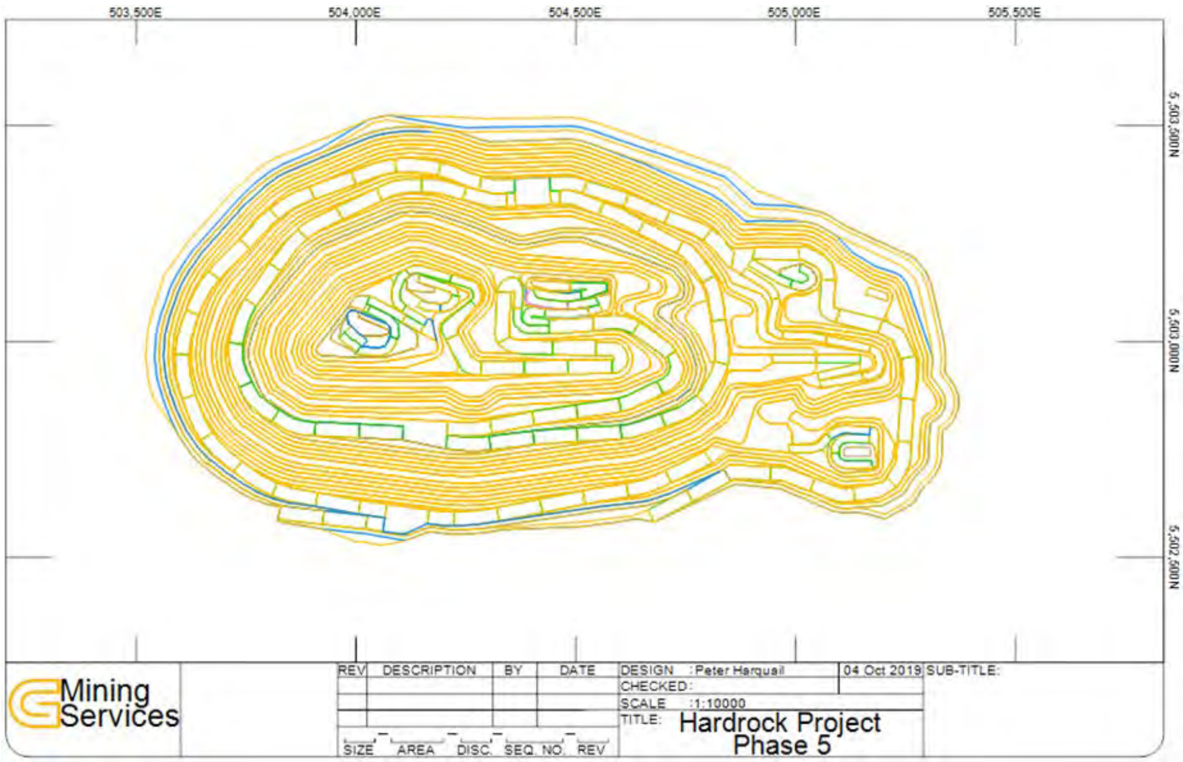
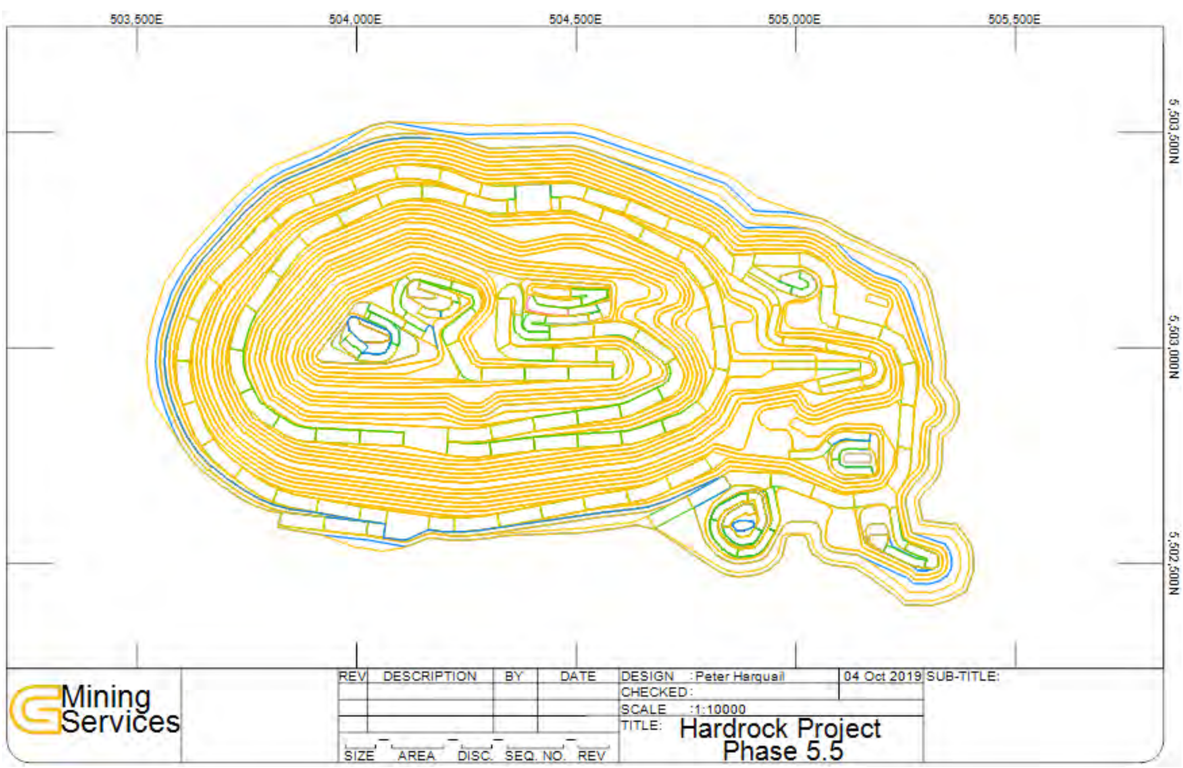


Figure 16.9: Phase 5.5 Design



16.2.2 Overburden and Waste Rock Storage

Waste rock will be disposed of in five distinct waste rock storage areas ("WRSA") of which four are located around the pit and one further to the south. The open pit generates 675.9 Mt of waste rock, 1.6 Mt of backfill, 3.16 Mt of historical tailings and 8.7 Mt of overburden that require storage. The tailings material will be transported for disposal within the Tailings Management Facility ("TMF").

The design criteria of each waste dump has been adjusted based on foundation stability assessments. It was recommended to use various waste dump design profiles which are shallower than the typical 2:1 in certain specific areas as presented in Table 16.4 to assure adequate safety factors. All rock waste dumps have 20 m high lifts to allow for wider catch benches to facilitate reclamation. Overburden dumps use 5 m high lifts to better control the material. All waste dump capacities are shown in Table 16.3.

Table 16.3: Waste Storage Capacities

Waste Dump	Capacity (Mt)	Capacity (Mm^3)	Surface Area (ha)	% Filled
Waste Dump A	47.1	22.7	51.2	100%
Waste Dump B	48.6	23.4	51.6	100%
Waste Dump C	135.9	65.4	111.2	100%
Waste Dump D	374.5	180.3	183.2	96.6%
In Pit Dumping + AB Extension	41.8	20.1	23.4	100%
Overburden Pile	12.6	7.9	55.7	70.2%
Total	**660.5**	**319.8**	**476.3**	**97.7%**

Note: This table does not include waste rock material that is planned for construction purposes (crusher ramp, roads, aggregates, TMF etc.).

Table 16.4: Waste Pile Design Criteria

Waste Dump	Avg. Catch Bench Width (m)	Pile Face Angle (deg)	Overall Slope Angle (H:V)	Maximum Elevation (m)	Approximate Height (m)
Waste Dump A	18.0	37	2:1 / 2.5:1/ 4:1	430	100
Waste Dump B	18.0	37	2:1 / 3:1 / 3.5:1	430	100
Waste Dump C	16.0	37	2:1 / 3:1	450	110
Waste Dump D	18.5	37	2:1 / 2.5:1/ 3:1	430	100
In Pit Dumping + AB Extension	14.0	37	2:1	390	300*
Overburden Pile	26.7	37	5:1	360	40

Note: (*) Height is from the bottom of the pit

16.2.3 Ore Stockpiles

The stockpile is designed with a maximum capacity of 12.9 Mt of ore. The total capacity is never reached throughout the mine life with the max stockpile material peaking at 9,652 kt in Year 12 ensuring a safety factor to account for changes in mine plan or stockpiling. There are four grade bins of stockpiled material ranging from marginal material to high grade. These bins are to be stored within the main stockpile in separate piles to reduce dilution.

Table 16.5: Ore Grade Bins Cut-off-Grades

Grade Bins	COG (g/t)
Bin 1	1.10
Bin 2	0.50
Bin 3	0.40
Bin 4	0.35

The stockpile pad has been designed to connect to the crusher pad, thus decreasing cycle time for ore re-handling. When the ore stockpile level is lower or higher than that of the crusher a temporary ramp of ore or waste is employed to ensure consistent and minimal rehandling times. The ore stockpile capacity is greater than required allowing for additional capacity in the future or the ore stockpile capacity could be reduced to add space for waste rock, if required.

The stockpile design criteria are presented in Table 16.6. The capacity found below is for one stockpile grade for all the ore. There is adequate space to have multiple grade stockpiles on the designated ore stockpile pad. The stockpile pad is created with 10 m lifts.

Table 16.6: Stockpile Design Criteria

Ore Stockpiles	Catch Bench Width (m)	Overall Slope Angle (H:V)	Maximum Elevation (m)	Approximate Height (m)	Max Capacity (Mt)
Stockpile Max Capacity	7.1	2:1	400	70	12.9

16.2.4 Mine Haul Roads

The haul roads, from the pit to the dumps, the crusher and the tailing storage facility, will be constructed mostly during the construction period. However, some haul roads will be constructed during operations as the pit evolves. Over the life-of-mine ("LOM") a total of 5.2 km mine haul roads will be constructed. In addition, the TMF access road is 3.8 km in length and will accommodate mine trucks. The TMF access road will circumvent the ore stockpile to prevent the haulage trucks from climbing and descending the crusher pad or passing through the processing plant.

16.3 Production Schedule

The mine production schedule is completed on a monthly basis during the pre-production period and first three months of commercial production. From the second half of the first year of operations to the end of Year 2, the schedule is developed on a quarterly basis and on an annual basis thereafter. The mine pre-production is initiated in Year -2 and transitions to commercial operations in Year 1 after commissioning and achieving a minimum of 60% of nameplate capacity for a period of 30 days. The mine pre-production period lasts a total of 20 months which is planned to allow for a gradual assembly and commissioning of mining equipment, training and timely delivery of waste rock for civil works.

The objectives of the LOM plan are to maximize discounted operating cash flow of the Project, subject to various constraints:

- Limit mining during pre-production;
- Supply best grade ore to the plant and feed to a nominal capacity which ultimately reaches 27,000 t/d (9.86 Mt/y);
- Limit the mining rate to approximately 70 Mt/y;
- Limit the vertical drop-down rate to approximately 8 benches, per phase, per year;
- Limit peak truck requirements;
- Utilize a grade segregation and stockpiling strategy with a maximum stockpile of 9.6 Mt which is roughly equivalent to one year of milling.

The mining schedule pre-production tonnage is 40.2 Mt over a period of 20 months. Mining will be conducted on day shift only for a period of four months and on two shifts by the 5th month. The peak mining rate of approximately 70 Mt is maintained for eight years (Year 2 to Year 9) and then gradually declines as either sufficient ore for the mill is available or to limit peak truck requirements. The annual mine production, stockpile inventory, mill production and gold production are presented in Figure 16.10 to Figure 16.13. Figure 16.14 to Figure 16.17, present the end of period mine infrastructure status at different dates.

The operating strategy is to mill at a finer grind size of P80 of 72 µm in the early months while reaching a rate of 24 kt/d and to increase the throughput to the targeted rate of 27 kt/d by relaxing the grind size to P80 of 90µm. During the early ramp-up phase, the plant will operate at a finer grind (72 µm) to achieve a higher recovery. These operating regimes are expected to impact the gold recovery. Metallurgical recovery equations were established for these two regimes which are also impacted by the gold head grade (Au in g Au/t), sulfur (S in %) and arsenic (As in %) levels. With these recovery equations two recovered gold grade attributes were estimated for the two grind sizes.

When calculating the metallurgical recovery, an adjustment is done on certain sulphur and arsenic grades based on the grade ratios:

- If the Au (g/t) / As (%) ratio is less than 8, then the As value to be used is: As (%) = Au (g/t) / 8;
- If the Au (g/t) / S (%) ratio is less than 0.276, then the S value to be used is: S (%) = Au (g/t) / 0.276.

The metallurgical recovery equations are as follows:

- Tails Grade (g Au/t) @ 72 µm = -0.0435 + 0.0349*(Head Grade g Au/t) + 0.660*(As) + 0.0312*(S) + 0.000516*72;
- Tails Grade (g Au/t) @ 90 µm = -0.0435 + 0.0349*(Head Grade g Au/t) + 0.660*(As) + 0.0312*(S) + 0.000516*90;
- Metallurgical Recovery = 1 – (Tails Grade / Head Grade);
- Recovered Gold Grade = Metallurgical Recovery x Head Grade (g Au/t).

Figure 16.10: Annual Mine Production

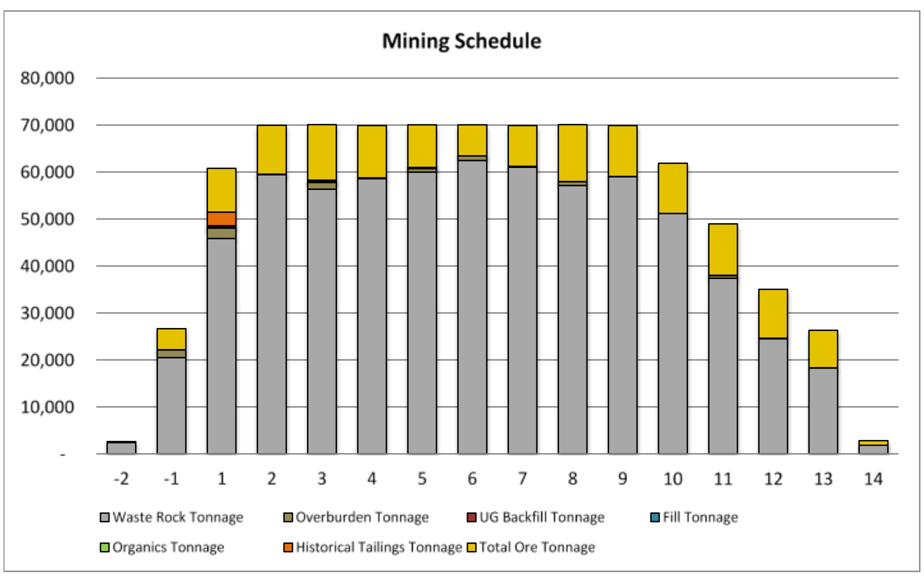


Figure 16.11: Annual Stockpile Inventory

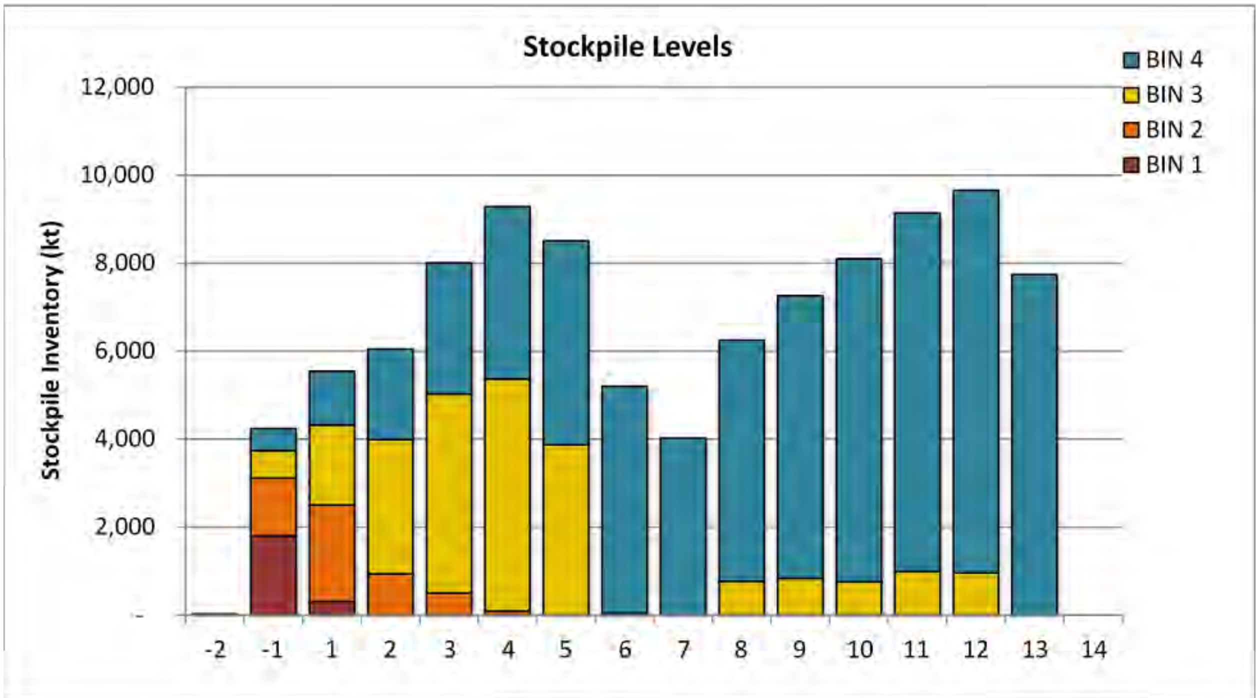


Figure 16.12: Annual Mill Production

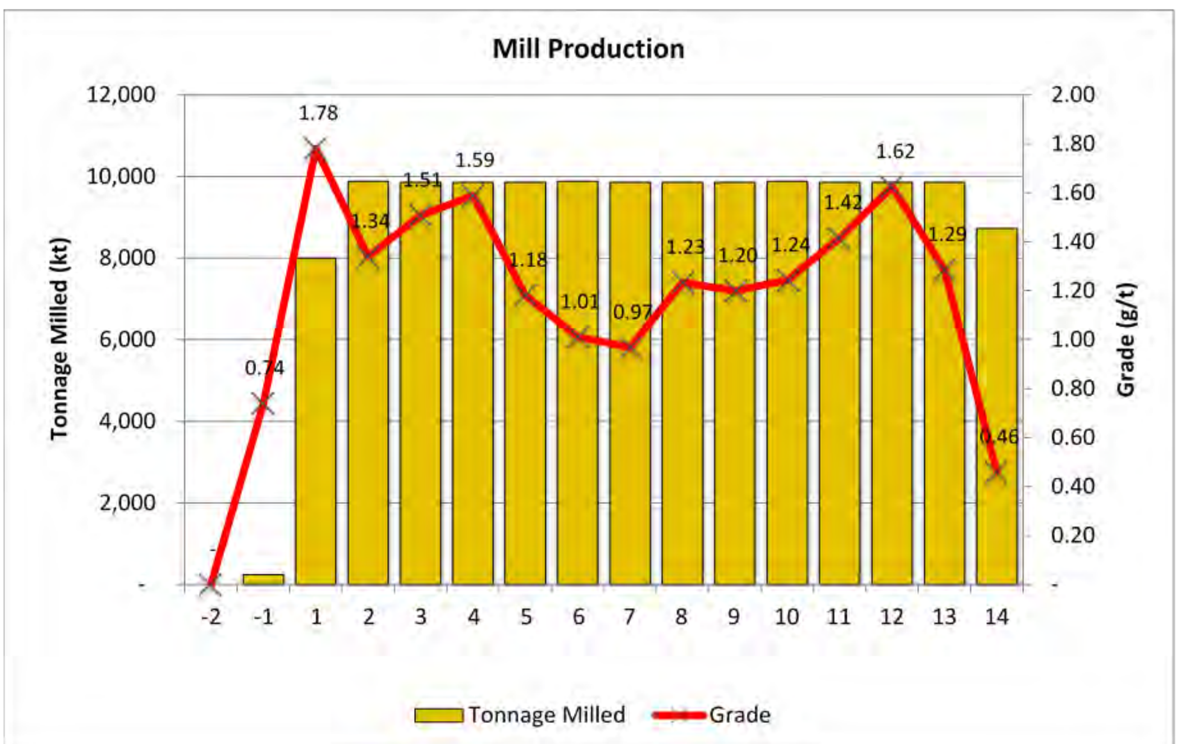


Figure 16.13: Annual Gold Production

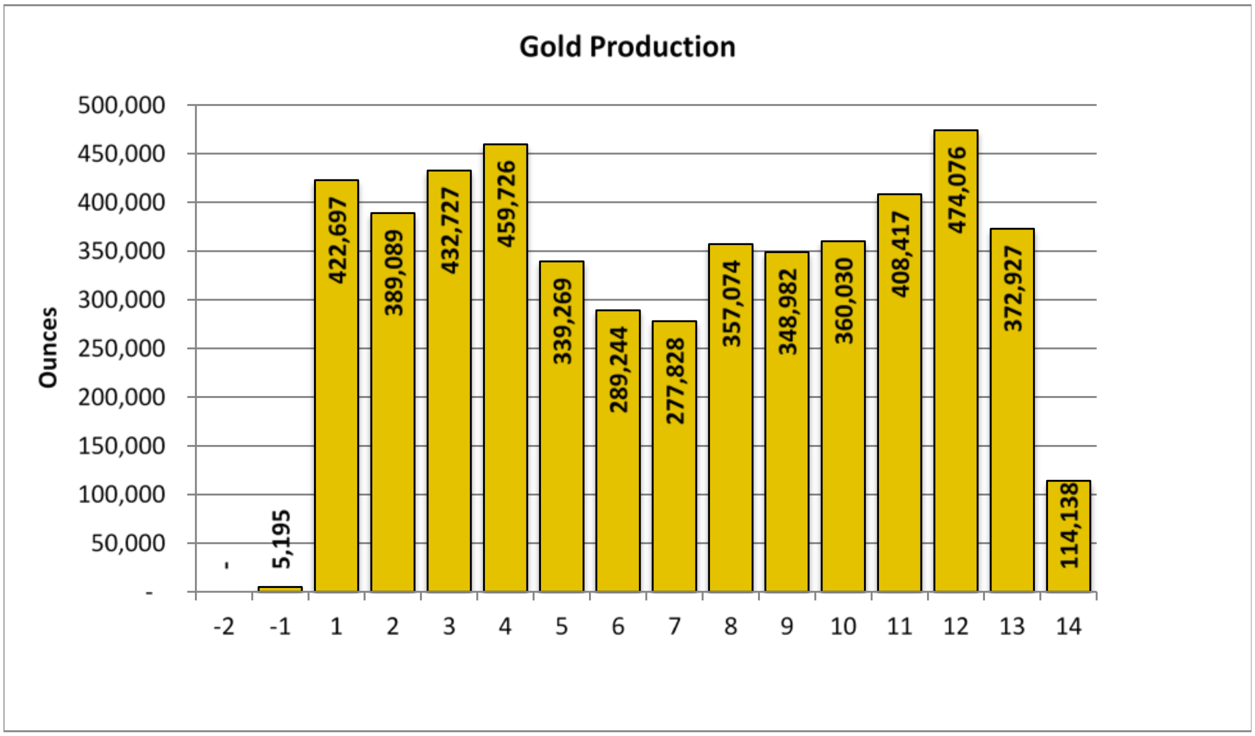
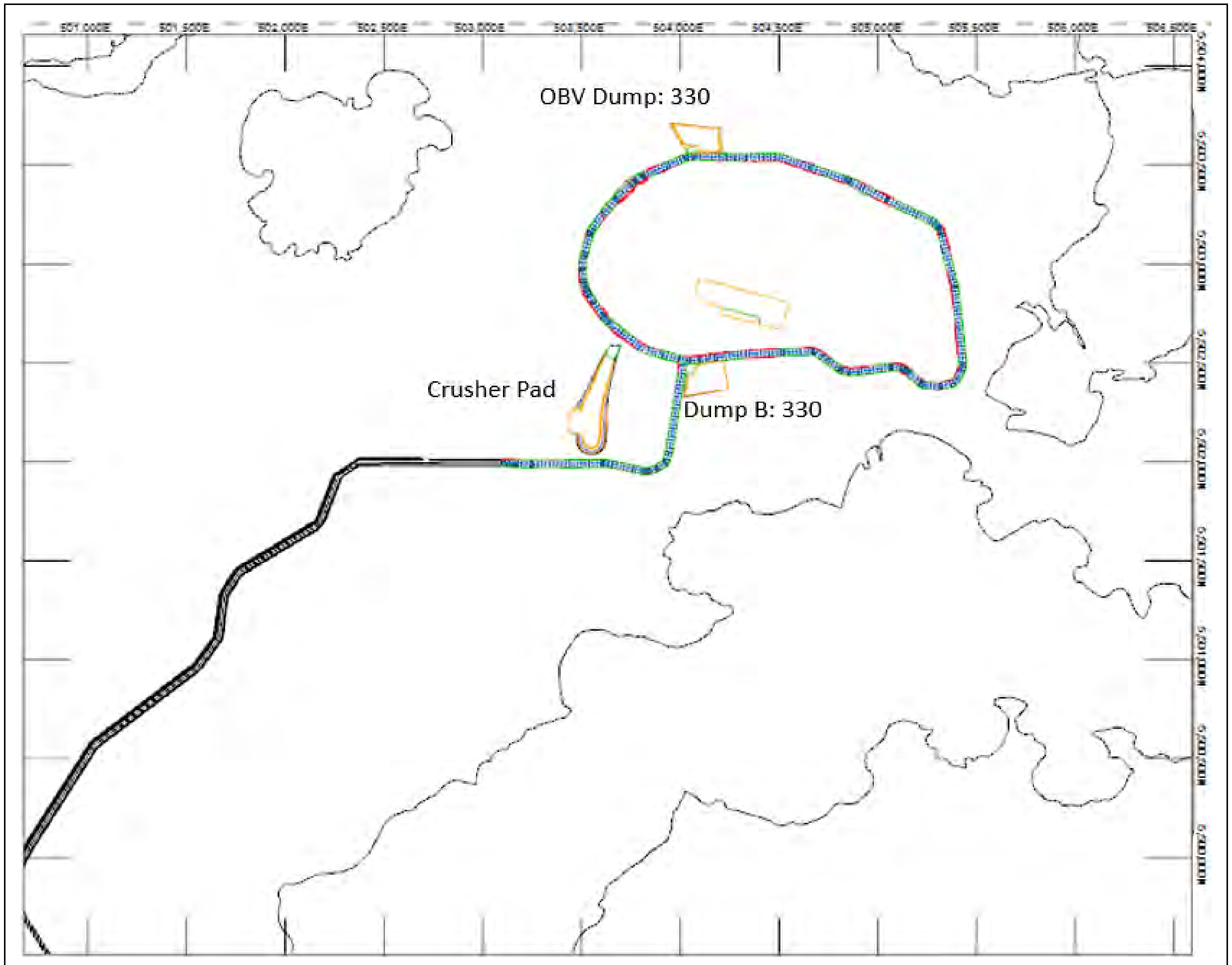


Figure 16.14: Production Schedule – Year -1

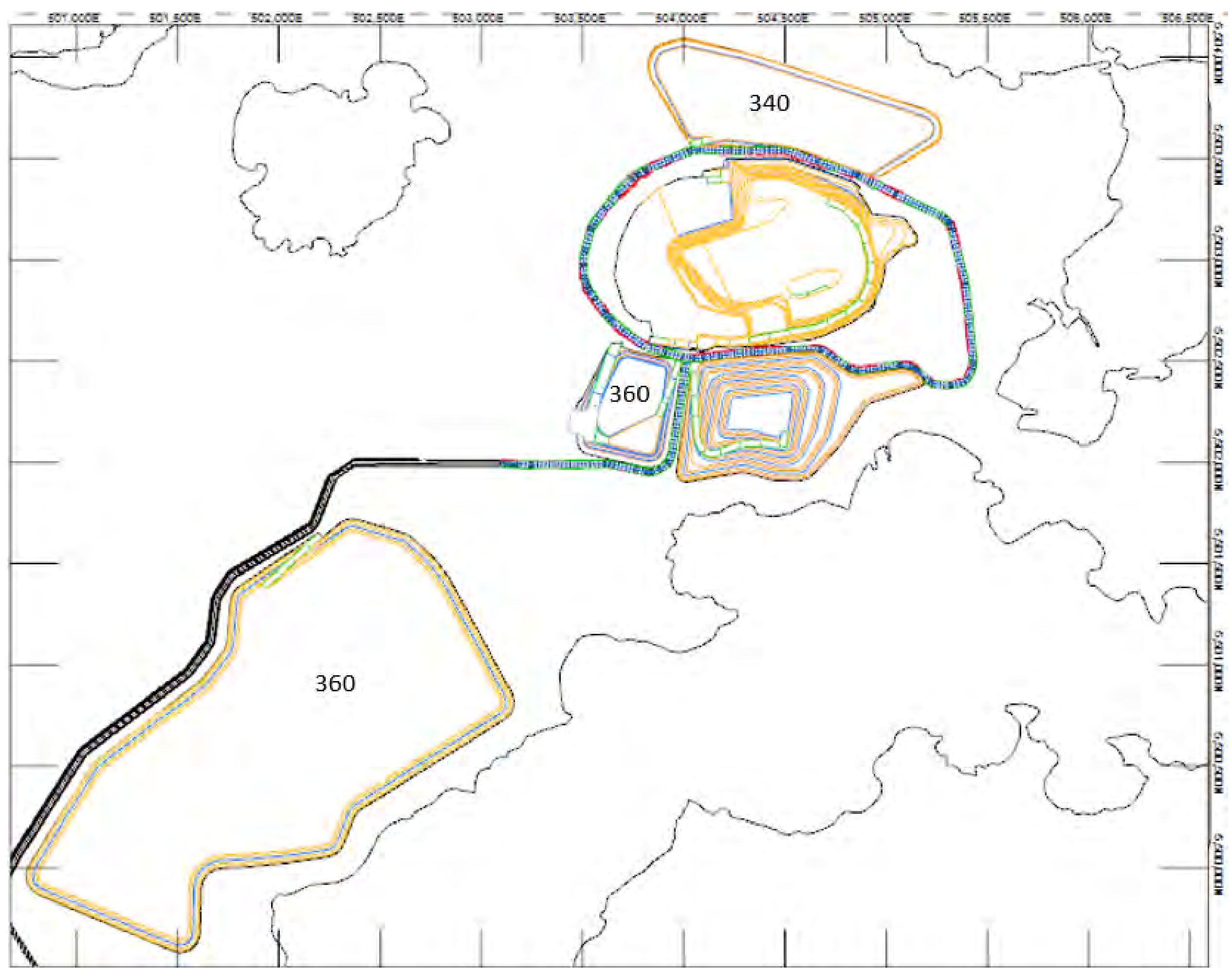


Mining Services

Pit Schedule – Year -1 - M12

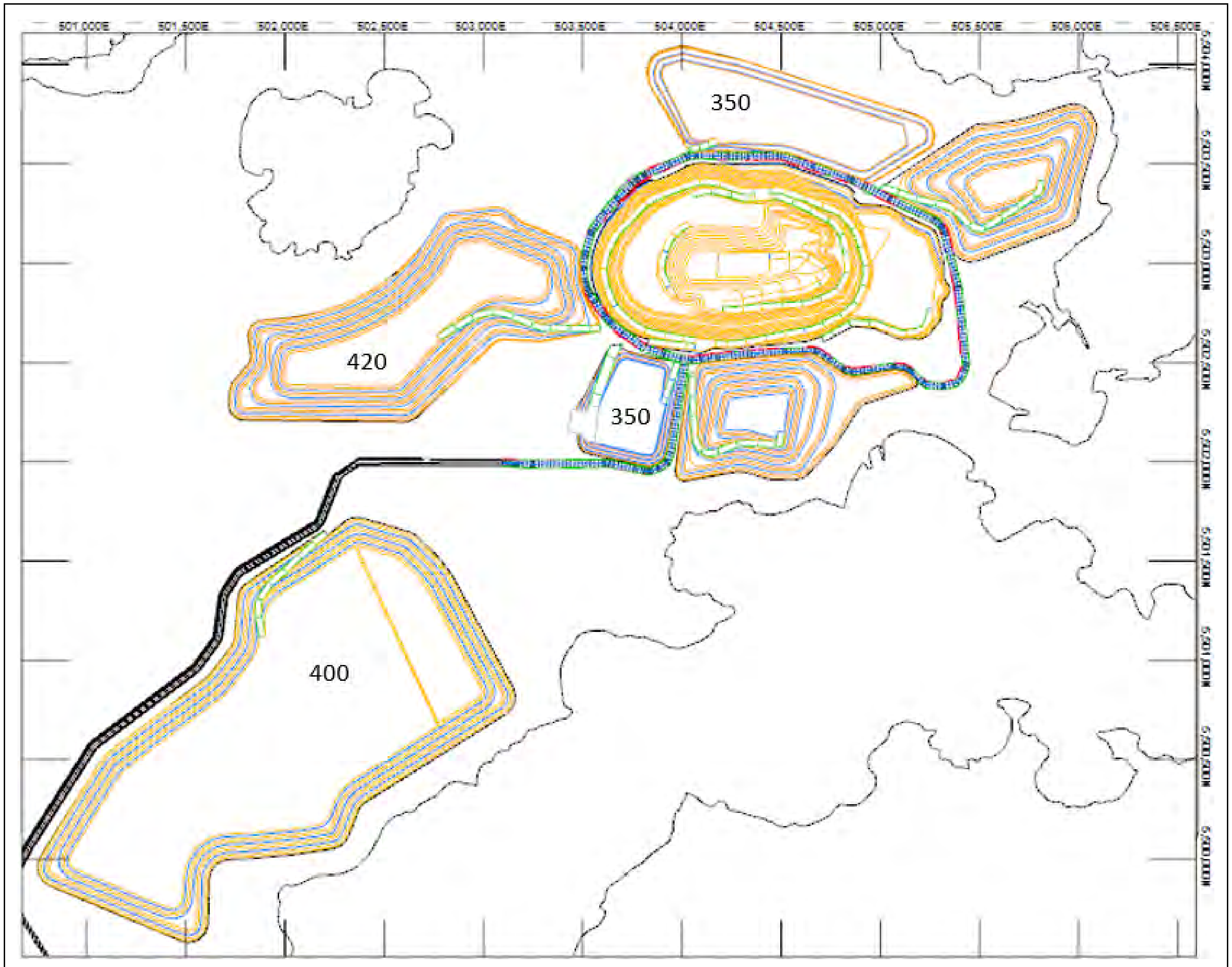
Source: Peter Harquail
Date: Oct. 2019

Figure 16.15: Production Schedule – Year 3



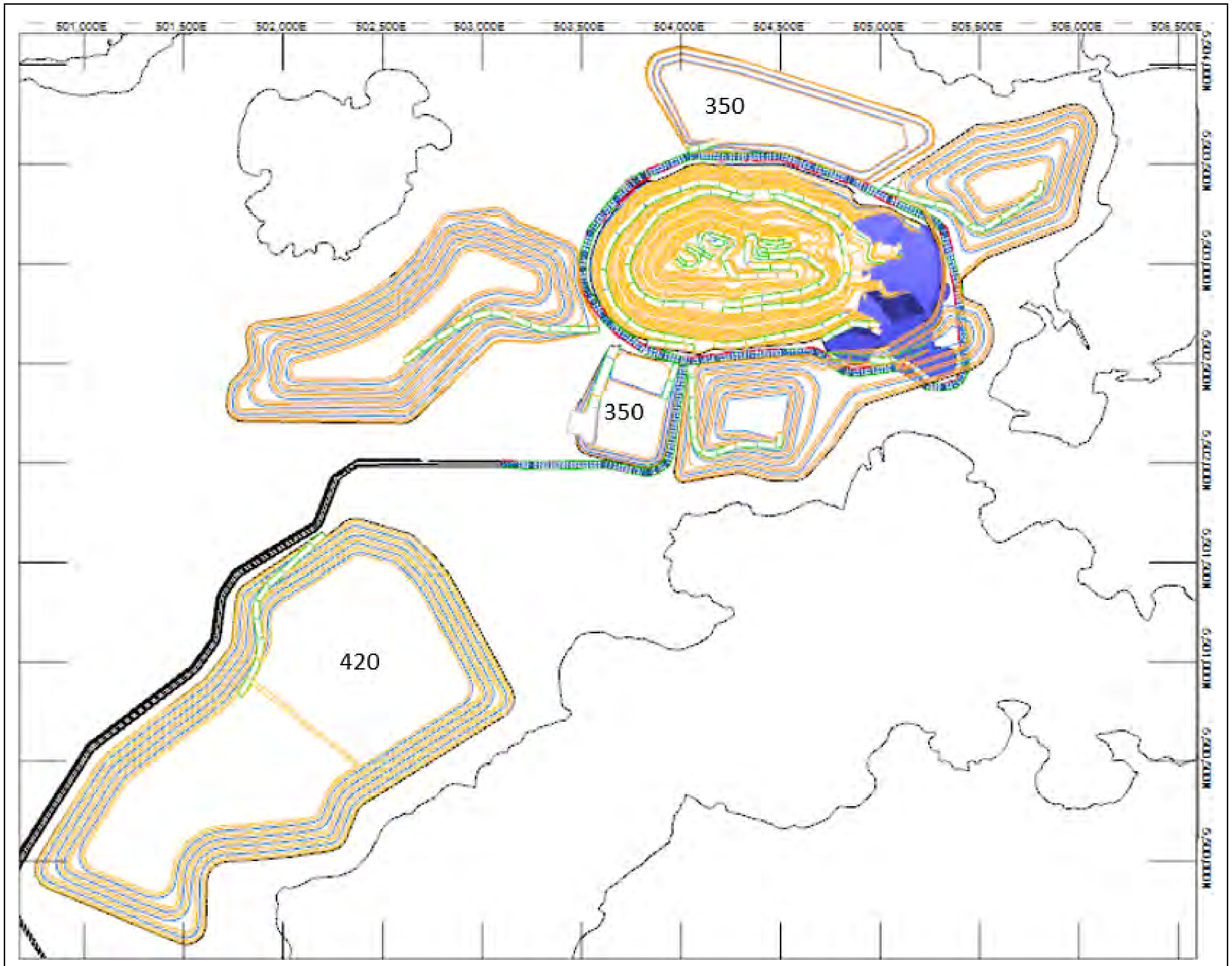
Mining Services	**Pit Schedule – Year 3**	**Source: Peter Harquail** **Date: Oct. 2019**

Figure 16.16: Production Schedule - Year 10



Mining Services	Pit Schedule – Year 10	Source: Peter Harquail Date: Oct. 2019

Figure 16.17: Production Schedule - Year 15



Mining Services	**Pit Schedule – Year 15**	**Source: Peter Harquail** **Date: Oct. 2019**

The mill production schedule is presented in Table 16.7. The milling rate will gradually increase from 12,000 t/d to 27,000 t/d in Year 1. The metallurgical recovery during the ramp up and commissioning period has been adjusted downwards from normal steady-state operating performance expectations.

Gold production averages 414 koz for the first five years of production (Year 1 to Year 5) with an average head grade of 1.45 g Au/t and an average metallurgical recovery of 91.2%. Over the LOM 5.05 Moz of gold are produced.

Table 16.7: Life-of-Mine Production Schedule

Year	Mining					Processing			
	Ore Mined (Mt)	Grade (g Au/t)	Contained Gold (koz)	Waste Mined (Mt)	Total Mined (Mt)	Ore Milled (Mt)	Grade (g Au/t)	Contained Gold (koz)	Recovered. Gold (koz)
-2	0.02	0.55	0.3	2.59	2.61	0.00	-	-	-
-1	4.47	1.26	181.2	22.19	26.66	0.24	0.74	5.8	5.2
1	9.30	1.29	386.6	51.46	60.76	8.00	1.78	458.0	422.7
2	10.38	1.20	399.7	59.56	69.94	9.88	1.34	425.7	389.1
3	11.82	1.31	496.5	58.28	70.10	9.86	1.51	477.8	432.7
4	11.13	1.44	513.8	58.76	69.89	9.86	1.59	503.3	459.7
5	9.08	1.24	362.6	60.98	70.05	9.86	1.18	374.5	339.3
6	6.59	1.32	279.9	63.47	70.06	9.88	1.01	321.5	289.2
7	8.68	1.06	294.6	61.22	69.89	9.86	0.97	306.9	277.8
8	12.08	1.07	416.1	58.00	70.08	9.86	1.23	391.2	357.1
9	10.87	1.12	390.9	59.05	69.92	9.86	1.20	380.2	349.0
10	10.71	1.17	403.2	51.17	61.89	9.88	1.24	394.7	360.0
11	10.90	1.31	460.0	38.05	48.95	9.86	1.42	448.6	408.4
12	10.37	1.56	520.2	24.64	35.01	9.86	1.62	514.9	474.1
13	7.95	1.51	385.6	18.34	26.29	9.86	1.29	407.4	372.9
14	0.99	1.51	48.1	1.83	2.81	8.73	0.46	128.7	114.1
Total	**135.32**	**1.27**	**5,539**	**689.60**	**824.92**	**135.32**	**1.16**	**5,539**	**5,051**

16.4 Mine Operations and Equipment Selection

16.4.1 Mine Operations Approach

Mining is to be carried out using conventional open pit techniques with hydraulic shovels, wheel loaders and mining trucks in a bulk mining approach with 10 m benches. An owner mining open pit operation is planned with the outsourcing of certain support activities such as explosives manufacturing and blasting activities.

16.4.2 Production Drilling and Blasting

Drill and blast specifications are established to effectively single pass drill and blast a 10 m bench. For this bench height, a 203 mm blast hole size is proposed with a 6.0 m x 6.5 m pattern with 1 m of sub-drill. These drill parameters combined with a high energy bulk emulsion with a density of 1.2 kg/m^3 result in a powder factor of 0.30 kg/t. Blast holes are initiated with NONEL 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.

Several rock formations are present in the pit including greywacke, gabbro, porphyry and BIF. The average rock properties based on testing show a range in hardness between 80 and 175 MPa with a weighted average hardness estimated at about 100 MPa.

A drilling test was conducted on site on various outcrops in the pit. A total of ten 165 mm test holes for a total 100 m was completed in three of the main rock formations (Porphyry, Greywacke and Iron Formation). The penetration rates did not vary significantly between the formations with an instantaneous penetration rate of 40.2 m/h.

The drilling test results were used to calibrate expected instantaneous penetration rates for the larger 203 mm diameter production blast holes. The average drill productivity for the production rigs is estimated at 39.1 m/h instantaneous with an overall penetration rate of 24.2 m/h. The overall drilling factor represents 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 is estimated at 2,364 t/h.

Table 16.8: Drill & Blast Parameters

Drill and Blast Parameters		Production Holes
Drill Pattern		
Explosive Type		Emulsion
Explosive Density	g/cm^3	1.2
Hole Diameter	in	8.0
Diameter (D)	m	0.203
Burden (B)	m	6
Spacing (S)	m	6.5
Subdrill (J)	m	1
Stemming (T)	m	2.6
Bench Height (H)	m	10
Blasthole Length (L)	m	11
Pattern Yield		
Rock Density	t/bcm	2.75
BCM/hole	bcm/hole	390
Yield per Hole	t/hole	1,073
Yield per Meter Drilled	t/m drilled	98
Powder Factor	kg/t	0.30
Weight of Explosives per Hole	kg/hole	327
Drill Productivity		
Re-drills	%	5%
Pure Penetration Rate	m/hr	39.1
Hole Length	m	11
Overall Drilling Factor (%)	%	62%
Overall Penetration Rate	m/hr	24.2
Drilling Productivity	t/hr	2,364
Drilling Efficiency	holes/hr	2.20

The blast hole rig selected for production drilling will have a hole size range of 152 mm to 270 mm with a single pass drill depth of 12.2 m with a 40 ft tower configuration. This rig will have both rotary and down-the-hole ("DTH") drilling capability. It is expected that DTH drilling mode will be most efficient. With the selection of the automation package for all units, a ratio of 1 operator per 2 units was assumed.

With the automation package, the drill rigs can be controlled remotely through a remote operator station (up to 3 drills per one remote station). From the remote station, the operator can initiate a complete automated drilling cycle, from the initial drilling to the retracting of the drill bit. During that time the operator can monitor the drilling process or switch to the other drills to either monitor, initiate a drilling cycle or remotely tram the drill to the next hole location.

The automation of the drilling process not only increases safety, but it also increases productivity as some time-consuming activities usually performed by the operator are not required anymore.

Blasting activities will be outsourced to an explosives provider who will be responsible for supplying and delivering explosives in the hole through a shot service contract. The mine engineering department will be responsible for designing blast patterns and relaying hole information to the drills via the wireless network.

16.4.3 Grade Control

The ore control program will consist of establishing dig limits for the four grade bins 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 a real-time view of dig blocks, ore boundaries and other positioning information.

In order to have optimal ore-waste boundaries identification, reverse circulation ("RC") drilling will target 100% of all ore material and also capture an average of 31% of the total waste in the pit. The first 6 benches will target more waste rock in order to refine the geological model.

The ore control boundaries will be established by the technical services department based on grade control information obtained through RC drilling and 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.

The samples collected will be sent to a nearby off-site laboratory for sample preparation and assaying for the LOM RC samples will be collected on the bench and properly tagged by grade control samplers on each shift.

Concurrent to the RC sampling, the production blast hole will target 100% of all ore material and also capture around 25% of the total waste in the pit (mainly the contact zones with ore).

16.4.4 Pre-Split

Pre-split drill and blast is planned to maximize stable bench faces and to maximize inter-ramp angles along pit walls as prescribed by the geotechnical pit slope study by Golder. The pre-split consists of a row of closely-spaced holes along the design excavation limit of interim and final walls. The holes are loaded with a light charge and detonated simultaneously or in groups separated by short delays. Firing the pre-split row creates a crack that forms the excavation limit and helps to prevent wall rock damage by venting explosive gases and reflecting shock waves. As a best-practice, it is recommended that operations restrict production blasts to within 50 m of an unblasted pre-shear line. Once the pre-split is shot, production blasts will be taken to within 10 m of the pre-shear and then a trim shot used to clean the face. Pre-split holes spaced 1.5 m apart will be 20 m in length and drilled with a smaller diameter of 127 mm (5 in.).

As presented in Table 16.9, blasting of the pre-split holes will use a special packaged pre-split explosive internally traced with 5 g/m detonating cord that ensures fast and complete detonation of the decoupled charge. For this specific application, a 41 mm diameter cartridge, 17 m long will be used which corresponds to a complete case of 25 kg. This load factor of 1.47 kg/m allows for a targeted charge weight of 0.83 kg/m^2 of face.

Table 16.9: Pre-Split Parameters

Pre-Split Parameters		Pre-Split Holes
Drill Pattern		
Hole Diameter	in	5
Diameter (D)	m	0.127
Spacing (S)	m	1.5
Bench Height (H)	m	20
Pre-Split Hole Length (L)	m	20
Face Area	m^2	30
Explosives Charge	kg	25
Charge Factor	kg/m^2 face	0.83
Cartridge Charge		
Nb Cartridges	qty	41
Cartridge Length	m	0.41
Cartridge Loading Factor	kg/m	1.47
Decoupled Charge Length	m	17
Decoupled Charge	kg	25
Drill Productivity		
Pure Penetration Rate	m/hr	41.2
Overall Drilling Factor (%)	%	58%
Overall Penetration Rate	m/hr	23.9
Drilling Efficiency	holes/hr	1.2
Meters of Drilling per m Crest	m/m of crest	13.33

The drill selected for this application is more flexible type of rig capable of drilling angled holes for probe drilling and pit wall drain holes. The hole size range of this rig is between 110 mm and 203 mm with a maximum hole depth of 31.5 m.

16.4.5 Loading

The majority of the loading in the pit will be done by two 29 m^3 face shovels and one 29 m^3 backhoe excavator. The shovels will be matched with a fleet of 216 t payload capacity mine trucks. The hydraulic shovels will be complemented by two production front-end wheel loaders (“FEL”) with 30 m^3 buckets.

Although interchangeable, the hydraulic shovels will primarily be operating in ore and overburden while the wheel loaders will primarily be operating in waste.

The loading productivity assumptions for both types of loading tools in ore, waste and overburden are presented in Table 16.10.

The two 29 m^3 face shovel are expected to achieve a productivity of 3,494 t/h based on a 4-pass match with the mine trucks and an average load time of 2.70 minutes. The productivity in overburden will decrease at 1,964 t/hr due to lower density of material and extra pass required for an average load time of 4.10 minutes. The 29 m^3 excavator, used for better selectivity, will have a 3,369 t/hr production rate in ore and waste rock at a cycle time of 2.80 minutes, whereas in overburden the excavator will reach a productivity of 1,964 t/hr with a loading cycle time of 4.10 minutes.

The wheel loaders are expected to achieve a productivity of 2,950 t/h based on a 4 pass match and an average load time of 3.2 minutes in ore and waste. The productivity in overburden is estimated at 1,677 t/hr.

Table 16.10: Loading Specifications

Loading Unit		Shovel (29 m^3)	Shovel (29 m^3)	Exc. (29 m^3)	Exc. (29 m^3)	FEL (30 m^3)	FEL (30 m^3)
Haulage Unit		**Truck (216 t)**	**Truck (216 t)**	**Truck (216 t)**	**Truck (216 t)**	**Truck (216 t)**	**Truck (216 t)**
Material		**Ore/Wst**	**Ovb**	**Ore/Wst**	**Ovb**	**Ore/Wst**	**Ovb**
Rated Payload	t	216	216	216	216	216	216
Heaped Volume	m^3	152	152	152	152	152	152
Bucket Capacity	m^3	29.0	29.0	29.0	29.0	30.0	30.0
Bucket Fill Factor	%	92%	89%	92%	89%	89%	86%
In-Situ Dry Density	t/bcm	2.75	2.00	2.75	2.00	2.75	2.00
Moisture	%	3%	5%	3%	5%	3%	5%
Swell	%	40%	25%	40%	25%	40%	25%
Wet Loose Density	t/lcm	2.02	1.68	2.02	1.68	2.02	1.68
Actual Load Per Bucket	t	53.98	43.36	53.98	43.36	54.02	43.34
Passes (Decimal)	#	4.00	4.98	4.00	4.98	3.99	4.98
Passes (Whole)	#	4.00	5.00	4.00	5.00	4.00	5.00
Actual Truck Wet Payload	t	216	217	216	217	216	217
Actual Truck Dry Payload	t	210	206	210	206	210	206
Actual Heaped Volume	m^3	107	129	107	129	107	129
Payload Capacity		100%	100%	100%	100%	100%	100%
Heaped Capacity		70%	85%	70%	85%	70%	85%
Cycle Time							
Truck Exchange	min	0.60	0.80	0.60	0.80	0.70	0.90
First Bucket Dump	min	0.10	0.10	0.10	0.10	0.10	0.10
Average Cycle Time	min	0.67	0.80	0.70	0.80	0.80	0.95
Load Time	min	2.70	4.10	2.80	4.10	3.20	4.80
Cycle Efficiency	%	75%	65%	75%	65%	75%	65%
Number of Trucks Loaded / hr	#	16.67	9.51	16.07	9.51	14.06	8.13
Production / Productivity							
Avg. Prod. Dry Tonnes / hr	t/hr	3,494	1,964	3,369	1.964	2,950	1,677

16.4.6 Hauling

Haulage will be performed with a 216-t class mine trucks. The truck fleet productivity was estimated in Talpac software. Several haulage profiles were digitized in Deswik with haul routes exported to Talpac to simulate cycle times. Cycle times have been estimated for each period and all possible destinations as there are several waste storage areas.

The assumptions and input factors for the Talpac simulations in Table 16.11, Table 16.12 and Table 16.13.

Two speed limits were applied in the simulation. On production benches, a speed limitation of 30 km/h was imposed to reflect the lack of proper road and less favorable rolling conditions in addition to having stopes in the pit floor. For all downhill ramps with an incline greater than 5%, the speed is limited to 25 km/h. Otherwise, the maximum truck speed reaches 59 km/h in the simulations.

Table 16.11: Speed Limits

Site Location	Speed Limit (km/h)
Pit on working bench, near dump face	30
Downhill Ramp < -5%	25
Mine Road and Ramps	No Imposed Limit

Table 16.12: Rolling Resistance

Road Type	Rolling Resistance (%)
Main Road	2.50
Ramp	3.00
Pit Floor and Near Dump Face	4.00

Table 16.13: Cycle Time Components

Cycle Time Component	Duration (min)
Truck Average Load Queue Time [1]	1.41
Truck Average Spot Time at Loader	0.60
Truck Average Loading Time	2.93
Truck Average Dump Queue Time	0.00
Truck Average Spot Time at Dump	0.30
Truck Average Dumping Time	0.20

Note 1: The Average Load Queue Time was set to 40% of the Loading Time and Spot Time

The multiple waste dumps were used to help level the truck requirements for the Project. During the critical years of the Project, leveling was achieved by sending waste rock to the closest dumps.

The Table 16.14 shows the haulage hours and the calculated cycle time by material type. Typically cycle time increases with the increase of the depth of the pit over the mine life. Cycle time is also dependant on the dumping schedule and the distance each dump is from the pit. The dump schedule was planned that cycle time tends to plateau at a max limit to allow for a consistent fleet over the majority of the mine life. This is depicted in the table in consecutive years in which the cycle time for waste holds near constant or decrease. Another factor for variable cycle time is the phasing. New phases have a lower depth and shorter cycle time associated with them. It is common for three or more phases to be active within the same year. The average cycle time shown in the table is a weighted average of all the phase cycles to all the used dumps. Figure 16.8 depicts the tonnage moved by year and the associated cycle times per material.

Figure 16.18: Cycle Time by Category and Material Moved

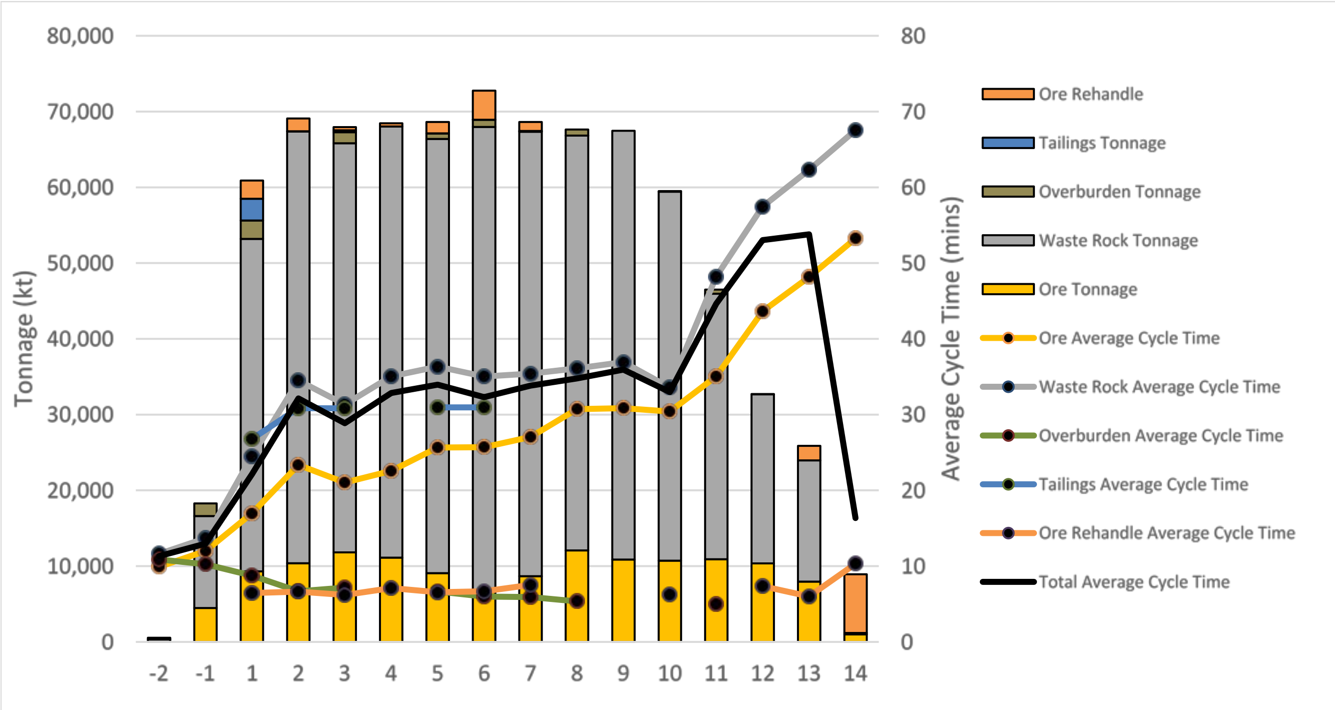


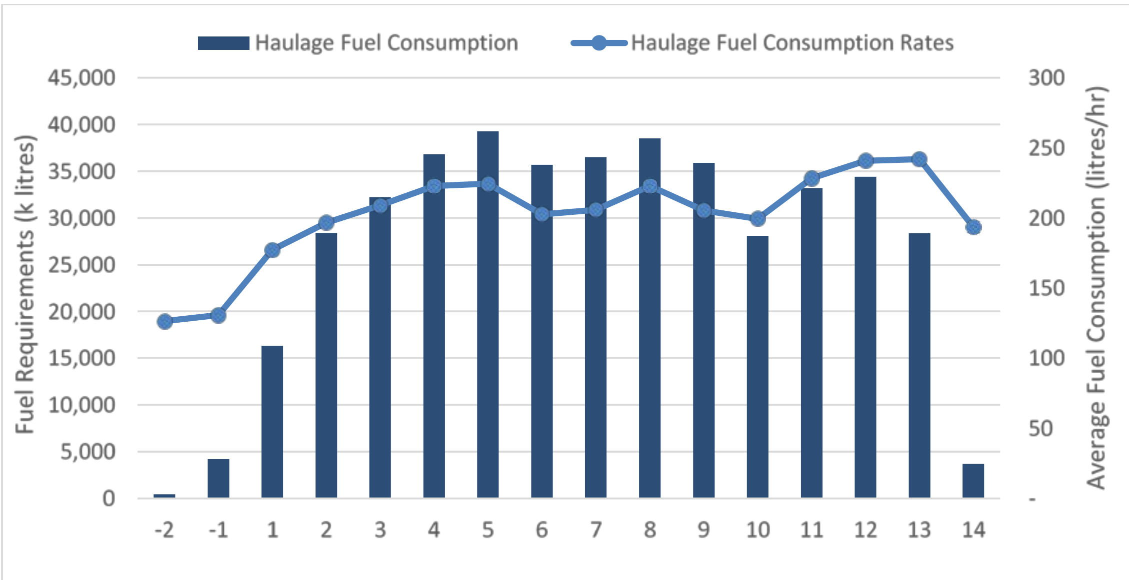
Table 16.14: Haulages Hours and Cycle Times by Material Type

Material Type			Total LOM	-2	-1	1	2	3	4	5
Ore	Hours	hr	**342,291**	15	6,052	12,108	17,829	22,016	20,185	18,483
	Tonnage	kt	**135,323**	16	4,467	9,300	10,376	11,824	11,129	9,077
	Av.Cycle Time	min	**28.77**	9.92	12.01	16.97	23.36	21.04	22.57	25.66
Waste	Hours	hr	**1,624,008**	1,370	16,598	68,657	123,420	127,899	143,007	152,402
	Tonnage	kt	**635,273**	331	12,151	43,901	57,017	54,001	56,909	57,306
	Av.Cycle Time	min	**36.04**	11.63	13.70	24.51	34.50	31.34	35.06	36.34
Overburden	Hours	hr	**7,535**	207	1,731	2,139	12	1,181	-	561
	Tonnage	kt	**8,857**	190	1,659	2,415	14	1,451	-	717
	Av.Cycle Time	min	**7.77**	10.89	10.25	8.74	6.69	7.16	-	6.66
Tailings	Hours	hr	**6,637**	-	-	5,979	-	589	-	68
	Tonnage	kt	**3,164**	-	-	2,888	-	247	-	28
	Av.Cycle Time	min	**27.16**	-	-	26.81	30.84	30.83	-	30.95
Ore Rehandle	Hours	hr	**12,983**	-	-	1,185	845	204	225	757
	Tonnage	kt	**21,186**	-	-	2,393	1,690	426	411	1,498
	Av.Cycle Time	min	**7.95**	-	-	6.44	6.64	6.19	7.10	6.54
Construction	Hours	hr	**39,826**	1,951	7,890	2,127	2,386	2,427	1,746	2,753
	Tonnage	kt	**42,303**	2,072	8,381	2,260	2,534	2,577	1,855	2,924
	Av.Cycle Time	min	**31.09**	20.52	20.52	20.52	28.34	25.96	26.53	30.46
Total	Hours	hr	**2,033,280**	3,544	32,271	92,196	144,492	154,314	165,163	175,023
	Tonnage	kt	**846,106**	2,610	26,658	63,157	71,630	70,526	70,304	71,551
	Av.Cycle Time	min	**33.60**	18.63	15.35	22.07	32.00	28.77	32.69	33.82

Material Type			6	7	8	9	10	11	12	13	14
Ore	Hours	hr	13,370	19,097	30,505	26,379	26,937	33,592	48,430	41,215	6,080
	Tonnage	kt	6,587	8,676	12,081	10,871	10,715	10,901	10,369	7,945	987
	Av.Cycle Time	min	25.71	27.05	30.73	30.86	30.41	35.05	43.65	48.17	53.25
Waste	Hours	hr	158,930	155,146	139,379	146,073	111,631	109,128	92,205	72,974	5,190
	Tonnage	kt	61,379	58,654	54,744	56,596	48,716	35,047	22,307	16,014	201
	Av.Cycle Time	min	35.03	35.41	36.11	36.92	33.58	48.22	57.46	62.31	67.53
Overburden	Hours	hr	704	81	554	-	-	364	-	-	-
	Tonnage	kt	954	110	800	-	-	547	-	-	-
	Av.Cycle Time	min	5.99	5.91	5.35	-	-	4.97	-	-	-
Tailings	Hours	hr	1	-	-	-	-	-	-	-	-
	Tonnage	kt	1	-	-	-	-	-	-	-	-
	Av.Cycle Time	min	30.95	-	-	-	-	-	-	-	-
Ore Rehandle	Hours	hr	1,979	685	-	-	33	-	18	886	6,165
	Tonnage	kt	3,836	1,180	-	-	68	-	32	1,911	7,740
	Av.Cycle Time	min	6.68	7.52	-	-	6.26	-	7.36	6.00	10.31
Construction	Hours	hr	1,071	2,312	2,312	2,312	2,312	2,312	2,193	2,193	1,531
	Tonnage	kt	1,138	2,456	2,456	2,456	2,456	2,456	2,329	2,329	1,626
	Av.Cycle Time	min	31.20	31.95	31.95	31.95	33.37	33.37	50.81	54.66	59.69
Total	Hours	hr	176,054	177,321	172,750	174,763	140,913	145,396	142,846	117,268	18,966
	Tonnage	kt	73,894	71,075	70,081	69,922	61,955	48,950	35,038	28,200	10,555
	Av.Cycle Time	min	32.29	33.76	34.69	35.80	32.99	44.06	52.89	53.88	23.03

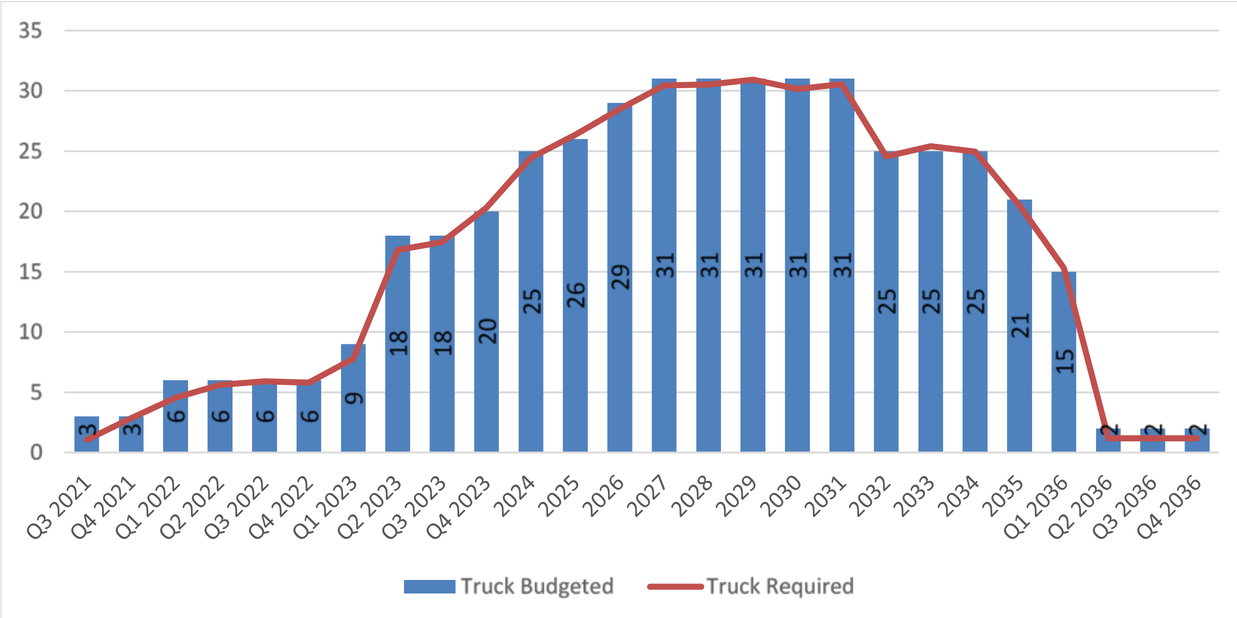
The fuel consumptions were also estimated with Talpac which generates a specific engine load factor depending on the proportion of the travel on ramp grades and on flatter gradients. Generally, the fuel burn rate increases with depth as a longer period of time is spent on grade. Total fuel usage is a function of fuel burn rate, haulage requirements and cycle times/paths. Over the LOM, the year with largest fuel consumption is Year 5 with 38.5 Ml of fuel consumed by the haulage trucks. Fuel consumption and fuel consumption rates by year are depicted in Figure 16.19 below.

Figure 16.19: Mining Truck Fuel Requirements



The total haul hours required by period coupled to the truck mechanical availability were used to determine the number of trucks required throughout the LOM. The truck fleet reaches a maximum of 31 units in Year 5 and remains at this level until Year 9 before it starts decreasing as a result of a decrease in mining rate.

Figure 16.20: Truck Requirement – Required vs. Budgeted



16.4.7 Dewatering

The open pit dewatering strategy will consist of using the underground opening and the connectivity of the past underground mines (Hard Rock and Mosher) to keep the water level below the working benches. Hardrock shaft will be used to dewater to the 200 level after which dewatering will need to be performed from the Mosher shaft (reach the -265 level). This groundwater dewatering will be performed using submerged electric pumps.

Surface water will be pumped by mobile diesel pumps placed in sumps on the mining level. With the deepening of the pit additional pumping capacity and HDPE pipes will be added to the dewatering system.

16.4.8 Road and Dump Maintenance

Waste and ore storage areas will be maintained by a fleet of six 630 HP track-type dozers. A 500 HP wheel loader will also be purchased and dedicated to mine roads and the loading areas.

Mine roads will be maintained by three 16 ft blade motor graders. A water/sand truck will be used to spray roads to suppress dust or spread road aggregate during winter months. Two small water trucks will spray the site roads to suppress dust and fill the production drills with water. The small water trucks will also be used in case the larger (76 kL) water truck is unavailable.

16.4.9 Support Equipment

All construction related work, such as berm construction, water ditch cleaning will be done by two 49 t excavators (one of them will be equipped with a hydraulic hammer) and one 90 t excavator for pit wall scaling.

Four pit buses will transport workers to their assigned workplace and a total of 37 pick-ups will be purchased for all departments (including Mill and G&A).

Several other equipment purchases are planned to support the mining activities. Part of this list of equipment are one 60 t crane, 1 boom trucks (28 t crane), one knuckle boom truck (10t), one 250 HP utility wheel loader for smaller work and a 100t low-boy trailer and tractor for moving the tracked equipment.

16.4.10 Mine Maintenance

The Hardrock Project does not intend to enter into a maintenance and repair contract ("MARC") for its mobile equipment fleet. Consequently, the maintenance department has been structured to fully manage this function, performing maintenance planning and training of employees. However, reliance on dealer and manufacturer support is planned for the initial 5 years of the project. Support for major components will however be maintained throughout LOM, such as through component exchange programs.

Tire monitoring, rotation and/or replacement will be outsourced to a contractor permanently installed on site, providing skilled labor and equipment to perform the job safely and efficiently.

A computerized maintenance management system will be used to manage maintenance and repair operations. This system will keep up-to-date status, service history and maintenance needs of each machine while being the source of data for KPIs and cost tracking purposes.

16.4.11 Mine Management and Technical Services

The mine is headed by a Mine Operations Manager who is responsible for the overall management of the mine. Superintendent positions in operations, maintenance, geology (Chief) and engineering (Chief) report directly to the Mine Operations Manager.

The operations department is composed of two foremen per crew (eight in total); one foreman for loading and hauling and another foreman for drilling and blasting. A mine dispatcher is planned on each shift. To increase operator level performance and organize structured training programs, one mine trainer is planned on day shift (under G&A). The operations department includes 11 staff employees at peak level.

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.

The rental of an equipment simulator has been planned at the end of pre-production to match with the increasing number of truck operators required.

16.4.12 Roster Schedules

A 3 on / 4 off, 4 on / 3 off rotating schedule has been planned on a twelve-hour shift. Four crews are required to operate on a continuous basis 24 hours per day 365 days per year.

16.4.13 Equipment Usage Model Assumptions

The typical equipment usage model assumptions are established by equipment groupings as presented in Table 16.15. The annual net operating hours ("NOH") varies approximately between 5,000 and 6,000 hours per year. During the first 20,000 hours the mine truck availability was set to 86%. This is to reflect the lower maintenance requirements during the truck early years. Following this interval, the availability is estimated at 83.5% (average of 84% for the life-of-mine).

Table 16.15: Equipment Usage Model Assumptions

Equipment Usage Assumptions		Shovels	Loaders	Trucks	Drills	Ancillary
Days in period	days	365	365	365	365	365
Availability	%	82.0	80.0	84.0	80.0	85.0
Use of Availability	%	90.0	90.0	90.0	90.0	85.0
Utilization	%	73.8	72	73.8	72	72.25
Effectiveness	%	87.0	85.0	87.0	85.0	80.0
Overall Equipment Effectiveness	%	64.2	61.2	64.2	61.2	57.8

16.5 Fleet Management

A fleet management system will be implemented to manage the operation, monitor machine health, and track key performance indicators ("KPIs"). The system will be managed by a dispatcher on each crew who will control the system which will send operators onscreen instructions to work at peak efficiency. A Dispatch system coordinator will be required to assure proper functioning of system hardware and software with ongoing annual vendor support.

A high-precision global positioning system ("GPS") for machine guidance is considered to mitigate the associated risk of working around historic underground workings. This will enable shovel operators to navigate safely in potentially hazardous areas. In addition to protecting people and equipment, the high precision system will improve the productivity and bench grade control. The results and usefulness of such a system have proven to be worthwhile at other mines where past underground mines have been developed. Similarly, high precision drill navigation systems will be installed on the production drills and auxiliary drills to guide rigs into position and assure holes are drilled to the correct depth and location.

16.6 Pit Slope Monitoring and Voids Management

16.6.1 Pit Slope Monitoring

Rock mass failure is not considered as a risk due to the high overall rock mass strength. However, slope movement monitoring is planned for the open pit to gather measurements and confirm assumptions in order to assure safe working conditions. Initial slope movement monitoring would consist of 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 will provide movement response data to verify visual observations and if the slope is performing adequately.

Pit wall mapping using routine digital mapping techniques using photogrammetry is recommended. Physical geological mapping is also recommended to supplement and qualify data derived from photogrammetry.

The slope movement monitoring data will be important for the calibration of numerical models required for detailed design updates during the mine life. The pit phasing approach will allow for adjustments to the final design based on observations and knowledge gained with the interim pit phases.

16.6.2 Voids Management

A number of open pit mines in Canada and Western Australia are mining orebodies that have previously been mined by underground methods. There are hazards with high risk potential when approaching and then progressively mining through underground workings.

The hazards related to underground workings include:

- Sudden and unexpected collapse of the open pit floor and/or walls;
- The loss of people and/or equipment into unfilled or partially filled underground workings;
- Loss of explosives from charged blast holes that have filled cavities connected to the blast hole;
- Overcharging blast holes where explosives have filled cavities connected to the blast hole;
- Risk of flyrock from cavities close to the pit floor and adjacent blast holes.

Previous historical underground workings are well documented and available in 3D electronic format. The workings are not a concern for the overall stability of the final walls. In the pit designs, the ramps were kept away from the known historical underground openings. Larger berms were designed to create access points around the bigger underground openings at different heights. As part of detailed design, each stope that will underlie the pit will require a detailed assessment to determine the best operating practices for safe mining. The assessment should be initiated at least a year before the pit deepens to a critical proximity.

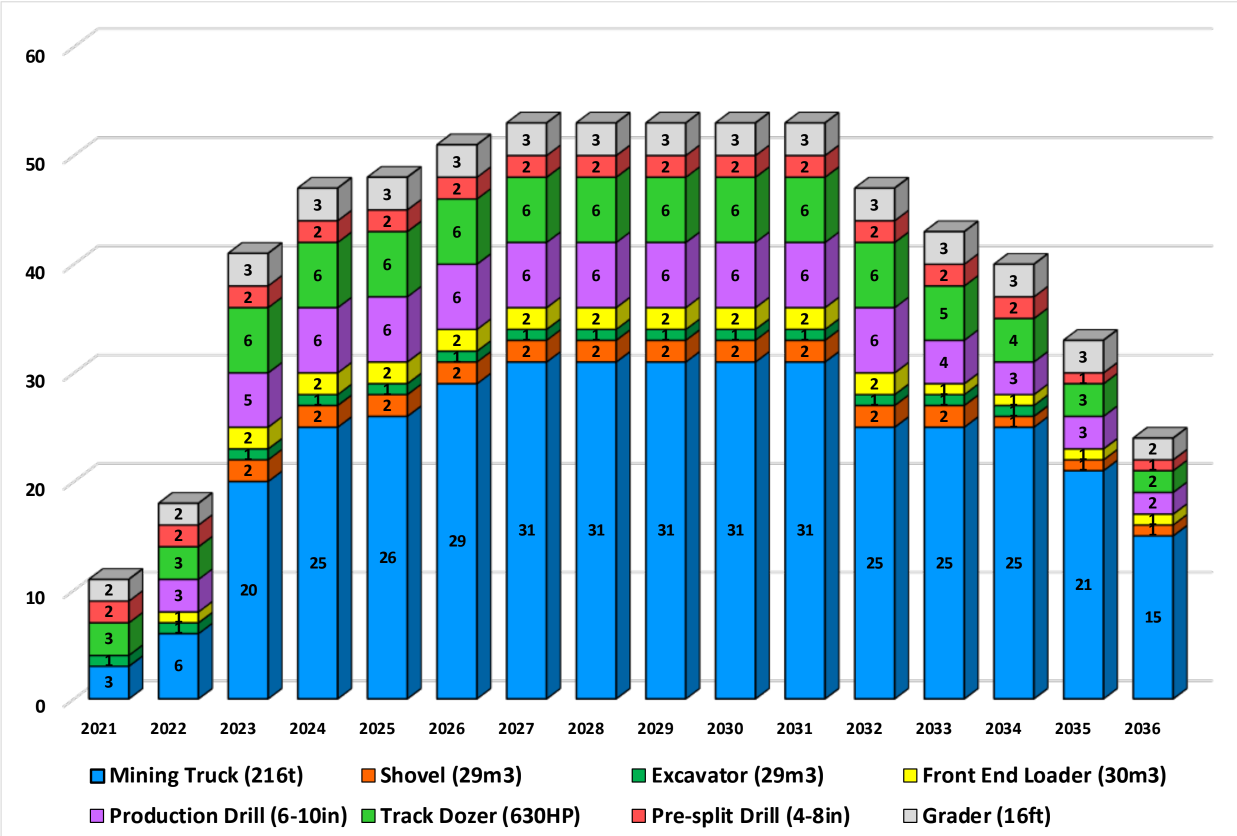
A boom truck with various attachments have been planned in the OPEX and CAPEX to facilitate some of the work around the underground openings. A cost for placement of crushed material has also been budgeted.

16.7 Mine Equipment Requirements

The main factors which influenced the selection of the major mine equipment included the annual production requirements and optimization of the fleet size.

An extensive analysis was performed to determine the optimal fleet size, equipment type and preferred suppliers. The requirements of major mining units are presented in Figure 16.21, Table 16.16 presents the equipment purchase schedule.

Figure 16.21: Mine Equipment Requirements



16.8 Mine Workforce Requirements

Table 16.17 presents the mine workforce requirements over the mine life with a reduction occurring when the tonnage decreases during the Year 10 of operation. The mine workforce reaches a peak of 371 individuals at Year 6.

Table 16.16: Equipment Purchase Schedule

Equipment Purchase Schedule	Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Major Equipment																		
Haul Truck Mining Truck (216 t)	31	-	3	4	18	1	3	2	-	-	-	-	-	-	-	-	-	-
Production Front Shovel (29 m^3)	2	-	-	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-
Production Backhoe Excavator (29 m^3)	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Front End Loader (30 m^3)	2	-	-	1	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Production Drill (6-10 in)	6	-	-	3	3	-	-	-	-	-	-	-	-	-	-	-	-	-
Track Dozer (630 HP)	12	-	3	-	3	-	-	-	-	-	3	1	2	-	-	-	-	-
Motor Grader (16 ft)	7	-	2	-	1	-	-	-	-	2	1	-	-	-	-	1	-	-
Wheel Loader (500 HP)	2	-	-	1	-	-	-	-	-	-	1	-	-	-	-	-	-	-
Water/Sand Truck (76kL tank)	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Auxiliary Pre-split Drill (4-8 in)	4	-	2	-	-	-	-	-	-	-	-	2	-	-	-	-	-	-
Total	**68**	**-**	**11**	**10**	**28**	**1**	**3**	**2**	**-**	**2**	**5**	**3**	**2**	**-**	**-**	**1**	**-**	**-**
Support Equipment																		
Excavator (49t)	4	-	1	-	1	-	-	-	-	-	1	1	-	-	-	-	-	-
Excavator (90t)	2	-	1	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-
Hydraulic Hammers for Excavator 49 t	5	-	-	1	-	-	1	-	-	1	-	-	1	-	1	-	-	-
Utility Wheel Loader - (250 HP)	2	-	1	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-
Small Water Truck (Hook lift)	6	-	2	-	-	-	-	2	-	-	-	2	-	-	-	-	-	-
RT Crane 60 t	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Boom Truck (28 t crane)	3	-	1	-	-	-	-	1	-	-	-	1	-	-	-	-	-	-
Knuckle Boom Truck (10 t)	2	-	1	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-
Telehandler	2	-	1	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-
Forklift 4 t	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Manlift 60 ft	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Electric Scissor Lift 32 ft	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mechanic Service Truck	6	-	1	1	1	-	-	-	-	-	2	1	-	-	-	-	-	-
Welding Truck	4	-	-	-	2	-	-	-	-	-	-	2	-	-	-	-	-	-
Equipment Purchase Schedule	**Total**	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**
Shovel & Drill Repair Trailer	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Fuel/Lube Truck	6	-	1	1	-	-	-	2	-	-	-	2	-	-	-	-	-	-
Lube Truck	2	-	1	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-
Lowboy & Tractor	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pickups	104	-	8	15	14	-	-	-	8	29	-	-	-	-	30	-	-	-
Pit Busses	11	-	1	1	2	-	-	-	2	2	-	-	-	-	2	1	-	-
Mobile Air Compressor 375 CFM	3	-	-	1	-	-	-	-	-	1	-	-	-	-	1	-	-	-
Electric Welding Machine	4	-	2	-	-	-	-	-	-	-	-	-	2	-	-	-	-	-
Mobile Welding Machine	8	-	2	-	2	-	-	-	-	-	2	2	-	-	-	-	-	-
Lighting Towers	28	-	4	6	-	-	-	-	4	6	-	-	-	4	4	-	-	-
Genset 6 kW	6	-	3	-	-	-	-	-	-	-	3	-	-	-	-	-	-	-
Genset 60 kW	6	-	1	-	1	-	-	-	2	-	-	-	1	1	-	-	-	-
Spare Box for Haul Trucks	2	-	-	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-
Spare Bucket for Shovels	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Spare Bucket for Excavator	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Spare Bucket for Loaders	2	-	-	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-
Spare Bucket for Small Excavator	2	-	-	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-
Dewatering Pump – 10 in	16	-	2	-	-	2	2	-	-	-	-	4	-	-	2	4	-	-
Dewatering Pump – 6 in	6	-	1	1	-	-	-	-	1	1	-	-	-	-	2	-	-	-
Trash Pump 3 in	18	-	4	-	-	-	4	-	-	4	-	4	-	-	2	-	-	-
Diesel Powered Air Heaters	6	-	2	-	-	-	-	2	-	-	-	2	-	-	-	-	-	-
Industrial Sweeper / Floor Cleaner	3	-	1	-	-	-	-	-	1	-	-	-	1	-	-	-	-	-
Mine Maintenance Equipment and Tooling	2	-	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Drill Automation Package	2	-	-	1	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Dispatch System (Stage 1+Stage 2)	1	-	-	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-
Mining Software Package 1+2+3	2	1	1	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-
Survey Tools (gear, base stations and drone)	3	-	1	1	1	-	-	-	-	-	-	-	-	-	-	-	-	-

Table 16.17: Workforce Requirements

Department	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Mine Operations	1	64	103	199	219	223	231	239	239	235	235	235	215	203	195	164	120
Mine Maintenance	0	29	41	88	96	96	99	99	103	103	103	103	91	87	83	75	65
Mine Geology	2	15	16	16	16	16	16	16	16	16	16	16	16	16	16	15	14
Mine Engineering	2	8	9	12	12	13	13	13	13	13	13	13	13	13	13	10	9
Total Workforce	**5**	**116**	**169**	**315**	**343**	**348**	**359**	**367**	**371**	**367**	**367**	**367**	**335**	**319**	**307**	**264**	**208**

17. RECOVERY METHODS

17.1 Process Plant Design Criteria

The process design criteria have been established based on: testwork results, trade-off studies, Greenstone Gold Mines Limited ("GGM") client and vendor recommendations and industry practices.

The plant will ramp-up to the nameplate capacity of 27,000 t/d in approximately one year (grind size of 80% passing (P_{80}) 90 µm. The grinding circuit includes a high-pressure grinding roll (HPGR), two identical ball mills and two identical gravity concentrators. The mill operation schedule is 24 h/d, 365 d/y with an overall availability of 92%. Crushing plant and processing plant equipment design factors allow for a margin of error in the sizing of the equipment. They are used in the calculations of the equipment feed rates and residence times. The key general process design criteria are presented in Table 17.1.

Table 17.1: Key General Process Design Criteria

Parameter	Units	Value
Throughput – Design	t/y	9,855,000
Throughput – Design	t/d	27,000
Throughput – Design	t/h	1223
Design Grind Size (P80)	µm	90
Crusher Utilization	%	67
Process Plant Availability	%	92
Operating Time	d/y	365
Operating Time – Concentrator	h/d	24
Au Feed Grade - Average	g/t	1.34
Au Feed Grade - Design	g/t	2.10
Ore Moisture	%	3.0
Ore Specific Gravity		2.81
Gold Recovery	%	91.0
Elution Vessel Capacity	t	10
Crushing Plant Equipment Design Factor	%	20
Process Plant Equipment Design Factor	%	10

17.1.1 Comminution Design Values

The comminution test work program determined grinding characteristics for the various lithologies. Based on the run-of-mine expected composition (refer to Table 17.2), the weighted averages were calculated to establish the plant feed grindability parameters. The results are compiled in Table 17.3. The 75th percentiles of hardness are used for design purposes to ensure sufficient equipment capacity to handle process variations.

The ore hardness data available when the ball mill design was selected was measured on the composite samples made up of a blend of 53 different core intervals originating from different lithologies representing the entire deposit. The weighted average of the composite samples Bond ball mill work index (BWI) obtained was 15.5 kWh/t which was used for the design of the ball mills. Modified BWIs, considered as more accurate for the prediction of the grinding circuit behavior, were also measured on samples from various lithologies. The overall 75th percentile result obtained for the samples from various lithologies was 15.5 kWh/t. That confirms that the design value of 15.5 kWh/t is satisfactory and conservative. Furthermore, the variability between the results obtained for each lithology is small, justifying the use of only one value of BWI to represent the entire deposit for the design of the ball mills. The HPGR also generates microcracks on the ore particles which typically reduces the power required at the ball mill. The microcrack effect was not considered when designing the ball mills which provides additional contingency.

Table 17.2: Global Composite Sample Composition

Lithology	Units	Content
Greywacke (S3E) and Gabbro (I1A)	%/ wt	53.2
Iron Formation (C2A)	% wt	30.2
Porphyry (I3P)	% wt	16.5
Conglomerate (S4) and Ultramafic (I0)	% wt	0.1
Total	**% wt**	**100.0**

Table 17.3: Comminution Parameters (Weighted Averages)

Comminution Parameters	Units	Value
Bond Rod Work Index (RWI) (80th percentile)	kWh/t	16.8
Standard Bond Ball Mill Work Index (BWI) (75th percentile)	kWh/t	15.5
Modified Bond Mill Work Index (BWI) (75th percentile)	kWh/t	15.5
Abrasion Index (Ai) (average)	g	0.127
Unconfined Compressive Strength (UCS) (80th percentile)	MPa	242
Unconfined Compressive Strength (UCS) (range)	MPa	206-291
JK Breakage Resistance Number (20th percentile)	(Axb)	26.1
JK Abrasion Resistance (10th percentile)	(ta)	0.22
HPGR Specific Grinding Force	N/mm^2	3.2
HPGR Specific Throughput (mf)	ts/hm^3	240
HPGR Specific Throughput rate (mc)	ts/hm^3	202
HPGR Specific Power	kWs/m^3	620

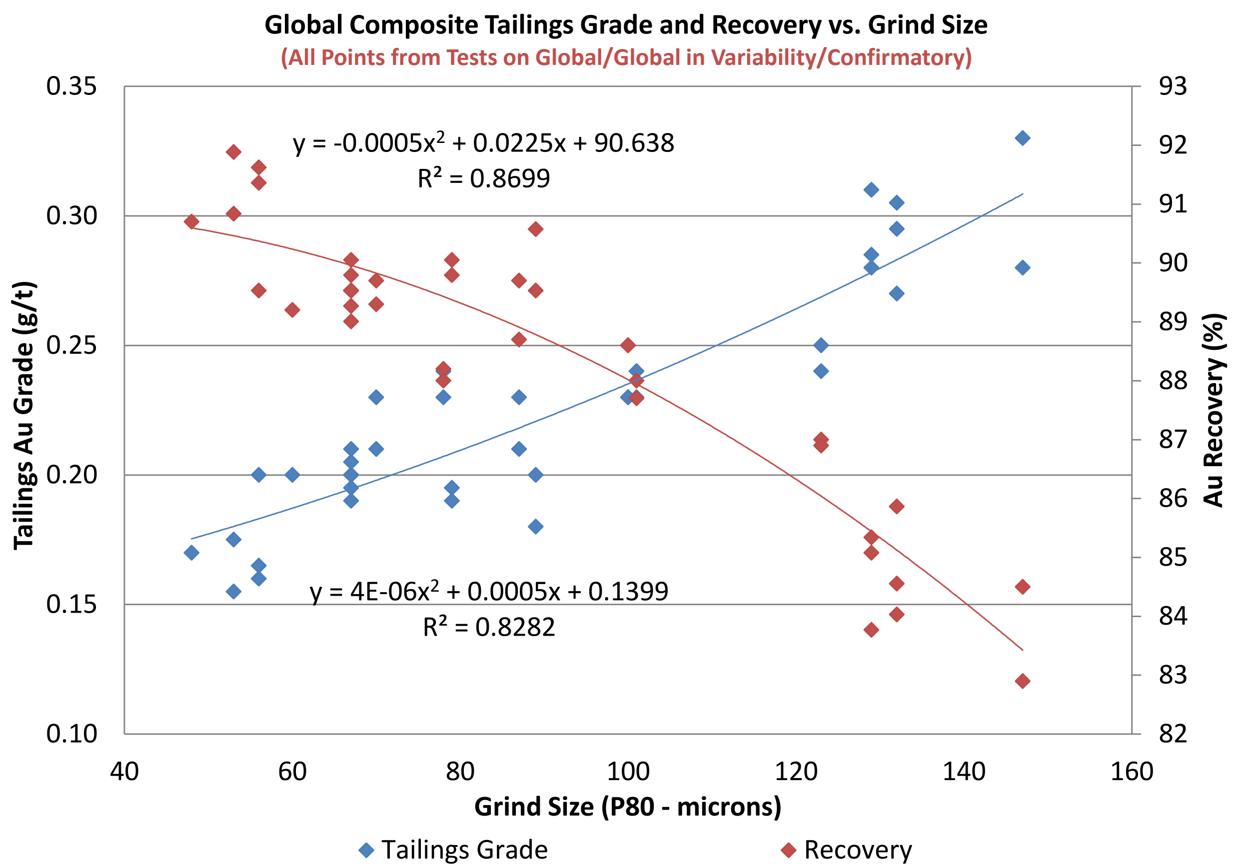
17.1.2 Grind Size Determination

The cyanidation test work established a correlation between grind size and gold recovery whereby a finer grind results in a higher recovery.

The Global Composite is considered to be the most representative of the run-of-mine over the life-of-mine, the results of the leach tests on the Global Composite were used to determine the optimal grind size (refer to Figure 17.1).

The analysis has also been conducted on the results from leach tests on the Variability Composites A to I, the Master Composite (representing the Feasibility Study mine design for the first three years of operation) and the Low Grade Composites. Results from these other samples are used to evaluate the impact of ore variability.

Figure 17.1: Global Composite Tailings Grade and Recovery vs. Grind Size



Title: Global Composite Tailings Grade and Recovery vs. Grind Size	Author: Soutex/WSP Date: 14/07/21

Economic evaluations completed in 2014 showed that a grind of P_{80} = 72 µm corresponds to the highest net revenue of additional recovery over incremental costs, but is constrained at 24,000 t/d with the ball mills selected. At 27,000 t/d, a compromise is made between grind fineness and throughput and a 90 µm grind is considered optimal. The lower recovery between 72 µm and 90 µm is offset by the higher revenue with the increased throughput. Lower throughputs during production ramp up will allow for finer grinds, and higher recovery in this period.

17.1.3 Impact of Mineralogical Composition on Leach Performance

Leach testing indicated that refractory arsenopyrite content in the deposit may corelate to recovery. A multivariate linear regression analysis was used to determine the correlation between leach residue grade and mineralogical composition. Multivariate regression models make it possible to describe how one variable (response) reacts to simultaneous changes in other variables (predictors). The method enables the combined impact of each predictor variable to be quantified on the response variable, which is not possible via simple regression analysis.

The results of the leach tests conducted during the feasibility study stage, described in Subsection 13.2.2 and the basic engineering stage, described in Subsection 13.3 were used as the basis for the analysis. The residual gold grade from the leach testwork was found to be highly correlated to the gold, arsenic and sulfur head sample grades. The strong correlation between the residual gold grade and arsenic and sulfur head grades suggests that arsenopyrite (FeAsS) contains refractory gold which is not recovered via leaching. Table 17.4 shows the composites used for the analysis, the number of tests made on each composite and the head grades. The composites A to I and WCE were produced for the feasibility study testwork completed in 2014. The composites 1 through 17 were prepared for the basic engineering tests completed at SGS Lakefield in 2019. The head grade of composite 12 was significantly higher than planned and was removed from the dataset for the model since it would lead to bias.

Table 17.4: Composite for Multivariate Analysis

Composite	Number of Leach Tests	Head Au (g/t)	Head As (%)	Head S (%)
Global	5	1.74	0.010	1.70
A	3	2.56	0.190	1.56
B	3	2.04	0.150	0.85
C	3	1.71	0.070	1.37
D	3	1.68	0.120	3.56
E	3	1.18	0.110	0.99
F	3	1.36	0.029	1.78
G	2	1.59	0.062	0.68
H	2	2.65	0.074	2.92
I	3	2.29	0.280	1.48
Master	10	1.94	0.200	1.88
S3E-0.5-WCE	2	0.55	0.040	0.37
S3E-0.7-WCE	2	0.67	0.027	0.51
I3P-0.5-WCE	2	0.46	0.002	0.27
I3P-0.7-WCE	2	0.75	0.029	0.42
C2A-0.5-WCE	2	0.34	0.027	1.06
C2A-0.7-WCE	2	0.85	0.014	1.55
1	1	0.55	0.060	1.50
2	2	0.57	0.002	0.23
3	4	1.66	0.038	1.46
4	7	0.73	0.011	0.57
5	1	0.51	0.072	1.03
6	2	0.50	0.006	0.44
7	4	1.06	0.059	0.95
8	2	1.55	0.008	0.44
9	8	1.35	0.017	0.54
10	1	0.40	0.052	0.99
11	2	0.44	< 0.001	0.30
13	2	1.02	0.002	0.35
14	2	0.99	0.008	0.62
15	6	0.73	0.007	0.43
16	3	1.09	0.038	0.88
17	5	1.15	0.023	0.58
Total	**104**			
Maximum Value		2.65	0.280	3.56
Minimum Value		0.34	< 0.001	0.23

A weaker correlation between leach residue grade and grind size was also established. The weak correlation was from the lack of variability in the tested grind sizes (most tests were conducted around the optimal grind size). Table 17.5 shows the range of values for both P_{80} and residual gold grade.

Table 17.5: Leach Tests Parameters Range

Parameter	Unit	Maximum Value	Minimum Value
P_{80}	µm	121	30
Residual Gold Grade	g/t Au	0.34	0.03

The impact of grind size along with gold, arsenic and sulfur head grades on the residual gold grade was modeled via the following multivariate linear regression equation:

$$Au_{Tails(g/t)} = -0.0435 + 0.0349Au_{Head(g/t)} + 0.660As_{Head(\%)} + 0.0312S_{Head(\%)} + 0.000516P_{80(\mu m)}$$

The limits of application of the model in terms of As and S values are set based on the range of As and S grades of the samples tested. The limits of application of the model are as follows and anything below these limits is considered outside of the model's range:

- Au(g/t)/As(%) ratio > 8;
- Au(g/t)/S(%) ratio > 0.4.

When one or both of these ratios is not met, a corrected As or S value is calculated using the minimum ratio and the head Au grade as follows:

- If Au (g/t) / As (%) < 8, the As value to be used is As (%) = Au (g/t) / 8;
- If Au (g/t) / S (%) < 0.4, the S value to be used is S (%) = Au (g/t) / 0.4.

This formula has been used in the block model and open pit optimization process to calculate the gold recovery of each individual block based on chemical composition and grind size. The overall gold recovery can then be computed for a determined time period.

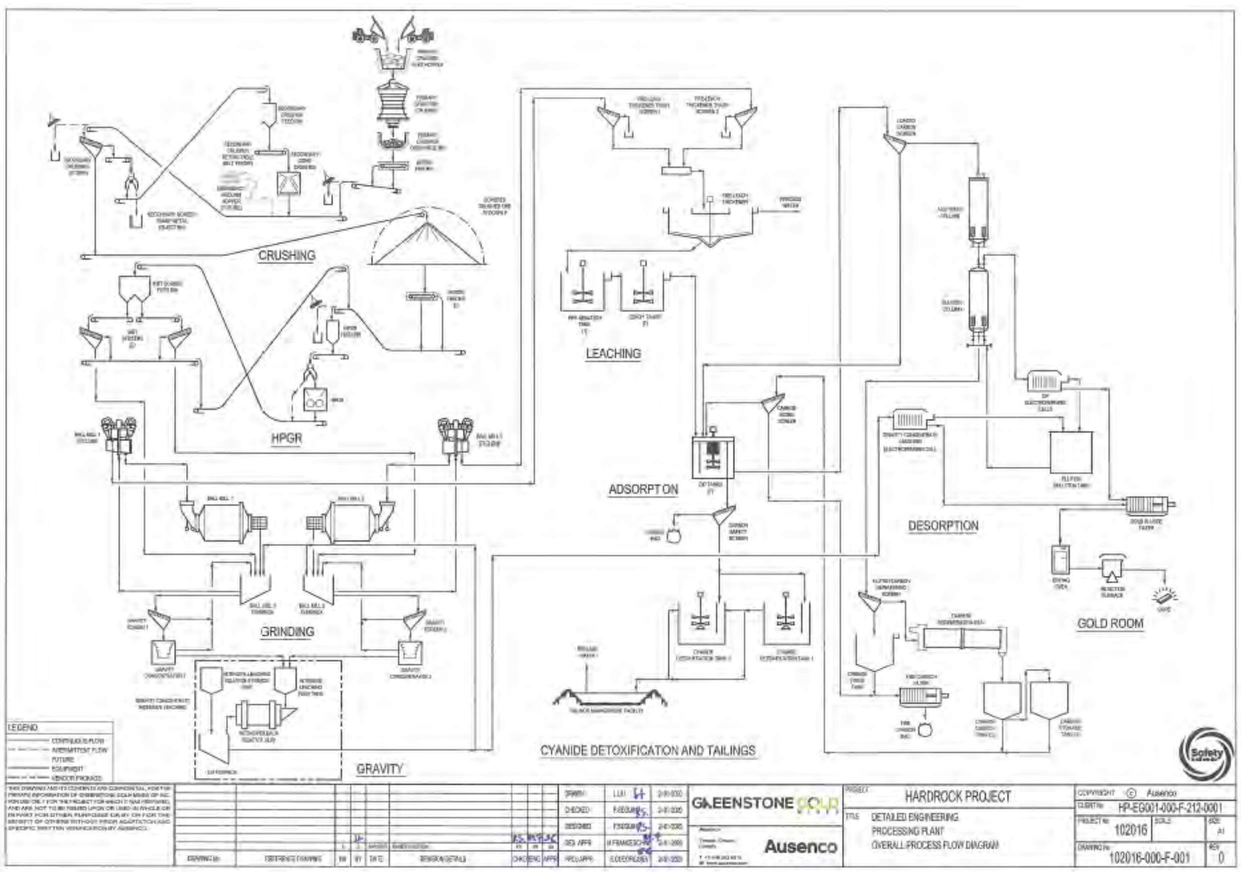
17.2 Flowsheet and Process Description

The gold recovery process for the Hardrock Project consists of a crushing circuit (primary gyratory and secondary cone), a HPGR and ball mill grinding circuit with gravity recovery, pre-leach thickening, cyanide leaching, carbon-in-pulp ("CIP") adsorption, elution and regeneration circuit, electrowinning and refining, cyanide destruction and tailings deposition.

The service areas include reagent preparation, compressed air, oxygen plant and sulphur dioxide storage and distribution. The water management system covers all the fresh, reclaim, process, potable, fire and gland water storage and pumping. An onsite sewage treatment plant will process domestic wastewater, discharging to the environment. Tailing reclaim and collected contact water will be used for process water, with excess contact water treated and discharged to the environment.

The simplified flowsheet is presented in Figure 17.2.

Figure 17.2: Hardrock Project Simplified Flowsheet



17.2.1 Crushing, Crushed Ore Storage and Reclaim Circuit

The objective of the crushing circuit is to reduce the size of the run-of-mine ore to the required particle size for the downstream HPGR and ball mill circuit. The crushing plant is a two-stage circuit consisting of a primary gyratory crusher and a secondary cone crusher. The crushing plant has a design availability of

67%. A 20% design factor has been selected such that crushing circuit equipment is sized to handle up to 2,025 t/h. The design factor is industry practice, providing extra production capacity to handle processing fluctuations due to changes in ore feed rate and ore hardness.

17.2.1.1 Primary Crushing

Run-of-mine ore is delivered by mine haulage trucks. The ore is dumped from the truck into a steel dump pocket that feeds the primary crusher. The pocket has two dump points and a live capacity equivalent to 1.5 times the haul truck payload. A rock breaker will be used to break oversized rocks. The 1,300 mm x 1,800 mm 450 kW gyratory crusher crushes the ore from a 1,000 mm top size (275 mm P_{80}) to a P_{80} of 120-160 mm product. The primary crusher operates with an open side setting (“OSS”) of 160 to 200 mm. The crushed ore falls into a steel discharge pocket with1.6 truckloads capacity and is reclaimed via an apron feeder. The main discharge and dribbles from the apron feeder are discharged onto the sacrificial conveyor that feeds the secondary crushing and screening circuit.

The primary crushing area is serviced by a dedicated compressed air system and a 40 t overhead crane. A dust collection system is installed to control dust emissions. The primary crusher is designed for maintenance from the top down rather than through the discharge pocket.

An electromagnet and metal detector are installed on the primary crusher sacrificial conveyor to prevent tramp iron from entering the secondary cone crusher. The ore contains magnetite in sufficient quantities that would overwhelm a standard magnet. The metal detector is used to detect spikes in magnetic susceptibility from tramp metal, which will then activate the electromagnet to remove the tramp. This feature is utilized for all of the tramp removal systems.

17.2.1.2 Secondary Crushing and Screening

The secondary crusher is a 950 kW standard cone crusher with a 45-60 mm closed side setting (“CSS”). The secondary cone crusher is installed in closed-circuit with a double deck screen to control the top size feeding the HPGR. Secondary crusher screen undersize is conveyed to the covered crushed ore stockpile.

The 4,250 mm x 8,500 mm double deck secondary crushing screen is fitted with a top deck 75 mm closing screen opening, and bottom deck with 50 mm closing screen opening. Combined oversize from both decks is returned to the secondary cone crusher. The crushing circuit produces a final crushed product with a 50 mm top size and a 35 mm P_{80}. Combined screen oversize flows onto the secondary screen oversize discharge conveyor and transfers to the secondary screen oversize return conveyor which feeds a mass flow bin with 17 minutes retention time. A second tramp metal electromagnet is installed before the secondary screen, and a metal detector-activated flop gate on the screen oversized recycle to protect the secondary crusher. In case tramp metal is not removed successfully, the secondary crusher retractable belt feed conveyor is equipped with a metal detector which will retract the chute and deposit the tramp steel into a separate chute. These measures prevent tramp iron from entering the secondary cone crusher.

There is an allowance for an emergency hopper and feeder installation at the discharge of the secondary crusher on the secondary screen feed conveyor to reclaim material using a surface loader, if necessary in the future. The secondary crushing area is serviced by a dedicated air compressor, a 55 t capacity overhead crane, a dust collector and a sump pump. A dust collection system is installed to control dust emissions. The screen area is serviced by a 90 t capacity overhead crane that is also used to service the HPGR. The secondary crushing recirculation and stockpile feed conveyors are equipped with belt scales to monitor throughput.

17.2.1.3 Crushed Ore Stockpile and Reclaim

The crushing circuit product is stored in a 21,196 t live capacity stockpile which provides 20 hours of operation. The stockpile is located North of the secondary crushing and the process plants. There is a single stockpile reclaim tunnel with three apron feeders located in a reinforced concrete tunnel underneath the stockpile. These apron feeders feed the crushed ore stockpile (“COS”) reclaim conveyor which discharges into the HPGR feed bin located inside the HPGR building. Cartridge type dust collectors are installed in the transfer chutes between the apron feeders on the skirt boards of the COS reclaim conveyor. Spile bar isolating systems are installed on each of the apron feeder reclaim hoppers, to isolate the apron feeders for maintenance.

The reclaim area is serviced by a dedicated air compressor and a sump pump. A reclaim hopper is installed on the COS reclaim conveyor to allow the tunnel area to be cleaned using a bobcat. Monorails and hoists are available in the tunnel to manipulate spile bars and their insertion / extraction tool and for maintenance purposes. On the COS reclaim conveyor, a scale is installed to control throughput as well as a metal detector to detect any tramp steel still present.

17.2.2 HPGR/Grinding and Gravity Recovery Circuit

The HPGR/grinding circuit crushes and grinds ore to the optimal grind size to maximize gold recovery in the leach and carbon-in-pulp circuit. The gravity recovery circuit removes gravity recoverable gold in the grinding circuit in order to decrease the load on the leach and CIP circuits.

The grinding and gravity circuit consists of two parallel operating lines each consisting of a wet screen (fed from one bin - HPGR crushed product), ball mill and gravity concentrator in closed circuit with cyclones. The gravity products from both concentrators are combined in a single gravity concentrate leaching unit.

At the design plant availability of 92%, the nominal circuit throughput is 1,223 t/h. A 10% design factor is used for the sizing of the equipment such as pumps, thickener, tanks, etc. The design factor accounts for the process fluctuations and upsets, ensuring that mill throughput is met.

17.2.2.1 High Pressure Grinding Rolls (HPGR)

A detailed comminution trade-off study recommended a two-stage crushing circuit followed by HPGR and ball milling circuit over other typical comminution flowsheets such as crushing followed by semi-autogenous ("SAG") milling and ball milling, to reduce throughput risk and increase energy efficiency from the high hardness levels measured in testing.

Wet screen oversize is recycled back to the COS reclaim conveyor, which discharges a blended product into the HPGR feed bin (mass flow bin equipped with a slide gate for isolation). A belt feeder reclaims ore from the HPGR feed bin and feeds the HPGR weigh bin, located above the HPGR. Belt feeder speed is controlled in order to ensure that the weight bin choke-feeds the HPGR. The HPGR is equipped with two, 2,650 kW motors for a total of 5,300 kW. The HPGR roll dimensions are 2.2 m in diameter by 2.0 m in length and have a rotating speed of 22 rpm. The HPGR discharge falls onto the wet screen feed bin conveyor and then into the wet screen surge bin feed (mass flow bin equipped with slide gates for isolation) that divides the HPGR product between two double-deck screens. The HPGR screens are double deck 3,600 mm wide by 7,300 mm long banana type. Process water is added in the pulping box ahead of the HPGR screens, sprayed on the HPGR screens and added in the HPGR screen undersize chute to reduce dust emissions and help flake deagglomeration. The top screen panels have 12 mm x 28 mm apertures and the bottom screen panels have 27 mm x 6 mm apertures. The screen undersize is about 80% passing 3.5 mm and falls into their respective ball mill pump box.

The wet screen surge bin has 17 minutes of retention time. The building is maintained at minimum of 10°C during cold weather operation. Modeling has shown the bin and its contents are always above freezing temperature.

The wet screen oversize is recirculated on the wet screen oversize transfer conveyor and the wet screen oversize return conveyor. The wet screen oversize return conveyor is discharged back on the COS reclaim conveyor to be crushed again in the HPGR. The HPGR circulating load is expected to be nominally 85%, with a design value of 110%.

The HPGR is installed in a dedicated building with the secondary crusher screen while the grinding mills and HPGR screens are installed in the process plant building. A 90 t capacity overhead crane is installed to service the HPGR and the secondary screen in the HPGR building and a 5 t capacity overhead crane to service the COS reclaim conveyor head pulley, electromagnet and HPGR feed bin. A sump pump is also installed in the HPGR / Secondary Screen area. The wet screen oversize return conveyor is located in a heated gallery to minimize the risk of freezing during cold weather months. Two dust collection systems are installed in this building, one for the Secondary Screen and one for the HPGR. In the processing plant building, a scrubber and a 50 t overhead crane are installed to service the HPGR wet screens area and grinding area.

17.2.2.2 Grinding

Undersize from each HPGR screen product is discharged to their respective ball mill pump box. Each pump box is equipped with two slurry pumps, one pump feeding the ball mill cyclones and one feeding the gravity circuit. Approximately 60% of the fresh feed is pumped to the gravity circuit.

The cyclone overflow P_{80} ranges from 72 µm to 90 µm. The ball milling circuit recirculating load is estimated at 300%, with a design value of 350% for pump selection. The cyclones are 600 mm diameter installed in a radial distributor. There will be seven cyclones operational per mill at a pressure of 105 kPa. Each distributor also has two installed spare cyclones.

The cyclone overflow feeds the pre-leach thickener trash screen while the underflow (approximately 75% solids) is directed to the ball mill feed chute. Lime is added to the ball mill feed to raise the slurry pH to between 10 and 11. The grinding mills are twin pinion ball mills equipped with motors totaling 10,500 kW per mill. Both mills are 6.7 m in diameter (inside liners) by 12.3 m in length (EGL). The ball mills have discharge trommel screens to remove scats. The ball mill discharge flows into the ball mill pump box where it is combined with the HPGR discharge slurry and pumped to the cyclones for classification. The plant can be operated at a lower throughput, by operating only one ball mill and one HPGR screen, slowing the HPGR RPM to accommodate.

A single liner handler can be used for the liner changes in either ball mill. Each mill has its own ball kibble for loading 65 mm diameter balls into the mill, lifting frame and inching drive. The hydraulic jacking unit, including drive powerpack and bolt removal tools are shared between the two mills. A dedicated sump pump is installed in each ball mill area.

17.2.2.3 Gravity Concentration

The gravity feed pump transfers a portion of the cyclone feed to the gravity circuit to recover gravity recoverable gold. Two gravity screens and two gravity concentrators are installed to process the material.

The vibrating gravity feed screen prevents particles coarser than 3.36 mm from entering the gravity concentrator. The screen oversize flows into the ball mill pumpbox. Screen undersize flows to the gravity feed chute onto each gravity concentrator. A by-pass line is installed on the gravity feed chute to the gravity concentrators to operate when the concentrators are transferring concentrate to intensive leaching. After each cycle, the gold concentrate is flushed from the concentrators and transferred to the gravity concentrate intensive leaching circuit. The gravity concentrator tailings flow to the ball mill pump box.

Process water is used to flush the concentrate and antiscalant is added to the water stream to avoid scale build-up. A 5 t overhead crane is installed to service the gravity screens, with each gravity concentrator having a dedicated monorail.

17.2.2.4 Gravity Concentrate Intensive Leaching

The gravity concentrate from both gravity concentrators is transferred to the single gravity concentrate intensive leaching circuit. This system is a packaged unit consisting of a feed tank, a drum leach reactor, a solution storage tank and a transfer pump.

Both the gravity concentrators and gravity concentrate leaching equipment are secured in a fenced area with controlled access and security cameras (not linked to the process camera network).

The gravity concentrate is leached in the reactor. A 98% dissolution efficiency is expected. The gravity leach tailings are returned to the ball mill pump box while pregnant solution is fed to a dedicated electrowinning cell.

17.2.3 Pre-Leach, Leach and Carbon-In-Pulp

The objective of the pre-leach, leach and carbon-in-pulp (“CIP”) circuit is to dissolve gold from the ground ore, adsorb it onto activated carbon and transfer the loaded carbon to the elution circuit. The circuit is made up of a pre-leach thickener, a series of leach tanks (one pre-leach and seven leach tanks) followed by seven CIP tanks.

17.2.3.1 Pre-Leach Thickening

The cyclone overflow from each ball mill circuit feeds a dedicated trash screen located above the pre-leach thickener. The screen undersize gravity flow to the pre-leach thickener feed box. Screen oversize is collected in a bunker which is periodically cleaned out.

The 50 m diameter thickener is located outdoors; the tank underside is cladded and heated. The thickener increases the slurry density from 35 to 55% solids and recycles water to the grinding circuit. A very high molecular weight and slightly anionic polyacrylamide flocculant is added at a nominal dosage of 15 g/t and design dosage of 30 g/t (grams of flocculant per tonne of dry solids) to promote sedimentation. Thickener underflow is pumped to the leach circuit and sampled prior to leaching.

17.2.3.2 Leach Circuit

The leach circuit consists of one agitated pre-aeration tank and seven agitated leach tanks located outdoors. All tanks are dimensionally equal (18.6 m D x 23.5 m H) and have the same level of agitation.

Slurry transfers from one tank to the next by overflow through an upcomer. Any tank can be bypassed if maintenance is necessary. The leach tanks are equipped with 185 kW agitators. The total residence time in the leach circuit is 28 hours and an additional four hours is available in the pre-aeration tank. Pre-aeration provides passivation of reactive sulphide minerals, minimizing their impact on cyanide consumption.

Lime is added in the pre-aeration tank, the second and the fourth leach tank to readjust the pH level between 10.5 and 11.0, as required. A 30% sodium cyanide solution is added to leach tanks Nos. 1, 3 and 5 to ensure an initial concentration of 0.5 g/L NaCN and tapering off as determined by leaching performance. A cyanide analyzer is installed in the leach circuit to measure cyanide concentrations and control cyanide addition. Oxygen from the on-site oxygen plant, is injected to reach the targeted 15 mg/L concentration of dissolved oxygen. The leach discharge flows by gravity to the CIP circuit.

Leach tanks and associated equipment are serviced by mobile crane as required. Drive-through access is provided along both sizes of the leach containment area, which is hydraulically linked with the pre-leach thickener containment area.

17.2.3.3 CIP

The leach circuit discharge flows into the CIP launder located above the CIP tanks. The circuit is composed of seven CIP tanks (six operational) with a total retention time of 1.5 hours. The seventh tank is included to ensure the residence time is maintained when one tank is not in operation. The CIP tanks are located indoors.

The CIP circuit is designed and operated in carrousel mode, which allows the CIP tanks to be installed on the same level and to be of the same dimensions (7.0 m in diameter by 12.0 m in height). Slurry feed and discharge positions are rotated to ensure a counter-current movement between slurry and carbon, without transferring carbon from one tank to another. The slurry passes through all the CIP tanks using the combined carbon retention screen and agitator mechanism for pumping between tanks. Carbon concentration is maintained at 50 g/L in each operational tank. After it reaches the last tank, the slurry gravitates to the carbon safety screen to recover fine carbon particles from the tailings and minimize associated gold losses.

The carbon safety screen undersize flows into the CIP tailings pump box and is pumped to the cyanide destruction circuit. The pump box is equipped with two slurry pumps (one in operation and one stand-by).

Each day, the lead CIP tank is taken off-line and 50% of the tank contents is transferred to the loaded carbon screen via a recessed impeller centrifugal pump. One carbon transfer is planned daily and is done in two hours. The carbon batch is 10 t. The next day, the remaining 50% tank contents are transferred to the loaded carbon screen via a recessed impeller centrifugal pump.

The loaded carbon screen is located in the acid wash and elution area. New carbon is added into the circuit after being screened on the carbon sizing screen located after and below the reactivation kiln and directed to the correct tank by the carbon distribution box.

17.2.4 Cyanide Destruction and Final Tailings

The cyanide destruction and tailings area comprise the equipment required for tailings detoxification, final tailings collection and pumping to the Tailings Management Facility ("TMF").

17.2.4.1 Cyanide Destruction

The cyanide destruction circuit consists of two agitated reactors, operating in parallel. The tailings from the CIP circuit are pumped to a sampler on top of the cyanide destruction tank where the feed to the circuit is sampled.

The SO_2/air process is used to reduce weak acid dissociable cyanide concentration (CN_{WAD}) in the tailings slurry to less than 10 mg/L. Liquid SO_2 and gaseous oxygen are injected at the bottom of the reactors and dispersed by an inverted cone diffuser. Dissolved oxygen is measure in each reactor, with the output used to control the oxygen addition to maintain a concentration of 4 mg/L. Copper sulfate is used as a catalyst for the reaction and is added to the slurry in a liquid form at a 20% concentration. About 20 mg/L of Cu ions are required nominally (40 mg/L design). Lime is added to control the pH of the reaction to approximately 8.5.The CN_{WAD} concentration is measured in the feed and SO_2 is added at a ratio between 4.5:1 and 5:1. The CN_{WAD} concentration is also measured in each reactor to ensure the target of 10 mg/L is achieved. The residence time required achieve the desired extent for the reaction is 60 minutes. Total retention time of 138 minutes is provided. If one tank is offline, desired discharge limits can still be met with available 60 minutes retention time in a single tank. Cyanide destruction is carried out at a slurry density of approximately 55% solids to minimize the volume of water pumped to the TMF.

17.2.4.2 Final Tailings

The cyanide destruction reactors overflow to the tailings pumpbox. The pumpbox also collects various tailings streams from the process, including sump pumps. Reclaim water is added to the pumpbox when the plant throughput is low to maintain adequate fluid velocity in the tailings pipeline.

The tailings pumpbox discharge is pumped by two parallel pump trains of three pumps installed in series (one train in operation and one train on stand-by) pumping the tailings to the TMF.

The tailings pipeline is separated into two sections to fulfill the pressure requirements, with two pumps in series at the process plant. A leak detection system will be installed on the HDPE pipeline and there will be a spigotting system at the TMF end to distribute tailings.

17.2.5 Acid Wash, Elution and Carbon Regeneration

The objective of the circuit is to elute or strip adsorbed gold from the carbon into a solution feeding the electrowinning circuit, where gold is deposited onto cathodes, washed off and smelted. Eluted or stripped carbon is regenerated and returned to the CIP circuit. Fresh carbon is conditioned and mixed with the regenerated carbon. The circuit is batch operated with a 10 t capacity.

17.2.5.1 Carbon Acid Wash and Elution

Loaded carbon from the CIP circuit is pumped to the loaded carbon screen located above the acid wash column. When a carbon batch is transferred, the screen oversize containing the loaded carbon flows into the acid wash column. Screen undersize gravity flows to the CIP feed launder.

The acid wash step removes scale and some adsorbed metals that collect onto the activated carbon during the adsorption process. A dilute hydrochloric acid solution (3% HCl) is circulated through the column to remove these impurities. The acid wash waste solution is pumped to the cyanide detoxification distribution box.

The acid washed loaded carbon is transferred to the elution column where adsorbed gold and other metals are stripped using the pressure Zadra process. A heated diluted caustic (1.0% NaOH) and cyanide (0.1% NaCN) solution is prepared in the elution solution tank and is circulated through the column to strip the carbon at a temperature of 140ºC. Solution is heated through a natural gas heater. A carbon elution cycle is completed within eight to twelve hours.

17.2.5.2 Carbon Regeneration

Eluted carbon is transferred to the eluted carbon dewatering screen. The dewatered carbon feeds the regeneration kiln. The water and the fine carbon that pass through the screen are recovered in the carbon water tank. The water is reused as transfer water while the carbon fines are filtered using a filter press and is collected into fine carbon bags.

The natural gas-fired kiln heats the carbon to a temperature of 750°C. At this temperature and under a slightly oxidizing atmosphere, fouling organics are removed. The reactivated carbon exits the kiln, is quenched in a tundish and flows over sizing screen where fine carbon is removed to the carbon fines system and screen oversize is transported into one of four carbon quench tanks (20 t capacity). Newly conditioned carbon is used to make-up for fine carbon losses and is agitated in a separate tank prior to addition to the CIP. When required (usually every 2 days), a carbon batch is transferred to the CIP circuit.

17.2.6 Electrowinning and Smelting

Gold from the pregnant solutions (gravity concentrate leaching and elution) is recovered onto the cathodes in the electrowinning circuit. The electrowinning gold sludge is recovered and smelted into doré bars in the refinery.

17.2.6.1 Electrowinning

Pregnant solution from the gravity concentrate leaching is pumped to its dedicated electrolyte tank located in the electrowinning area. The electrolyte solution is circulated between the gravity electrowinning cell and the electrolyte tank.

The pregnant solution from the elution circuit is transferred to a flash tank to cool the solution prior to being split between two electrowinning cells operating in parallel.

Gold sludge is washed from the cell cathodes into the cathode wash pumpbox and is pumped to the plate and frame sludge filter press. The barren solution from the cells flows by gravity to the elution circulation tank. The sludge filter filtrate is recirculated to the sludge settling tank and recirculated back to the sludge filter.

17.2.6.2 Smelting

The gold sludge is dried in the oven in preparation for smelting. Dried sludge is transferred to the mixer where refining fluxes are added. The mixture of sludge and fluxes is fed to the gas-fired furnace where the slag material is separated from the gold as the doré bars are poured.

17.2.7 Gas and Reagents

The process plant includes a compressed air system and oxygen supply system as well as various reagents offloading, preparation and storage equipment.

17.2.7.1 Compressed Air

The compressed air system is composed of three air compressors (two operating, one on stand-by). The compressed air is stored in two air receivers with an air dryer between them. Compressed air is produced at 690 kPa. A first distribution loop provides the instrument air while a second delivers compressed air to the various equipment requiring compressed air (filters, vents, dust suppression systems, etc.).

17.2.7.2 Oxygen (O_2)

The oxygen requirements for leaching and cyanide destruction are met by an on-site, over-the-fence vacuum pressure swing adsorption (“VPSA”) plant. The VPSA plant is a packaged unit installed outside the plant, adjacent to the leach tanks. Two liquid oxygen tanks (total 100 t or 4 days consumption at design rate) are also installed on site as a back-up source of oxygen.

17.2.7.3 Cyanide (NaCN)

Cyanide is delivered in isotainers (or equivalent) containing 18 t of solid NaCN briquettes. Up to four isotainers can be stored in the process plant, providing about nine days of storage at 27,000 t/d plant throughput. Water is added to the preparation tank and the solution is circulated between the tank and the isotainer until complete dissolution of the briquettes. The preparation tank is equipped with an immersion heater to aid in mixing by warming the mix solution. The water addition is controlled in order to produce a 30% NaCN concentration solution. The live volume of the preparation tank is 57 m^3, which is the equivalent of an isotainer batch.

The storage tank is 57 m^3 and contains one mixed isotainer batch; it is equipped with two distribution pumps (one in operation and one on stand-by). Cyanide solution is distributed to the gravity concentrate leaching circuit, the leach circuit and the elution circuit.

17.2.7.4 Caustic (NaOH)

Caustic is delivered by 28 t bulk tanker truck, in liquid form at a 50% solution concentration. The solution is transferred to the storage tank, which can hold the contents of approximately one and a half tanker trucks. The onsite storage capacity is a live volume of 26 m^3 and is sufficient to last about one week at 27,000 t/d plant throughput. Since the caustic users are all intermittent (cyanide preparation, elution circuit and gravity concentrate leaching), only one distribution pump is required.

17.2.7.5 Quicklime (CaO)

Pulverized quicklime is received in bulk. It is transferred to a 263 t capacity silo which provides four days of storage at 27,000 t/d plant throughput. Dry lime is fed to the slaker via a screw feeder. Water is added to the slaker to produce hydrated lime slurry. The hydrated lime production rate closely matches the consumption rate to ensure the slaker operates as continuously as possible.

The slaker discharges onto the vibrating grits screen that removes oversize particles from the lime slurry. The slurry gravity flows into the pumpbox where it is further diluted to produce 20% lime slurry. The lime pumpbox is equipped with two transfer pumps (one operating, and one stand-by). The transfer pump pumps to a storage tank from which there are two distribution pumps (one operating, and one stand-by). The lime distribution header is in closed-loop with the storage tank. Hydrated lime addition points include the ball mill circuit, the leach circuit and the cyanide destruction circuit.

17.2.7.6 Flocculant

Flocculant is received in a powder form in bulk bags. The bags are unloaded to a hopper and a screw feeder transfers the flocculant to an eductor to the mix tank to a 0.5% concentration solution. The storage tank is located below the mixing tank and the transfer is done via a valve once the mixing is complete. The flocculant is metered to the thickener by two dosing pumps (one operating, and one stand-by).

17.2.7.7 Hydrochloric Acid (HCl)

Hydrochloric acid is delivered by 22 t bulk tanker trucks as a 33% concentrated solution. It is unloaded to a storage tank that has the capacity to hold one and a half truck delivery and holds about 12 days storage at 27,000 t/d plant throughput. The HCl is distributed to the acid wash and elution circuit using a single pump.

17.2.7.8 Copper Sulfate ($CuSO_4{\cdot}5H_2O$)

Copper sulfate pentahydrate is received in 500 kg bulk bags. Bulk bags are lifted by hoist into the hopper equipped with a bag breaker from where it flows to the mixing tank where water is added to make a 20% concentration solution. The copper sulfate storage tank is located adjacent to the mixing tank and the transfer is done via a transfer pump once a batch is complete. Both tanks have a live capacity of 19 m^3, about 23 hours storage at 27,000 t/d plant throughput· Two distribution pumps are installed (one operating, and one stand-by) to transfer copper sulfate to its addition point in the cyanide detoxification distribution box.

17.2.7.9 Sulphur Dioxide (SO_2)

Liquid Sulphur dioxide is delivered by a 30 t bulk tanker truck. It is transferred to two 50 t capacity pressure vessel storage tanks equipped with a padding air system to maintain the sulphur dioxide in liquid form. The two storage tanks provide approximately 8 days of storage at 27,000 t/d plant throughput. Sulphur dioxide is metered to the cyanide detoxification circuit in gas form. A vent system is included in the sulphur dioxide addition piping to vent any vaporized sulphur dioxide into the cyanide detoxification tanks.

17.2.7.10 Antiscalant

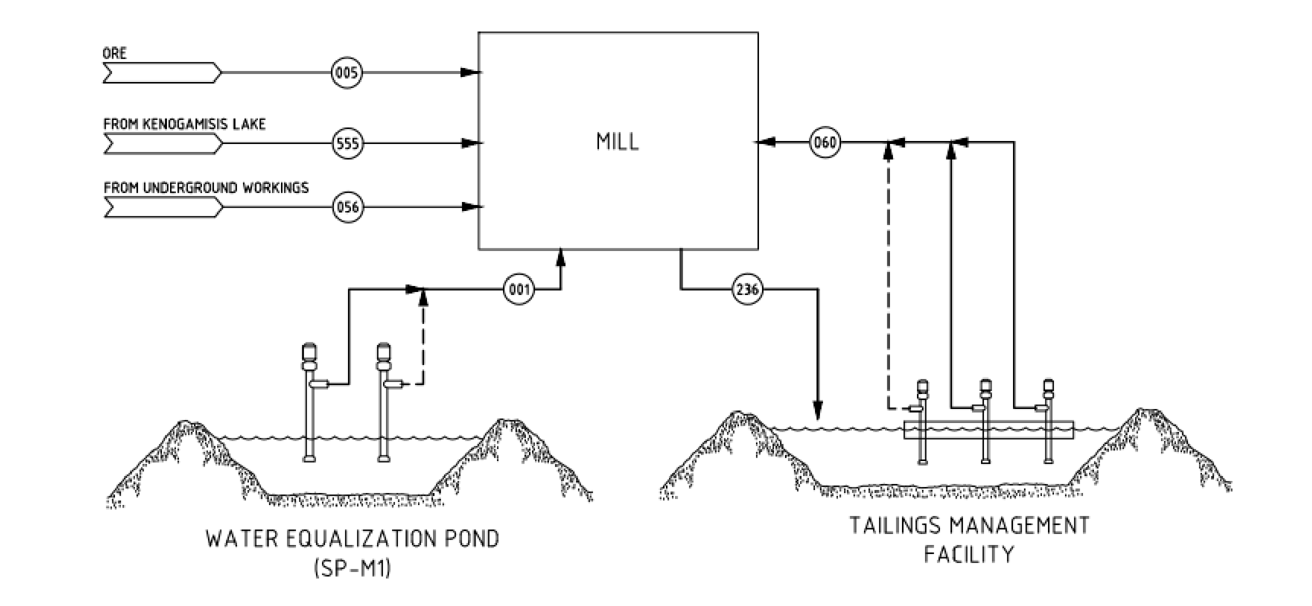
Antiscalant is received in a liquid form in 1 m^3 ISO containers. Containers will be directly fitted with pumping systems from where it is pumped to the process. Antiscalant is added in the gravity concentrators' water lines, the elution solution tank, the process water tank and the reclaim water tank.

17.3 Mass and Water Balance

A detailed mass balance was developed for the process plant to track all flows in and out of the process equipment.

A comprehensive water balance was developed to track all fresh and waste water flows to ensure that each type of water is managed. The process plant requirements in fresh water will come from the underground workings collected in the water equalization pond M1. No water is planned to be withdrawn from the Kenogamisis Lake. Most of the water required for the process plant operations is recycled from the TMF. The TMF is fed by the slurry discharged from the process plant that contains a significant amount of water, Figure 17.3 outlines the water balance results calculated for the process plant.

Figure 17.3: Hardrock Project Process Plant Water Balance



Title: Hardrock Project Process Plant Water Balance

Author: Ausenco
Date: 19/09/05

Stream No.	001	005	056	060	236	555
SOLIDS	0.0	1226.0	0.0	0.0	1226.0	0.0
SOLUTION	200.0	1057.0	8.6	957	1057.0	0.0
SLURRY	200.0	2283.0	8.6	957	2283.0	0.0
% SOLIDS	0.0	97.0	0.0	0.0	53.7	0.0
S.G.	1.0	2.7	1.0	1.0	1.5	1.0
SLURRY	200.0	471.0	8.6	957	1490.0	0.0

17.4 Process Equipment

Process plant equipment was designed using the information obtained from the mass balance and design criteria. For most equipment, at least three budget quotations were requested from suppliers. A bid evaluation was then conducted including a technical and a commercial evaluation of the proposed equipment. During the technical evaluation, the process, mechanical, electrical and other relevant aspects were analyzed to determine if the equipment complied with the specifications. The commercial evaluation, in turn, was made considering the budget price of the equipment, the transportation cost and the delivery lead time. The selected bidder was determined for each piece of equipment by combining the technical and commercial recommendations.

17.5 Cyanide Management

The International Cyanide Management Institute ("ICMI") was established for the purpose of administering the "International Cyanide Management Code for the Manufacture, Transport and Use of Cyanide in the Production of Gold" (ICMI Code), and to develop and provide information on responsible cyanide management practices and other factors related to cyanide use in the gold mining industry. Throughout the design of the process plant, ICMI Code requirements were considered, to allow a third party to confirm compliance if necessary.

17.6 Power Requirements

The power requirement was determined for the process plant using the power demand indicated by the selected equipment supplier. The annual power consumption for the mill and ancillaries is estimated to be 299.1 GWh at 27,000 t/d.

17.7 Plant Layout

17.7.1 Process Plant Location

The process plant site is located southwest of the open pit. The site main entrance is on the South west end to allow for a connection to Highway 11. The natural gas power plant location is on the eastern side of the process plant and other infrastructures in consideration of the prevailing winds.

17.7.2 Building Architecture

All buildings designed for the process plant, including crushing and transfer towers, are planned to be covered with pre-assembled insulated wall and roof panels to reduce construction time and workforce on site.

A National Building Code study was conducted to ensure appropriate emergency and safety requirements are met in terms of design and construction methods (fireproof staircases, etc.). The National Building Code study also confirmed that the requirements for the buildings which house workers were met (toilets, wall composition, travel spaces, office arrangements, etc.).

17.7.3 Heating, Ventilation and Air Conditioning (HVAC)

The mill building, crushing areas, and the ore storage dome tunnel and administration building are heated with a water/glycol mix system having 50% / 50% proportions to prevent freezing. The glycol mix temperature will be 85°C for the heating distribution circuit with a 55°C return temperature. The system is designed to heat the functional spaces and the heating load of the ventilation created by thirteen make-up air units (5 for the process plant, seven for the crushing, COS and HPGR building and one for the ore reclaim tunnel).

A primary/secondary system type is selected. This type of system allows a separation of the heat generation equipment installed in the power plant from the heat distribution equipment in the process plant (circulation pumps, manifolds expansion tank, pressurized glycol tank, etc.). The estimated capacity of the water/glycol heating system for the process plant is 33,000,000 BTUH (9,663 kW).

The ventilation system was designed considering the different functions of the areas in the mill building; four functional areas were created to meet the specific needs. These areas are the following:

- Ventilation, heating and cooling of the mill building work areas;
- Ventilation, heating, and cooling (refrigerant based) of the offices;
- Ventilation and cooling of compressor and other mechanical rooms;
- Cooling of electrical rooms.

17.7.4 Fire Protection

The fire protection water reserve will be in the freshwater tank located at the process plant. A dedicated portion of this tank is exclusively available for fire protection water, sized according to the largest consumer building.

Water sprinkler systems will be installed for the administration offices, the electrical and mechanical shops, as well as the process plant areas that have been identified as a fire hazard (conveyors, hydraulic units, etc.). Fire protection cabinets containing water hoses and portable extinguishers will also be installed throughout the plant. The electrical room will be protected by an inert gas system. For unheated conveyor galleries, fire protection will be provided by dry-pipe sprinkler system.

17.7.5 Electrical Distribution

Power to Process Plant shall be supplied from onsite Power Plant at 13.8 kV.

The Power Plant will consist of seven gas fired engine generating units in N+2 configuration. With seven units of 9.7 MW each, the maximum power generation capacity is 48.5 MW, one unit hot standby and one unit may be in maintenance. The availability of Power Plant will be 100% excluding a major fault at main supply switchgear or power plant control system failure, both of which are very rare.

Breakers at Power Plant Main Switchgear will supply power (1,200 A) to 13.8 KV/4.16 KV and 13.8 KV/600 V dry type stepdown transformers installed inside modular, pre-fabricated and pre-assembled Electrical Rooms (E-rooms). The E-rooms will house the medium and low voltage MCCs, lighting & services transformers, instrumentation transformers, VFDs, soft-starters, etc. to distribute power to the Dry End equipment including Crushers and HPGR, Ball Mills, Process Plant, etc.

The Power plant will also supply 13.8 kV Overhead Power Lines to outlying pumping stations, shops and service buildings.

An Isolation Transformer will be provided for overhead line feeders to provide a grounded wye on the secondary, isolating it from the grounded wye system of power plant for grounding detection and coordination.

17.7.6 Control System

The process control system will be a dedicated, microprocessor based, scalable, deterministic, control system with the ability to monitor and control centrally and remotely. It will be a PLC/SCADA solution and referred to as the Process Control System ("PCS"). Touch screens of 7" size and larger are used for local operator interface (HMI).

The plant control network architecture will use Ethernet protocol for input/output ("I/O") control, Modbus / TCP or Ethernet IP for motor control, switchgear and connection to vendor packages. Communications between the PLC and HMI, and PLC to PCS will be Ethernet TCP/IP. Communications to MCC's and VFD's will be Ethernet based. PLC's supplied with package equipment, fully interface with the PCS, using standard protocols, for the purpose of monitoring and remote control from the PCS operator's consoles.

Remote I/O wiring will be available through project approved terminal blocks. If I/Os need to be powered, this shall be done through terminal boards with individual fuses. All package Vendors will recommend a list of monitor and control interface signals for communicating with the Owner's Process Control System, as well as a list of the hardwired signals required between vendor PLC and plant PCS. Termination points and/or data link devices will be provided for easy Owner hook-up.

All instruments and control equipment information connected to local PLC’s will be available through a communication link to the PCS. All controls and alarm switches will be ‘Fail Safe (i.e., an abnormal condition shall cause a loss in output signal). Upon loss of power, control circuits and alarms will go to the “fail safe” condition. Solenoid valves and actuating relays will be normally energized and shall de-energize on protective action or alarm.

18. PROJECT INFRASTRUCTURE

18.1 General

This section describes the infrastructure and service facilities required to support the Project's mining and processing facilities such as power generation, roads, water and tailings management, camp, mine equipment maintenance shops (truck shop), warehouses, information technology ("IT") systems, laboratories and offices, diesel fuel and the natural gas distribution pipeline.

18.2 Tailings Management

Wood performed specialized geotechnical and hydrologic engineering services for the design of the Tailings Management Facility ("TMF") for the Project, including geotechnical site investigations, tailings deposition planning, and design of the tailings dams and ancillary hydraulic structures.

Tailings impoundment at the TMF is affected by construction of mainly perimeter embankment dams. Embankment dams will be primarily constructed out of mine rock. The upstream slopes of the dams will be provided with low permeability compacted glacial till core keyed into relatively low permeability foundation soils. Where the low permeability foundation units are deeper and/or water table is too close to surface, a Cement Bentonite slurry wall will be provided keyed into the low permeability foundation soils. The till core and foundation slurry wall serve as a low permeability element to mitigate dam body and foundation seepage.

The TMF site was selected based on a balance of environmental, social, economic and operational risk considerations. Prior to construction of the TMF, Goldfield Creek ("GFC") will be diverted around the north side of the ultimate TMF footprint and into a permanent channel designed to provide fisheries compensation. The TMF is set back from other waterbodies by an environmental buffer width of approximately 125 m.

The TMF dams have been designed to meet the requirements of the Lakes and Rivers Improvement Act (MNR, 2011) and the Canadian Dam Association guidelines (CDA, 2014). Filter and transition zones are provided downstream of the till core and between the foundation soils and rockfill embankment to protect against piping of fines into the rockfill due to seepage forces. The rockfill embankment is constructed of geochemically benign mine rock.

Brief descriptions of each component considered during the TMF design are provided in the subsections below. For detailed information, please refer to the following documents.

- 2019 Geotechnical Investigation Report for Tailings Management Facility and Construction Material Borrow Areas – December 2019;
- Goldfield Creek Diversion Dyke Design Report – November 2018;
- Tailings Management Facility Design Report – December 2018;
- Tailings Management Facility Dams Till Core Deformation Analysis – April 2019; and
- Varved Clay Liquefaction Analyses memo- September 2019.

18.2.1 Geotechnical Subsurface Investigations

In the early stages of the Project assessment, geotechnical investigations were carried out in the TMF area by Stantec for environmental and hydrogeological baseline data. Wood carried out the subsurface characterization of the TMF dam footprint geotechnical investigations from 2014 to 2019. The investigations also included search for sand and gravel filter material and glacial till core required for the dam construction.

Generally, the boreholes penetrated 15 m into bedrock before termination. Piezometers/monitoring wells were installed in the boreholes at various depths to facilitate groundwater monitoring and testing. Standard Penetration Tests ("SPT") were carried out in the overburden at regular depth intervals in the boreholes. Soil samples were collected from the boreholes and test pits. The soil samples were tested in a geotechnical laboratory for various index properties such as particle size distribution, Atterberg limits and moisture content, monotonic simple shear, cyclic direct simple shear, consolidation, and direct shear testing were completed on undisturbed soil samples (extracted using thin wall sampling tubes) of the fine-grained overburden soils.

Overburden hydraulic conductivity values for soils in the TMF dam footprint were estimated using results from rising head slug testing on screened intervals of foundation soil from selected piezometer/monitoring wells. Bedrock hydraulic conductivity values were estimated using rising and falling head and constant head test methods from continuous single packer testing in bore holes in the TMF area.

The Quaternary stratigraphy at the TMF consists of four major stratigraphic units based on lithology, depositional environment, and relative stratigraphic position. In order from oldest to most recent these are: Glacial Till, Glaciofluvial (sand and gravel), Glaciolacustrine (sands, silts, clays), and recent deposits (Eolian, Organics, Fluvial). Isolated pockets of sand and gravel were also encountered below the glacial till in the southwest dam close to the downstream toe that were likely deposited prior to the last glaciation. Glaciolacustrine deposits of silt interbedded with thin clay layers (typically 1 mm to 10 mm in thickness) were generally encountered underlying the upper sand to silt deposits in the low-lying areas of the foundations at all dam sections, with the exception of the North Dam. These units are hereby referred to as

"Varved Clays". The groundwater table was found to vary from near surface to about 3.5 m below ground surface in the TMF area. Bedrock encountered was generally good to excellent quality based on RQD measurements.

18.2.2 Design Criteria

Operation of the Hardrock Mine will result in approximately 140 Million tonnes ("Mt") of tailings from an industry standard crush, grind, leach gold extraction process. A cyanide destruction system will be used to process all water sent to the TMF impoundment area. An allowance for an additional 5 Mt of reclaimed historical tailings has been made in the design criteria for a total capacity of 145 Mt.

In accordance with LRIA Hazard Potential Classification ("HPC") the TMF perimeter dams have been classified as having 'Very High' hazard potential. This classification is based on the potential environmental impacts in the event of a catastrophic failure. The starter dam is designed at 11 m high with the ultimate dam raised to a maximum section height of 35 m.

Dam design criteria include storage for the Environmental Design Flood ("EDF") defined as a 100-year return hydrologic event (24-hour storm or 30 day spring freshet) with no discharge through the spillway. An emergency spillway will be maintained to safely pass the Inflow Design Flood ("IDF") consisting of a routed Probable Maximum Flood ("PMF") of 24-hour duration. The dams are designed for seismic events of 1:10,000 year (Maximum Credible Earthquake).

Tailings will be deposited in the TMF by conventional tailings slurry method of deposition during Mine operations. This method provides flexibility for flooding of the deposited material at closure should it be required to achieve geochemical stability of the tailings. Selection of conventional tailings slurry deposition method includes spigotting of tailings slurry from the perimeter dams to produce a wide exposed beach, displacing the tailings pond away from the perimeter dams towards natural ground along the western perimeter. This is designed to enhance dam stability.

18.2.3 Tailings Characteristics

Particle size distribution tests were conducted on two tailings samples. The tailings samples tested consist of non-plastic hard rock particles and are predominantly silt and sand sized with approximately 75% to 82% of the particles by mass finer than 0.075 mm diameter (silt and clay sized), out of which 11% to 16% by mass are clay sized.

Tailings geochemistry has been evaluated by Stantec which indicate that less than 10% of the ore is considered potentially acid generating (“PAG”). This amount will be reduced through oxidization during processing, estimated at 26%, reducing the overall Acid Rock Drainage (“ARD”) potential for the tailings.

18.2.4 Tailings Deposition Plan

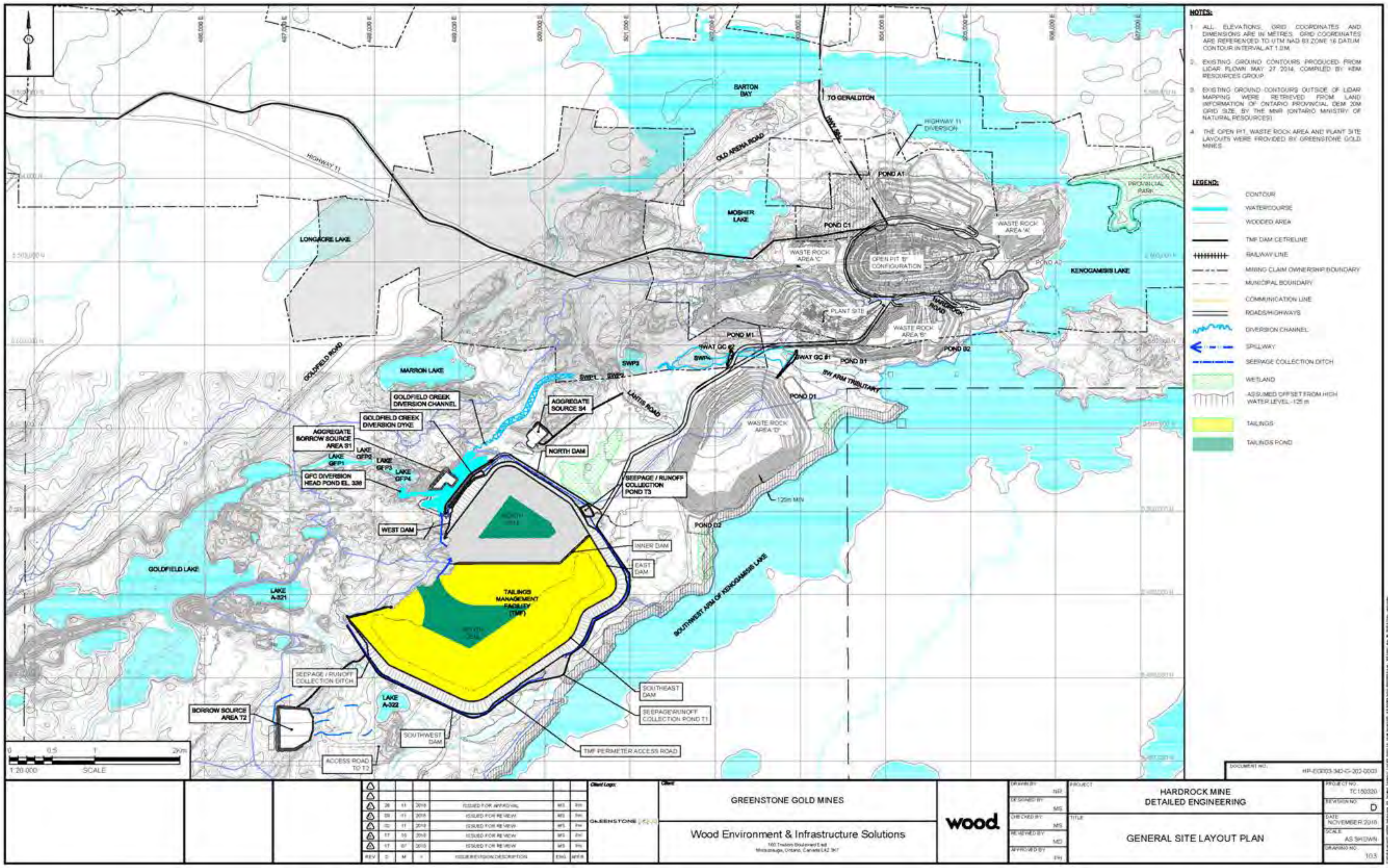
Key tailings operating data include:

- Total tailings production: 140 Mt
- Dry density of deposited tailings: 1.34 t/m^3 (for short-term planning)
- Total deposited tailings volume: 105 Mm^3

The tailings deposition plan involves spigotting tailings from the crest of the embankment dams to produce a wide beach to enhance dam safety and minimize seepage under or through the dams. The TMF South Cell has been designed with capacity to hold the tailings for the first year of operation. The South Cell dams will be raised annually during the first four operational years. During this initial period of four years, a temporary diversion channel will be maintained to divert freshwater from the North Cell around the TMF and reduce the quantity of water to be managed and pumped to the process plant. North Cell dams will be built during the fourth year of operation to facilitate tailings deposition in the north cell during the fifth year of operation. Thereafter, tailings deposition will alternate between the North and South Cells with deposition into the North Cell targeted for early completion to facilitate early reclamation and closure. A key objective of the deposition plan is to maintain the TMF pond against natural ground on the western side of the TMF. This design will help to mitigate dam safety risks during operations, reduce dam seepage rates, and facilitate the ultimate closure design for the TMF.

Figure 18.1 shows the general arrangement of the TMF.

Figure 18.1: General Arrangement Plan TMF



18.2.5 TMF Water Management

During detailed engineering, a TMF water balance was completed which estimates that the site will have a positive water balance. Construction of the TMF will be completed in stages to minimize the spread/extent of the facility. Staged development of the TMF, initially with only one of two cells, will achieve this. Progressive rehabilitation of the North Cell will take place early in the mine life, thus reducing water management in the North Cell.

The TMF water balance has been carried out for mill operation on a continuous basis considering precipitation conditions of 1 in 25 wet year, 1 in 100 wet year, and 1 in 100 dry year. The water balance indicates that the TMF is expected to supply 92% of the mill makeup water for most of the years of operation. The balance of 8% is moisture in the ore and freshwater supply required in the mill. Under 1 in 100 dry year condition, the TMF can supply 65% to 90% of reclaim water to the mill considering the dry years occur in the initial years of operation. The balance of mill make-up water will be sourced from other ponds within the project area such as waste rock storage area ponds, and the historical underground workings. The pond volume expected in the TMF varies from 3 Mm^3 for average precipitation conditions to a maximum of 4 Mm^3 during 1 in 100 year wet precipitation conditions. The emergency spillway invert levels will be established at the high-water level corresponding to the EDF event at all stages of operation. This will ensure capacity to contain the EDF without any discharge to the environment.

During the first four years of operation the South Cell will be used for tailings deposition exclusively. Runoff in the future North Cell which would have previously reported to GFC will be routed through a temporary diversion channel to an existing gully to the east leading to Kenogamisis Lake. The remnant GFC runoff will be separated from the South Cell tailing seepage pond by an inner dam seepage collection dyke built in the North Cell. From operational Year 5 onwards, connecting transfer spillway channels between the North and South Cell will allow equalization of pond levels. The mill reclaim pumping system will be located in South Cell. TMF emergency spillways will be constructed in the North Cell for safe passage of flows from storm events greater than EDF. The flows will be directed towards the GFC diversion head pond.

18.2.6 TMF Seepage Mitigation and Control

Seepage analyses have been carried out and seepage mitigation measures have been included in the TMF design. Geotechnical field investigations in the southwest dam identified a thick high permeability layer of sand/sandy silt overlying low permeability varved clays. To limit seepage, a cement bentonite slurry wall will be constructed, penetrating the upper sand silts and keyed into the varved clay units.

At the southeast and east dam locations, the thickness of sandy silt units is between 3 and 4 m over the varved clay units. Excavation of a cut-off trench through this material would be challenging given the depth of sandy silts and a high, near ground level water table. A cement bentonite slurry wall will be constructed for these dams. Similar to the Southwest dam, the varved clay units in the valley section of the West dam are also deep seated under the upper high permeability sand/sandy slit units. Therefore, a cement bentonite slurry wall has been proposed in parts of West dam. For north dam, parts of West dam and Southwest dam closer to the abutments a cut-off trench backfilled with compacted till is proposed.

A system of perimeter seepage collection ditches and ponds downstream of the dams will capture seepage and runoff from the external dam slopes. These perimeter ditches will drain to three seepage collection ponds located on the perimeter of the TMF namely T1, T2 and T3. The collected seepage will be generally pumped back into the TMF. In case there is surplus water, which cannot be pumped into the TMF, water will be treated prior to discharging to the environment.

18.2.7 Dam Design

The following three alternative dam design sections were considered:

- Central low permeability till core abutting compacted waste rock buttresses;
- Compacted waste rock with upstream low permeability liner;
- Compacted waste rock with upstream inclined low permeability till core.

The compacted waste rock with upstream inclined low permeability till core was determined to be the preferred section. The dams will be downstream raised so that the entire downstream slopes are placed on a prepared, inspected and approved foundation. Till core deformation analyses has been carried out by computer modelling and the deformations are noted to be well within the acceptable ranges.

The perimeter dams and inner dam for TMF cell division will be raised in stages depending upon capacity requirements. Dam foundation preparation, till core, filter construction and approximately 5 m width of mine rock abutting the till core will be executed by contractors, with the mine rock embankment raised using the mine fleet.

Stability analyses have been carried out for the TMF perimeter dams under different loading conditions. This modelling was based on comprehensive characterization of foundation units from geotechnical investigations during 2018 and 2019. Specific attention was given to the deepest sections of overburden and critical foundation locations. Varved Clay unit underlying the foundation governed design of the TMF perimeter dams. Modelling of the final design indicated that all target factors of safety are met.

The dams have been designed considering undrained behavior of varved clay units which require relatively flatter upstream and downstream slopes with toe stabilizing berms. Post-liquefaction analyses of the foundation soil units were conducted during design. Overburden characterization based on cone penetration testing completed in 2015 and 2019 indicated that the varved clay units may liquefy under seismic loading from the design earthquake. To ensure dam stability under these conditions, longer downstream toe berms are required. Currently, the starter dams for the south and north cells have been designed for post-liquefaction conditions. It should be noted that the starter dam configuration designed for the post-liquefaction scenario involving toe berms is merely additional mine rock placed on the downstream shell which would otherwise be placed at a later, subsequent stage as a part of dam raising for intermediate stages of operation.

Liquefaction potential of the varved clays has been subsequently assessed by performing ground response analyses (SHAKE 2000) to determine the cyclic stress ratio ("CSR") in the foundation units under the design earthquake. About 18 cyclic direct simple shear ("CDSS") tests were carried out on undisturbed varved clay samples at Carleton University laboratory to estimate the cyclic resistance ratio (CRR) of varved clay units.

The results of the analyses indicated that the varved clay units have the potential for liquefaction at the dam toe and the liquefaction zone is limited from the dam toe towards interior up to 1.5 m high rockfill.

Ultimate dams' stability analyses were updated for the post liquefaction scenario of the varved clays using post liquefaction strength determined from the CDSS tests. The results indicated that the limited extent of varved clay liquefaction did not have any impact on the current ultimate dam design sections. Therefore, no changes are required for the ultimate dam design.

The objectives of dam foundation preparation are to ensure stability of dams and manage seepage through the foundations. To prevent piping of fine soil particles in the rockfill, foundation filters are provided under the upstream dam shells, further improving dam stability.

Foundation treatments include removal of organics and unsuitable materials. In deeper overburden areas, excavation of seepage cut-off trenches complete with cement bentonite slurry walls are designed. Over fractured bedrock outcrops compacted till will be placed to mitigate seepage.

A TMF Dam inspection and monitoring program will be employed during construction and operation of the TMF Dams to assess the performance and safety and to verify that actual conditions are consistent with the design considerations. Instrumentation will include vibrating wire piezometers in foundation units and till core, inclinometers to measure deformations, surveying of dam crests, and magnetic settlement system to monitor differential settlements.

18.2.8 TMF Dam Raising Schedule

Figure 18.2 shows the struck-level capacity curve for the TMF with dam raising schedule for each year of operation.

The crest level of the dam is determined from required tailings capacity. The tailings discharge elevations at each stage of TMF operation will be 0.3 to 0.5 m below the dam crest at each stage. The emergency spillway invert level for each stage dam is set based on the storage requirement to contain the EDF. The emergency spillways will safely pass the PMF with adequate freeboards to the dam crest and till core.

The still water level in tailings pond during passage of PMF will be minimum 2.6 m below the till core level. Maximum expected wave heights for the design wind condition during PMF will be 1.1 m providing a clear freeboard of 1.5 m during the passage of PMF.

18.2.9 Closure Considerations

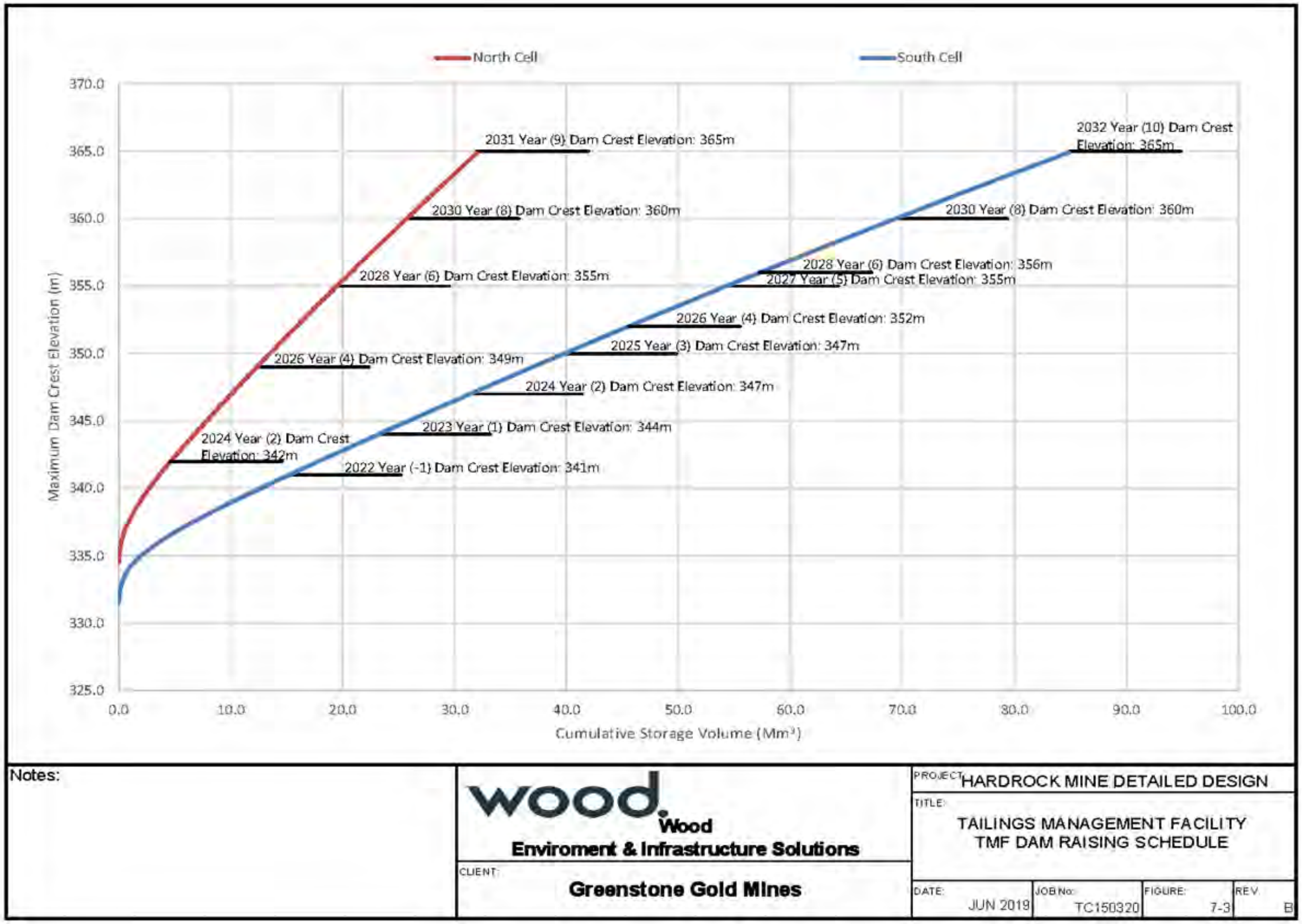
Closure of the TMF involves lowering of the spillways and re-vegetation of the exposed beaches. Runoff will be directed through overflow spillways constructed in natural ground when deemed suitable for discharge to the environment.

18.2.10 Construction Borrow Materials

The Till borrow source area has been identified and is located in the high ground south of Goldfield Lake. Glacial till deposits underlay a thin organics layer (0.1 to 0.5 m thick) and are predominantly fine grained till varying in composition from silty sand to sandy clayey silt. The till contains trace gravel with cobbles and boulders. The indicated maximum thicknesses of the identified till borrow area is 21 m.

Two sand and gravel borrow sources have been identified within the Project area. Both borrow areas S1 and S4 are located close to the northern perimeter of the TMF. These gravel borrow sources will be used for roads around the TMF and some for dam filters. Finally, to complete sand quantities for dam filters other local borrow pits have been identified and quality and quantities have been confirmed.

Figure 18.2: TMF Struck Level Capacity



18.3 Goldfield Creek Diversion

Wood has provided design and permitting support services for the proposed diversion of Goldfield Creek, a small watercourse that currently drains through the proposed TMF footprint into the Southwest Arm of Kenogamisis Lake. Services include geotechnical site investigations, design of the channel components, design of the access road and haul road crossings, and provincial and federal permitting associated with the diversion. The diversion alignment was selected based on a balance of environmental, social, economic and operational risk parameters. The diversion is required to direct flows of the existing Goldfield Creek away from the proposed TMF area. This will be achieved through a combination of natural channel design and engineered structures to contain and or direct the natural flows into the existing Southwest Arm Tributary drainage, which also reports to the Southwest Arm of Kenogamisis Lake.

Section A: A Goldfield Creek Diversion Pond (19.17 hectares) (3) will be developed through construction of the Goldfield Creek Diversion Dyke (2). The pond will attenuate natural flows from the creek and convey them into the new diversion channel, Section B.

Section B: A realigned channel, similar to the existing Goldfield Creek, will be developed in an existing valley feature that measures approximately 1,600 m in length. A low flow meandering channel (4), about 2,600 m long, designed using natural channel design principles will convey flows to the beginning of the Southwest Arm Tributary.

Section C: The existing Lahti's Road crossing (5) and the channel downstream of the crossing to the next pond (SWP3) will be reconstructed to accommodate the increased flows associated with the diversion. Lahti's Road crossing will be designed to convey a minimum of a 25-year hydrologic event.

Section D: To accommodate the increased flows, two permanent valley-wide grade controls (6) will be constructed to create slow-moving ponded water conditions. This will replace the fluvial characteristics of the watercourse with those of a low energy pond system, thereby minimizing risk of channel erosion. Within this section, between the two grade controls, a haul road crossing will be constructed to facilitate traffic between the plant site and the TMF. The haul road crossing has been designed to pass the 100-year hydrologic event.

Section E: No changes necessary in this area as the channel is under the existing backwater effect of the lake and is wide enough to accommodate the increased flow.

Figure 18.3 shows the general arrangement of the Goldfield Creek Diversion in relation to the TMF and other site features. The diversion can be described by discrete sections (A through E) and features (numbers 1 through 7) as shown in Section A: A Goldfield Creek Diversion Pond (19.17 hectares) (3) will be developed through construction of the Goldfield Creek Diversion Dyke (2). The pond will attenuate natural flows from the creek and convey them into the new diversion channel, Section B.

Section B: A realigned channel, similar to the existing Goldfield Creek, will be developed in an existing valley feature that measures approximately 1,600 m in length. A low flow meandering channel (4), about 2,600 m long, designed using natural channel design principles will convey flows to the beginning of the Southwest Arm Tributary.

Section C: The existing Lahti's Road crossing (5) and the channel downstream of the crossing to the next pond (SWP3) will be reconstructed to accommodate the increased flows associated with the diversion. Lahti's Road crossing will be designed to convey a minimum of a 25-year hydrologic event.

Section D: To accommodate the increased flows, two permanent valley-wide grade controls (6) will be constructed to create slow-moving ponded water conditions. This will replace the fluvial characteristics of the watercourse with those of a low energy pond system, thereby minimizing risk of channel erosion. Within this section, between the two grade controls, a haul road crossing will be constructed to facilitate traffic between the plant site and the TMF. The haul road crossing has been designed to pass the 100-year hydrologic event.

Section E: No changes necessary in this area as the channel is under the existing backwater effect of the lake and is wide enough to accommodate the increased flow.

Figure 18.3, and described as follows:

Section A: A Goldfield Creek Diversion Pond (19.17 hectares) (3) will be developed through construction of the Goldfield Creek Diversion Dyke (2). The pond will attenuate natural flows from the creek and convey them into the new diversion channel, Section B.

Section B: A realigned channel, similar to the existing Goldfield Creek, will be developed in an existing valley feature that measures approximately 1,600 m in length. A low flow meandering channel (4), about 2,600 m long, designed using natural channel design principles will convey flows to the beginning of the Southwest Arm Tributary.

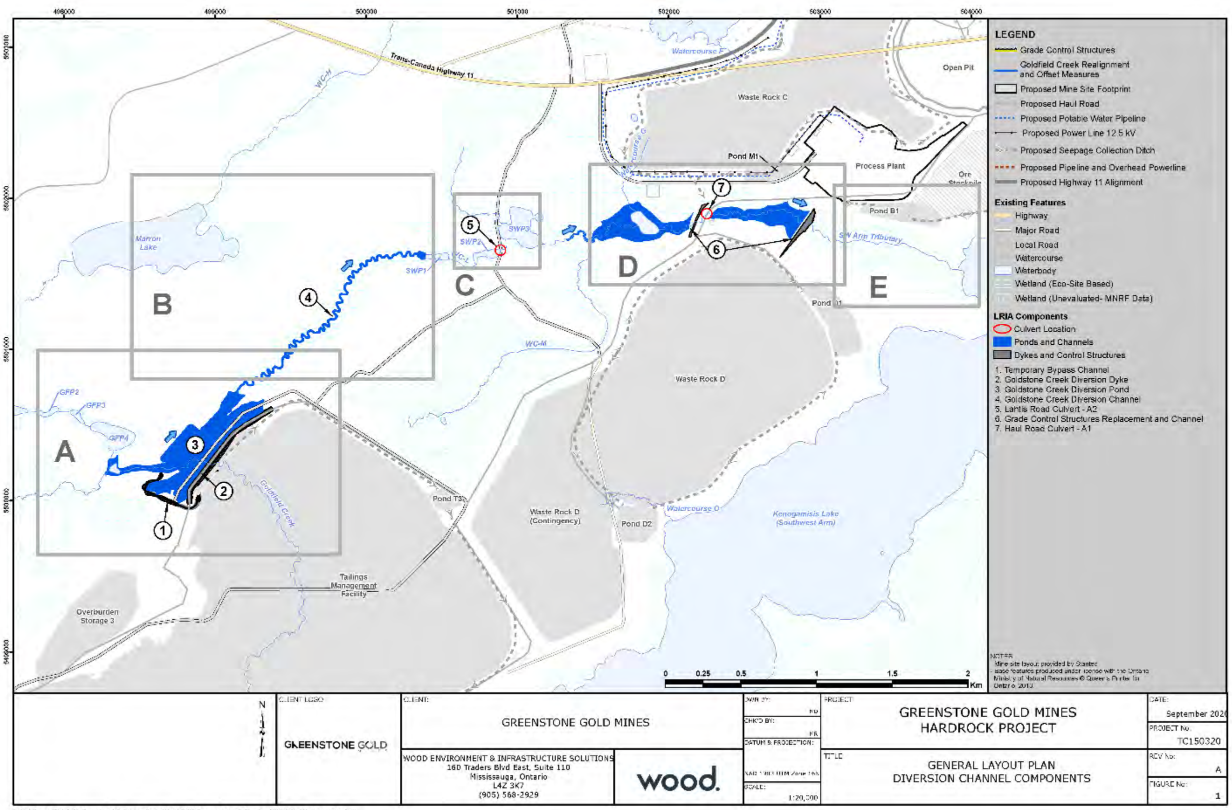
Section C: The existing Lahti's Road crossing (5) and the channel downstream of the crossing to the next pond (SWP3) will be reconstructed to accommodate the increased flows associated with the diversion. Lahti's Road crossing will be designed to convey a minimum of a 25-year hydrologic event.

Section D: To accommodate the increased flows, two permanent valley-wide grade controls (6) will be constructed to create slow-moving ponded water conditions. This will replace the fluvial characteristics of

the watercourse with those of a low energy pond system, thereby minimizing risk of channel erosion. Within this section, between the two grade controls, a haul road crossing will be constructed to facilitate traffic between the plant site and the TMF. The haul road crossing has been designed to pass the 100-year hydrologic event.

Section E: No changes necessary in this area as the channel is under the existing backwater effect of the lake and is wide enough to accommodate the increased flow.

Figure 18.3: Diversion Components



18.3.1 <u>Design Criteria</u>

The diversion channel was originally designed to pass the 100-year return hydrologic event (24-hour storm or freshet). Subsequent consultation with regulators during the environmental assessment revised the design for the channel to pass a minimum of the 500-year return hydrologic event. The Goldfield Creek Diversion Pond and Diversion Dyke were designed for the Probable Maximum Precipitation ("PMP"). The diversion channel can also carry the Probable Maximum Flood ("PMF") from the catchment by flooding along the banks.

Road Crossings associated with the Diversion Channel were designed according to risk and purpose including:

- Lahti's Road, designed to minimum 25-year event, (local access road);
- Haul Road crossing of the Southwest Arm Tributary, designed to 100-year event, (main haul road to TMF).

18.3.2 Diversion Design

The Diversion Channel was designed using engineered structures (diversion dyke, grade controls and crossings) and natural channel design principles based on a 2016 geomorphological analysis of the existing and proposed channels.

The Goldfield Creek Diversion Pond and Diversion Dyke were designed to a normal operating water level of 340 masl, and to convey the PMF event without overtopping. The crest level of the diversion dyke has been set at 342.3 masl. Design features incorporated into the diversion also include ecological values and habitat characteristics in response to comments from communities and regulators. Detailed descriptions of the habitat components and ecological features can be found in the Fisheries Act, Paragraph 35(2)(b) Authorization, Offset Plan and MDMER Schedule 2 Fish Habitat Compensation Plan; (Wood 2019).

18.3.3 Diversion Dyke Design

Geotechnical investigations have been carried out in the GFC diversion components and TMF in 2014, 2015, 2018 and 2019 to understand the subsurface conditions required for the design. The GFC Diversion Dyke has been designed in accordance with the Lakes and Rivers Improvement Act (LRIA) and Canadian Dam Association ("CDA") guidelines. The GFC Diversion Dyke has a "Very High" hazard potential because of the potential loss of life, loss of fish habitat, and loss of property (economic). Therefore, the dyke is designed for the most severe flood criteria, being the PMF. The TMF located immediately southeast of the GFC dyke is designed to contain the EDF corresponding to a 100 year event. A TMF Emergency Spillway will convey the flows above the EDF event. The GFC diversion pond will receive the overflows from the TMF Emergency Spillway. All flows will be diverted down the permanent diversion channel to the Southwest Arm Tributary.

The dyke section is a rockfill dam constructed on a stripped and prepared foundation with a low permeability inclined till core along the upstream face of the dyke. Graded filters and transition layers are provided upstream and downstream of the core, as well above the prepared foundation downstream of the core. Clean, non-acid generating rockfill is designed for construction of the shell downstream of the core. A soil-bentonite slurry seepage cut-off wall is planned to penetrate the upper outwash layers of sandy silt/silt and tied into the low permeability varved clays.

Goldfield Creek Diversion Dyke Design Report of November 2018 provides further details on the diversion dyke design.

18.3.3 Closure Considerations

The diversion is a permanent realignment of the Goldfield Creek system, and as such will remain in place at mine closure. However, most road crossings will be removed at or before closure when their project needs are met. The diversion dyke and valley wide grade controls will be permanent structures.

18.4 Water Management

18.4.1 Administrative Water Services

A tie-in to the municipal network will be constructed to the Project site for potable water to support the change room, the lunchroom, the various washrooms and showers and the emergency showers.

The wash bay water at the truck shop is required for the mining fleet and will be provided from the water equalization pond which is principally fed from shaft dewatering activities. The wash bay will be equipped with filtering system to recirculate most of the water to the wash bay. Sludge will be disposed through the process plant tailings management system.

18.4.2 Effluents

Two types of effluents will be generated during Project activities: mine effluent and sanitary effluent. The water quality standards applicable to mine effluent are the Provincial Water Quality Objectives (“PWQOs”) (MOE, 1994), Ontario Regulation (“O.Reg.”) 560/94-MISA Metal Mine Sector Effluent Criteria, and Federal Metal Mining Effluent Regulations (“MMER”) Effluent Criteria. The water quality standards applicable to sanitary effluent are the PWQOs. The Assimilative Capacity Study (Stantec, 2016) conducted for the Project identified effluent discharge locations and proposed effluent criteria for both mine and sanitary effluents discharging to the Southwest Arm of Kenogamisis Lake which are protective of the receiving environment. The proposed effluent criteria meet and exceed MMER and O.Reg. 560/94 criteria at end of pipe and the PWQOs for all parameters are met within a reasonably small mixing zone in the receiving waterbody.

Mine Effluent: All collected mine water is directed through distributed runoff and seepage collection ponds to the centralized mine water Collection Pond M1, which is designed to provide store for all mine site water with controlled release to the Southwest Arm of Kenogamisis Lake following treatment in the effluent treatment plant.

Sanitary Effluent: The Project has two sources of sanitary sewage: the temporary construction camp located near Barton Bay and the main mine site which includes the offices, process plant and mine buildings. The temporary construction camp will be connected to the municipal sewage system by incorporation of a new sewage lift station.

For the plant and mine site buildings, a permanent modular sewage treatment plant will be installed to accommodate up to 100 persons at any given time.

18.4.3 Site Runoff and Spillage Control

Runoff refers to runoff over ground surfaces as well as seepage to the surface from groundwater/subsurface sources. Four main sources of site runoff produced by mining operation have been identified:

Runoff from Open Pit: Precipitation on the open pit will be directed to the historical underground workings associated with the MacLeod-Cockshutt and HardRock mines. The use of the historical underground workings provides a storage reservoir for open pit seepage and runoff during the mining operation and allows for flexibility in the water management approach.

Runoff from WRSA and Ore Stockpiles: Runoff from the Waste Rock Storage Areas (“WRSAs”) and ore stockpiles will be collected in a series of perimeter ditches which will drain by gravity or be pumped to one of seven local collection ponds and subsequently directed to the centralized Mine Water Collection Pond M1. The seepage collection ditches and ponds will be designed to the 100 year design event. After mine closure and WRSA rehabilitation, water will continue to be collected and directed to the open pit to accelerate the filling of the pit to form a pit lake. Once water quality is acceptable for discharge, the ponds will be decommissioned, and water directed to the environment.

Runoff from Mill Yard: Surface runoff and contact water from the processing plant, truck shop yard and ore stockpiles will be collected in perimeter ditches draining to the Collection Pond B1.

Runoff from TMF Dams: The TMF will include a perimeter seepage collection system to collect runoff and seepage from the perimeter dams. Water from the collection ponds will be pumped back to the TMF for reuse to meet process plant reclaim demand. The TMF has been designed to contain up to the Environmental Design Flood with emergency spillways designed for the PMF. The collected seepage will be generally pumped back into the TMF. In case there is surplus water, which cannot be pumped into the TMF, water will be treated prior to discharging to the environment.

18.4.4 Collection Ponds

Selection of collection pond locations was based on a balance of several objectives including the ability to collect runoff and seepage from the respective WRSAs by gravity flow, constructability and subsurface soil considerations, and satisfactory environmental protection during operations and into closure. The requirement to collect the seepage and runoff collection by gravity flow means the ponds are generally constructed by excavation, with small perimeter berms to provide ample storage to contain runoff from storm events.

Seepage from the ponds is inhibited through controlling the operating levels such that the pond level is maintained below the surrounding groundwater level, thus creating hydraulic containment. Pond M1, the central collection pond, is provided with a geomembrane liner because it requires a higher operating level to receive pumped flows from the six remote collection ponds and open pit. Pond C2 will be lined with geomembrane because granular soils in the area mean the hydraulic containment concept is not suitable.

Specific operating volume ranges (live storage) have been set for each pond according to the contributing catchment area. Storm reserve capacity is provided above the maximum operating water level in each pond to contain the runoff from a 100-year return Environmental Design Flood (EDF) with no discharge to the environment. Each of the ponds has a pumping station with sufficient capacity required during normal operations and following storm events. Table 18.1 summarizes the maximum operating volumes and EDF storage required in each pond, and the total capacity below the emergency spillway invert.

Table 18.1: Maximum Operating Volumes and EDF Storage for Each Pond

Aspect	Pond M1		Pond A1	Pond A2	Pond B1	Pond B2	Pond C1	Pond D1	Pond D2
Catchment Area Received	Open Pit, WRSA-C [81.9 ha]	Open Pit, WRSA-C (south) [107.8 ha]	Overb. Stockpile [75.1 ha]	WRSA-A [62.1 ha]	Plant Site, Ore Stockpile [97.8 ha]	WRSA-B [32.2 ha]	WRSA-C (north) [38.3 ha]	WRSA-D (north) [140.9 ha]	WRSA-D (south) [70.4 ha]
Maximum Operating Volume (m^3)	68,000	43,000	14,500	8,500	18,000	8,000	5,500	13,000	6,500
EDF Runoff Volume (m^3)	78,000	103,000	72,000	59,000	112,000	31,000	37,000	134,000	67,000
Volume at Emergency spillway Invert (m^3)	146,000	146,000	86,500	67,500	130,000	39,000	42,500	147,000	73,500

Note: The ultimate size of Pond M1 will need to be confirmed when the waste stockpile progressive reclamation plan is finalized.

The pond berms have been assigned "significant" Hazard Potential values based on the potential environmental impacts in the event of a failure. Emergency spillways have been designed to safely pass the peak flow from a 1,000-year return 24-hour storm of 142 mm precipitation depth.

18.4.5 Effluent Treatment Plant

The effluent treatment plant consists of a series of stages designed to increase water clarity, meet discharge water quality targets and adjust pH. In the first step, pH of the effluent is adjusted by the addition of acid and caustic in the first of three metal precipitation reactors. The presence of hydroxide$^-$ ions in contact with residual metal ions results in the formation of insoluble metal hydroxide precipitate.

The next step is pre-coagulation and coagulation where coagulant is added to the second metal precipitation reactor to assist in decantation and precipitate arsenic from solution. Proper mixing in the last two metal precipitation reactors ensures a homogenous diffusion of the coagulant in the water. Water is then transferred to the maturation tank where flocculant and microsand are added. Baffles and efficient mixing accelerate contact between the flocs, the flocculant and microsand, thereby ensuring the formation of ballasted flocs. Water is then transferred to a counter-current lamellar settling tank where the ballasted flocs sink to the bottom forming a sludge and the clarified water is decanted at the surface. The sludge is drawn out of the clarifier and directed to hydrocyclones to recover the microsand from the sludge. The microsand exits the hydrocyclone underflow and recycled, while the sludge evacuated at the overflow is directed to the sludge pumpbox before being transferred to the tailing pumpbox.

To achieve the final effluent discharge quality criterion, the clarified water is further polished in a disc filter. The clarified water is sent to an intercoagulation tank where coagulant and flocculant are added to help in the formation of flocs. A final pH adjustment is made by the addition of caustic or acid. After proper mixing and adequate retention time, the water is filtered to remove the precipitated flocs and obtain the clarified water that can be sent to the environment.

18.5 Power Supply and Distribution

18.5.1 Power Demand Estimates

The average power demand for the Project was calculated using the Project's process plant load list compiled by Ausenco and received by Greenstone Gold Mines ("GGM"). This included the average load required for the supporting infrastructure, such as the truck shop/warehouse, the administration office, the sewage treatment plant, etc. Peak power was calculated by considering all the connected nameplate loads and applying demand and efficiency factors. Table 18.2 illustrates the estimated average and peak power requirements.

Table 18.2: Average and Peak Power Demand

Description	Demand (MW)
Average Power Demand	34.1
Peak Power Demand	46.5

Note: Peak power = nameplate power x demand factor / efficiency, where demand factor is:
Duty loads = brake power at design condition / installed power
Intermittent loads = 0.3

18.5.2 Power Plant Design

The power plant design was prepared to optimize the fuel efficiency and minimize the capital expenditures while also providing enough spinning power capacity for peak loads. The optimized operating point of a reciprocating engine usually hovers close to the 85% engine load figure.

The power plant will operate on an N+2 operating philosophy, whereby five (5) units will generate power to the Project electrical network to meet average and peak power demand, one unit will be a hot standby off grid, and one unit will be in maintenance. The availability of the power plant under this operating philosophy is considered to be 100%.

Considering the design requirements, it was decided to design the power plant with seven generating sets having an electrical output of approximately 9.3 MW, at a voltage of 13.8 kV and at a frequency of 60 Hz, along with all necessary auxiliary equipment. The designed capacity of the plant under the described operating philosophy is 46.5 MW.

The average demand as illustrated in Table 18.2 is 34.1 MW for an operating load of 73 to 80%. Peak power demand is unlikely to be achieve since it would require all duty loads to simultaneously operate at design conditions in addition to 30% of intermittent nameplate loads. If this condition was achieved the five duty generating sets would operate at 100% engine load, or the sixth standby generator could be switch on for additional capacity.

18.5.3 Power Distribution

The process plant main electrical loads will be fed from the power plant's 13.8 kV Switchgear. Project loads located outside of the process plant such as the TMF, the effluent treatment plant, the office buildings, the truck shop and warehouse, communication tower, explosive reagent facility and all the various pumping stations of the Project will be fed through overhead power lines with tie-in points to the Project electrical network.

18.6 Other Project Infrastructure

18.6.1 Truck Maintenance Shop and Warehouse

The truck shop and warehouse complex will be located on the southwest of the process plant site with direct access to the mine haul road.

Eight heavy duty maintenance bays will be provided. The truck shop building dimensions are 121 m (L) by 39 m (W) by 21 m (H) for five bays, and the building is 3.6 m wider for the last three bays. Four bays (346 m^2 each) will mainly service the mining haul trucks. One bay (259 m^2) is to be used for smaller support equipment. One bay (404 m^2) will be used to service the big loader, then one more bay (303 m^2) will be for the maintenance of the drill rigs. The last bay (419 m^2) will be the wash bay for all mobile equipment. The concrete slab for tracked equipment will be constructed with embedded rails. Large fabric doors will be used for the heavy-duty maintenance bays. Three 30/5 t overhead cranes will service the maintenance bays. Two of these overhead cranes will be synchronized to work in tandem to lift one truck box.

The wash bay (419 m^2) will be located on the east end of the truck shop complex. Water cannons and steam washers will be located on both sides. Water, rocks, sediment and hydrocarbons will be collected in a settling pond and the water reused.

Compressed air will be provided throughout the maintenance facility. Lubricants will be stored in the truck shop in a specific enclosed area then distributed to four reel stations.

The main HVAC system will have units in strategic locations to provide heat and air exchange as needed in all areas of the truck shop/warehouse building. The heat provided to the units will come from the heat recovery system of the power plant and the glycol loop. The system installed in the wash bay will be designed to melt frozen chunks of mud on the mine fleet equipment being washed and serviced.

The warehouse (593 m^2) is designed on the northwest corner of the truck shop building. Heavy and light-duty racking will be installed. The warehouse will be used to store consumables and maintenance parts. A separate room with 116 m^2 will be used to store hoses, tools and cribs.

Office space (265 m^2) will be provided on the east side of the warehouse. A lunchroom (102 m^2) will be located on the mezzanine floor. An enclosed storage area (292 m^2) with handrails and stair access will be provided on the second floor.

18.6.2 Reagents Cold Storage Facility

This pre-engineered folding building (669 m^2) will be located at the plant site west of the mill building. The building will be used for process reagent storage and, long-term storage for major spares, and temporary warehousing during construction. The building is not heated.

18.6.3 Explosives Reagent Facility

This pre-engineered folding building (632 m^2) will be located South-East of the process plant site and along the road giving access to the Kenogamisis Lake. This location of the Site Mixed Emulsion ("SME") Plant has been chosen in order to respect the minimum quantity-distance requirements from the infrastructure. The single storey building will be supported on slab on grade. A temporary facility will be located northeast of the S4 access road near the two permanent magazines used for detonators, and booster as well as the storage of any packaged product.

18.6.4 Sewage Treatment Plant

Located at plant site, the sewage treatment plant ("STP") is made of containerized units (three) that will be installed over a concrete slab on grade. The STP is expected to treat sewage flows from the average daily plant site population but will be equipped with an equalization tank able also to handle peak flows at shift change.

18.6.5 Fuel Supply Storage & Distribution

Fuel will be stored at two sites, the Process Plant and the pit, to serve light vehicles and mine equipment respectively.

Diesel and gasoline will be stored in double walled tanks at the plant site fuel storage area. Each product will have a designed storage volume of 50,000 L, and 7,500 L respectively.

Diesel and urea will be stored in double walled tanks at the pit fuel storage area. Each product will have a designed storage volume of 150,000 and 50,000 L respectively. Liquid urea will be used with diesel fuel to reduce the NOx emissions of the mine fleet.

18.6.6 Information Technology and Communications Systems

The Project IT Strategy will be implemented at the start of Construction. The strategy addresses, all aspects of an IT system required during construction, commissioning, and steady state operations of the Project. It includes the applications to support the core business functions and supporting technology. Applications and technology will be phased in as the Project advances.

A site-wide, fibre optic communications network installation will follow the power line installation. Network cabling will be run between the various buildings for connectivity within the local area network ("LAN"). Closed Captioned Video ("CCTV") for security, a wireless connectivity for LAN extensions, and an external Wide Area Network ("WAN") for connectivity onto the Internet Web and the Cloud service providers, such as ERP, email, and office tools such as Microsoft Office 365. Within the LAN will be service to support printing, telephony, server data and storage management.

As cellular phone coverage in the area is currently inadequate, a 100-metre, guyed cell tower will be erected south of the plant site and host all cellular, radio and LTE (mine dispatch) system antennas. Boosters and in-building antennas will be added where and when necessary in the Administration and plant buildings. Quality of coverage will be assessed throughout the life of mine as stockpiles grow and the Open Pit deepens with additional repeaters and/or antennas added as necessary.

18.6.7 Roads

The Project benefits from direct access to Trans-Canada Highway 11, approximately 275 km northeast of Thunder Bay and 600 km west of Timmins/Matheson. A section of Trans-Canada Highway 11 will have to be deviated and reconstructed to avoid the Hardrock open pit mine.

A new site access road will be constructed off Trans-Canada Highway 11 to the administration, mineral processing, power generation, and shop facilities.

18.6.8 Assay Laboratory

Assaying requirements will be divided amongst a third party offsite arrangement for sample preparation and gold assaying, and in-house geochemical lab for digestions and assaying of other elements, and a second third party offsite laboratory for certified water analysis. Assaying services for the Project will be outsourced.

18.6.9 Admin Building

The administration offices complex will be located on the southwest side of the process plant building. The building will house change rooms for both men and women, offices, conference rooms, and a lunchroom for the office staff. The first aid station will be located on the first floor. It will also house a training room at the main entrance for site induction sessions and other training.

18.6.10 Fire Protection

The Plant site fire protection system consist of two main loops which serve the wet sprinkler system for the processing plant (grinding and west plant buildings), HPGR building, crushed ore storage and reclaim tunnel facility, administration building and truck shop complete with fire cannons. A dry section of the sprinkler system serves the crushing building (primary and secondary crushers).

A perimeter fire hydrant network is installed around the plant infrastructure and process plant. Fire hydrants and wet sprinklers servicing the site buildings and effluent treatment plant are connected to the main rings via buried HDPE pipes.

The plant site fire protection system will be fed from a dedicated fire pumping station located by the process plant and close to the treated/fire water tank. The system has centrally controlled automatic fire detection and alarms. The main alarm panel will be located in the administration building in the security office at the main entrance where there will be a 24-hour presence of security guards.

18.6.11 Security

Access to the processing, power, and administration area will be secured by a remotely operated vehicle gate, controlled by security guards on 24-hour duty. The processing facilities and truck shop will be monitored by CCTV surveillance. There will be additional CCTV surveillance of the yard area around the processing, power and administration areas, including the employee parking area and main gate. The gold refinery and gravity circuit area will have an additional level of security.

18.7 Infrastructure Relocation and Offsite Infrastructure

The Project is located over or close to existing infrastructure. The plan is to relocate (rebuild or dismantle) the affected infrastructures.

18.7.1 Private Properties

The future open pit mine is located over the MacLeod townsite and very close to the Hardrock town site. GGM has acquired all 65 private properties in the vicinity of the Project. An additional property of the edge of the Project Development Area is currently being assessed for purchase which would bring the total to 66 private properties acquired.

The Husky Gas Station located at the junction of Highway 11 and Michael Power Boulevard has been purchased from the current operator and is now closed. This gas station will be removed after the realignment of Highway 11.

The Surface Rights for nine mining patents owned by Tombill Mines have been acquired in order to allow the construction of portions of the Highway 11 and Goldfield Creek realignments. The purchase of the surface rights for an additional three Tombill patents in the Goldfield Creek area are in progress and are expected to be completed by end of 2019.

18.7.2 Government and Municipal Properties

An Ontario Provincial Police (“OPP”) station is located in the vicinity of the future open pit mine operation and needs to be relocated. The new proposed location is at the junction of the new Highway 11 and existing Michael Power Boulevard. GGM plans to buy the existing OPP station when the project is approved for construction and the new OPP station will be built without any further involvement of GGM.

The municipality’s golf course is located just north of the future open pit mine on land owned by GGM. This 18-hole golf course will be reduced to the original 9-hole course on the west side of Michael Power Boulevard to accommodate the mine development.

The Geraldton Heritage Interpretive Centre is located over the MacLeod High Tailings area next to the golf course. The plan is to demolish this infrastructure at the end of the mine pre-production period or during the first year of operations.

A historical mine headframe was esthetically refurbished to be displayed as a monument at the junction of the existing Highway 11 and Michael Power Boulevard. The plan is to demolish this headframe toward the end of the mine pre-production period.

18.7.3 Relocation of Highway 11 and MTO Patrol Station

Highway 11 passes through the future Hardrock open pit mine and must be relocated to accommodate mining. Newer segments of Highway 11 (4.7 km) and Michael Power Boulevard (0.6 km) will be constructed to route traffic to the north of the mining operation. Various stakeholders including the Ontario Ministry of Transport ("MTO") and the local community were involved during the detailed design of the Highway 11 relocation. A portion of the new highway will be over the historic MacLeod tailings. Detailed geotechnical investigations in this area have been done to finalize the design. The intersection between Michael Power Boulevard and Highway 11 will be relocated to the west allowing for improved alignment. Construction will take place over a 2-year period during which minimal interference with existing traffic will be encountered. Detailed Engineering for the new Highway segments is over 90% completed.

The MTO owns a highway patrol station east of the Hardrock Project which is within the relocated Highway 11 right of way and must be relocated. A new location has been identified at the west end of the Project Development Area along Highway 11, opposite the mine entrance. Design of the new Patrol Yard is complete and included significant involvement from MTO. Upgrades were necessary for the new facility to align with current standards. Construction of the new MTO Patrol Yard is expected to take place over a one-year period.

18.7.4 Natural Gas Distribution Pipeline

To supply natural gas required for power generation, a new tap off the TransCanada pipeline ("TCPL") Mainline and a new distribution pipeline will be constructed from the Geraldton metering point on the TCPL Mainline, through the town of Geraldton to the Project site.

Enbridge (previously known as Union Gas), TransCanada's delivery agent for the Greenstone, Ontario region, has now completed basic engineering work for this pipeline. The pipeline will have a delivery pressure of 350 psi to the site metering station, which will be located at the intersection of Highway 11 and the site's access road. At the metering station pressure will be reduced to 100 psi for site-wide distribution to the power plant, process plant, etc. The new Enridge station will be designed for a maximum hourly flow of 11,000 m^3/h.

18.7.5 Relocation of Hydro One Electrical Infrastructure

The electrical infrastructure near the Project will be moved out of the footprint of the Open Pit. GGM and Hydro One have developed a plan by which the existing 115 kV Longlac Transmission Station ("TS") will be relocated approximately 2.2 km west of its current location, along with the incoming 115 kV power line and the outgoing 44 kV feeders servicing the municipalities of Geraldton and Longlac, amongst others.

The dismantling of the existing Longlac TS will be carried out only when the new Longlac TS and all feeders are ready to be energized. GGM will perform the dismantling.

The exiting Hydro One Networks Inc. ("HONI") operating centre that is annexed to the existing substation will be relocated separately to its own location in Geraldton’s designated industrial area.

18.7.6 Historical Tailings Relocation

The extent and volume of historical tailings has been characterized by others and summarized in the memorandum by Stantec dated July 9, 2019 titled “*Historical Tailings Inventory*, *Greenstone Gold Mines Hardrock Project*”. In 2016, an Environmental Assessment (“EA”) was completed by Stantec which identified that only a portion of the historical tailings would be removed as part of the Project development, with the remaining historical tailings being left in place.

There are two sources of historical tailings; the Hardrock Tailings and the MacLeod Tailings. The volume of the historical Hardrock tailings to be relocated to the TMF was developed based on a 30 m setback from the highwater mark of Kenogamisis Lake (as discussed in the EA). The setback was required to protect the riparian zone and reduce potential effects to fisheries habitat and water quality associated with working in close proximity to the lake. Approximately 70% of the historical tailings will be relocated to the TMF with an estimated volume of overlying soil and historical tailings of 330,400 m^3. It is anticipated that the historical Hardrock tailings will be relocated between 2028 and 2031.

Approximately 23% of the historical MacLeod tailings will be removed during construction of the open pit footprint and relocated to the TMF, with an estimated volume of overlying soil and historical tailings of 2,144,500 m^3. The initial historical MacLeod tailings to be mined as part of the starter pit and pit expansion from mid-2023 to the end of 2024 and be relocated to an engineered cell within the south portion of the TMF (approximate volume of 1,000,000 m^3). The TMF cell is being constructed prior to 2023 specifically for historical tailings placement as the full TMF will not have been constructed during the early mining operational stage. To stabilize the historical tailings remaining in place, a buttress will be constructed along the south edge in the area of the open pit.

A seepage subdrain installed around the north side of the MacLeod High Tailings will facilitate the collection and pumping of seepage from the historical tailings for treatment before discharge to the environment.

The removal of the historical MacLeod and Hardrock tailings will be completed using a combination of dozer pushing and excavation with either front end loaders or excavators. As the excavation will be advanced below the water table, water management will be required during construction and is anticipated to include a series of collection ditches and sumps, and potentially the use of well point systems to lower the water table in advance of the excavation works. Water collected during excavation works will be pumped to the ETP for treatment prior to discharge to the environment.

18.8 Temporary Construction Infrastructure

18.8.1 Temporary Camp

Due to limited accommodations in the area a temporary camp is required for the construction and pre-production period to lodge the construction workers. The camp will be located on GGM land approximately 2.5 km north of the process plant site on Old Arena Road near the intersection with Michael Power Boulevard. The camp will have an average occupancy of approximately 400 persons, with a peak of approximately 600 persons.

Natural Gas, potable water and power will be organized by GGM and provided to a centrally designated point on the camp site, after which the contractor will be responsible for its own distribution. The camp will require the collection and trucking of wastewater to a local disposal/treatment facility for the first few months of operation until it can be connected to the upgraded municipal system.

A shuttle bus service will be set up to shuttle workers from the temporary camp to the plant site.

19. MARKET STUDIES AND CONTRACTS

Neither G Mining Services Inc. nor Premier Gold Mines Limited have conducted a market study in relation to the gold metal that will be produced from the Project. Gold is freely traded, at prices that are widely known, so that prospects for sale of any production are virtually assured. Prices are usually quoted in USD dollars per troy ounce.

The gold doré refining agreement will be negotiated once the Project is approved for construction.

20. ENVIRONMENTAL STUDIES, PERMTTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

This Section provides an overview of the environmental studies and consultation efforts that have been completed to support the federal and provincial environmental approval requirements for the Project. Information on environmental studies and environmental effects is summarized from the approved final environmental effects assessment submitted in July 2017 (Stantec, 2017) with supplemental information provided in August 2018 to meet the requirements of the federal and provincial EA processes. Federal approval was received in December 2018 and provincial approval was received in March 2019. Any known environmental issues that could materially impact the Project from design through operations and closure are also discussed.

Environmental baseline studies were initiated for the Project in 2013 and used to identify environmental constraints during the development of preliminary layouts and designs for the Project. This includes consideration of siting and layout of Project infrastructure as well as consideration of design alternatives from an environmental management and approvals perspective. This environmental baseline was the basis for determining incremental changes and predicting environmental effects associated with the Project.

The Project is subject to both federal and provincial environmental assessment ("EA") (refer to Subsection 20.4.1). The final environmental impact statement / environmental assessment ("EIS/EA") has been completed and reviewed by regulatory agencies, Indigenous communities and the public for review and comment. Thirteen valued components ("VCs") were identified by Greenstone Gold Mines ("GGM") in accordance with the approved EIS Guidelines issued by the Canadian Environmental Assessment ("CEA") Agency as relevant to the effect assessment process in the EA. The VCs are the atmospheric environment, acoustic environment, groundwater, surface water, fish and fish habitat, vegetation communities, wildlife and wildlife habitat, labour and economy, community services and infrastructure, land and resource use, heritage resources, traditional land and resource use, ("TRLU") and human and ecological health. Project interactions with the VCs were analyzed to determine potential environmental effects associated with the Project's construction, operation, and closure phases (refer to Subsection 20.4.1.3). In addition to the VCs, the effects assessment also considered effects of the environment on the Project, accidents and malfunction scenarios and cumulative effects.

Conceptual environmental management and monitoring plans ("EMMPs") were provided in the final EIS/EA, including measures related to both compliance and EIS/EA monitoring for all phases of the Project. The conceptual program (refer to Subsection 20.6) was intended to demonstrate the commitment of GGM, as the proponent, to an appropriate and thorough process of verifying predicted effects from the Project and the effectiveness of mitigation measures. The collective monitoring activities associated with the Project will also be used to inform adaptive management for the Project, if required. The EMMPs were advanced beyond the conceptual stage throughout 2019 and are ‘living’ documents as the Project progresses.

The final EIS/EA also included a Conceptual Closure Plan, described in Subsection 20.7. The Closure Plan has now been advanced to a final version that includes details on closure and rehabilitation. The Closure Plan was approved by the MENDM in January 2020.

The results of the final EIS/EA, including implementing the identified mitigation measures, supports the conclusion that the Project will not cause significant adverse environmental effects, including effects from accidents and malfunctions, effects of the environment on the Project and cumulative effects. There are no issues identified to date that would materially affect the ability of GGM to extract minerals from the Project. Since completing the final EIS/EA, GGM has completed slight modifications of Project components as detailed engineering advances, which form the basis for the final mine plan used for this Report.

Consultation with stakeholders (community members, agencies, interested parties) and Indigenous communities is an integral part of the Project. Active participation through consultation during Project planning helps to achieve an open and transparent process, build trust, enhance awareness of the Project and strengthen the quality and credibility of results. Active consultation has been undertaken throughout Project planning including the preparation of the EIS/EA and permit applications and will continue as the Project progresses. Impact benefit agreements have been established with five Indigenous communities as planned. Consultation and engagement activities are described in Subsection 20.5.

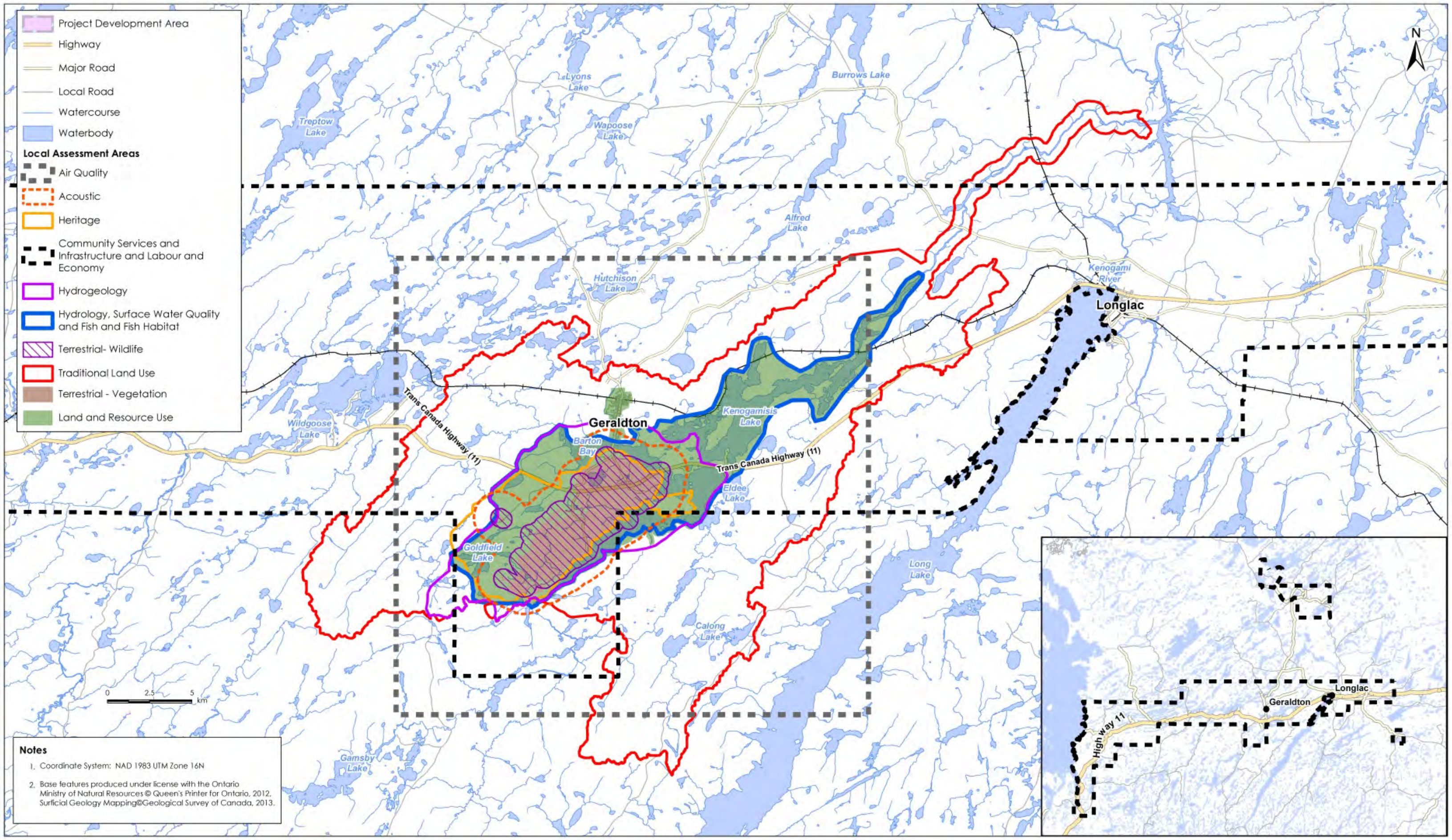
20.2 Environmental Studies

20.2.1 Overview

Baseline environmental studies were completed to characterize the natural, social, economic, cultural and built aspects of the environment that may be potentially impacted by the Project or affect Project design.

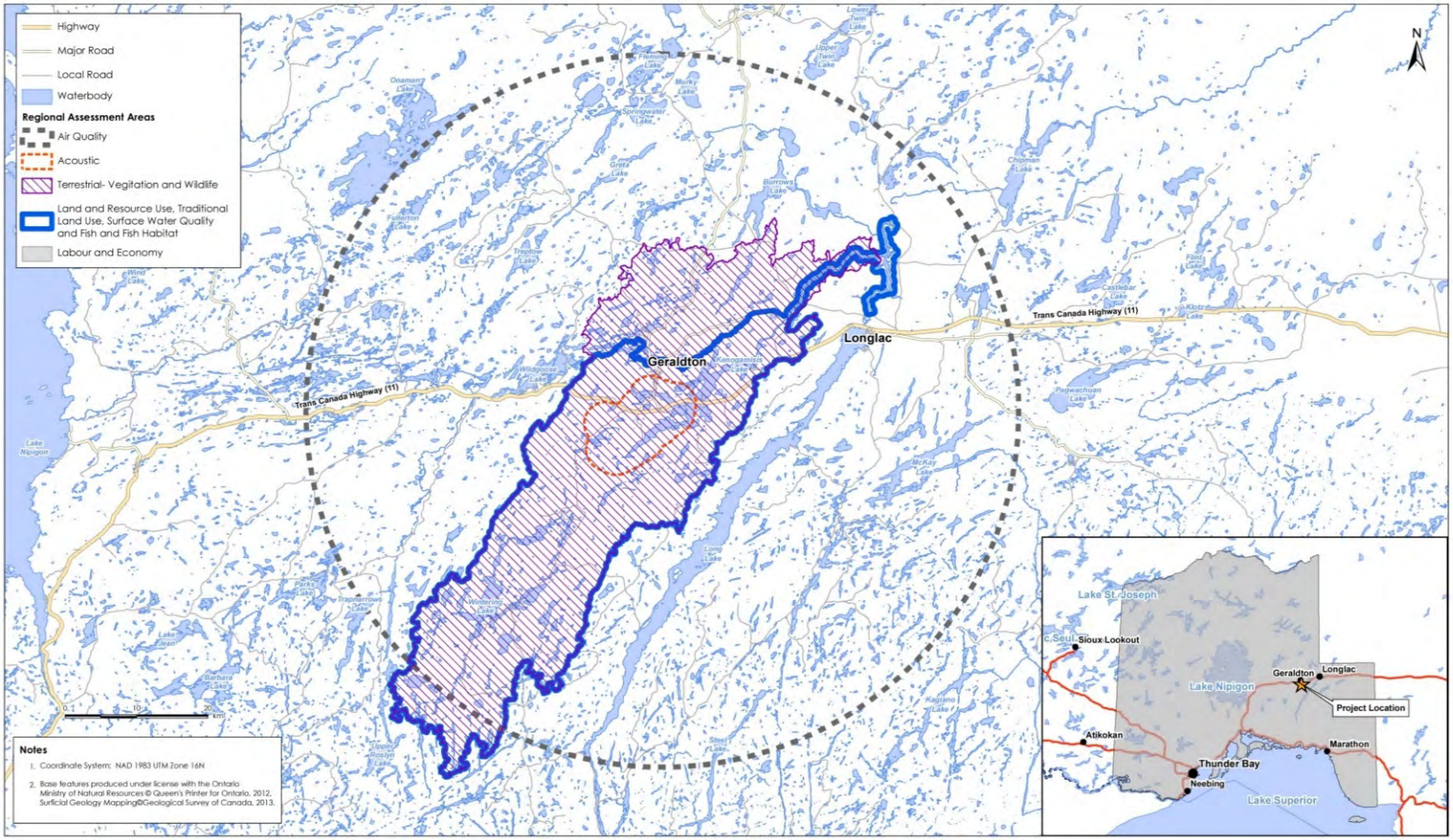
A Project development area (“PDA”) was identified (Figure 20.1) and encompasses the Project footprint and is the anticipated area of physical disturbance associated with the construction and operation of the Project. In addition, local assessment areas (“LAA”) and regional assessment areas (“RAA”) were identified to encompass the areas where there is potential for effects on the environment from the Project (refer to Figure 20.1 and Figure 20.2). Spatial boundaries are defined in each baseline study component, but for the purpose of this Report, the LAAs and RAAs are referred to broadly as the “study area”.

Figure 20.1: Local Assessment Areas



Stantec	**70 Southgate Drive Suite 1 Guelph, ON N1G 4P5**	**Title: Local Assessment Areas**	**Author: Dan Harvey Date: 16/28/04**

Figure 20.2: Regional Assessment Areas



Stantec	**70 Southgate Drive Suite 1 Guelph, ON N1G 4P5**	**Title: Regional Assessment Areas**	**Author: Dan Harvey Date: 16/28/04**

20.2.2 Geology and Geomorphology

20.2.2.1 Physiography

The Project lies within the Boreal Shield, a Canadian ecozone where the Canadian Shield and the boreal forest overlap. Precambrian bedrock at or near the surface plays an important role in shaping the biophysical landscape. Lakes, ponds, and wetlands are abundant in this landscape and drainage patterns are typically dendritic, with sporadic angular drainage as influenced by bedrock outcrops.

Topography in the study area is relatively flat to gently rolling with ground surface elevations ranging from 375 masl in the western portion of the PDA to 335 masl along the shoreline of Kenogamisis Lake. Lower lying areas within the PDA are characterized by swamps and ponds with poor drainage throughout the area. The PDA is bounded to the south, east and north by Kenogamisis Lake, which forms the main watershed within the study area. Local water features and topography where an important consideration in the siting and design of key Project components, including the Tailings Management Facility ("TMF") and associated watercourse diversions and the WRSAs.

20.2.2.2 Surficial Soils and Geology

The surficial soils and geology in the study area are typical of the Boreal Forest region overlying the pre-Cambrian shield in northern Ontario. Soils are relatively young, exhibiting less than 10,000 years of development and consist of organic muck (comprising about 36% of the total area) and well-drained brunisols over thin bedrock (comprising about 35% of the area), with poorly drained gleysols accounting for 13% of the area. The remaining 16% of the PDA is either developed land or water.

Surficial geology consists of large areas of glacial till, outwash and glaciolacustrine sediments, and glaciofluvial deposits. Unique to this area is the presence of a high percentage of calcareous (carbonate rich) substrates. Carbonates are commonly found throughout all modes of soil deposition within the study area. Till and other discontinuous drift (gravelly silty sand to sandy silt) is mapped in the northern and western portions of the PDA, generally near the proposed open pit and northern portion of the TMF. Subaqueous outwash and associated glaciolacustrine sediment (rippled silty fine to very fine sand, silts, and minor clay as thin interbeds) occur along the eastern portion of the PDA, primarily to the south of the open pit in the areas of WRSA D and the southern portion of the TMF. Organic deposits such as peat or muck are present in wetlands and river valleys and are typically between one metre and three metres thick. Ice contact glaciofluvial sediments (sand and gravel) or thick till (gravelly clayey silt to gravelly sandy silt) are located along the western boundary of the PDA and correspond to an esker that extends southwest from Mosher Lake to the eastern reaches of Goldfield Lake.

20.2.2.3 Bedrock Geology

A detailed description of the bedrock geology and controls on mineralization is presented in Section 7 - Geological History and Mineralization.

20.2.3 Acid Rock Drainage/Metal Leaching Potential

A comprehensive geochemical testing program was initiated in 2013 to characterize waste rock, ore, overburden and tailings associated with the Project. Testing included Acid-Base Accounting ("ABA"), Shake Flask Extraction, total metals and laboratory and field kinetic tests with the field kinetic testing program continuing through 2019. The geochemical characterization of waste rock derived from the testing is being used for mine planning and development of a detailed waste rock management plan. The following section presents a summary of the results of the geochemical testing program up to the end of 2018.

Overall, the ore, waste rock and tailings materials contain relatively low Acid Rock Drainage ("ARD") potential but will still require consideration of how to best manage effects from existing Potentially Acid Generating ("PAG") material in the design of these Project components. Overburden will not require management for ARD potential. Measures to mitigate potential effects to water quality due to metal leaching have been documented in the Water Management and Monitoring Plan for the Project.

20.2.3.1 Overburden

Overburden is classified as non-Potentially Acid Generating ("non-PAG") material and is unlikely to generate acidic leachate. The potential for leaching of arsenic, and potentially cobalt and copper, was identified for soils in the area of the historical MacLeod-Mosher and Hardrock plant sites.

20.2.3.2 Waste Rock

Waste rock sample/block was conservatively classified as PAG if C_{total}/S_{total} ratio was below 0.8, which corresponds to a net potential ratio ("NPR") of 2. Up to 1.3 weight percent of PAG material was estimated based on the C_{total}/S_{total} ratios obtained from the block model. PAG waste rock is mainly associated with the sulphide replacement zones. The minimum onset time for ARD conditions is estimated to be 70 years after exposure to the atmosphere. The relatively low percentage of PAG rock and the long ARD onset time provides management flexibility for this material. Co-deposition of PAG and non-PAG waste rock has been identified as the preferred option as outlined in the preliminary Waste Rock Management Plan developed for the Project.

The data from the field kinetic tests were demonstrated to be more reflective of actual site conditions than leaching rates obtained in the laboratory humidity cell data, providing a longer testing period under field conditions for evaluation of long-term leaching behaviour. Results from field kinetic testing indicate that average annual concentrations were above the Schedule 4 criteria under the MDMER for arsenic (0.5 mg/l) in one test (S3-2) containing greywacke and in one test containing mafic intrusives (I1-1). These tests have rock with elevated arsenic grades above average, 110 and 310 ppm, respectively. The following parameters were above the Ontario Provincial Water Quality Objectives ("PWQO") in major rock types:

- Clastic sediments – S (72% of waste rock): the PWQO for arsenic and cobalt and the Interim PWQO for arsenic, antimony, aluminum, and uranium;
- Intrusive rocks – I (11% of waste rock): the PWQO for arsenic and the Interim PWQO for arsenic, antimony, and aluminum;
- Chemical sediments – C (17% of waste rock): the Interim PWQO for arsenic and antimony.

Field leaching rates of these elements generally declined between 2014 and 2018 in samples representing major rock types, indicating a significant reduction in leaching rates over time. Additional laboratory and field testing are being conducted to investigate relationships of arsenic leaching with other parameters to support the development of potential waste rock management strategies. Arsenic and metal leaching under neutral conditions was a key issue evaluated in the final EIS/EA and in the development of the mine plan.

During operation, runoff and seepage from waste rock will be collected and treated prior to discharge to the environment. At closure, contact water from WRSAs will be diverted to the open pit to expedite filling of the open pit.

20.2.3.3 Future Tailings

Ore samples and tailings have similar ABA characteristics before and after metallurgical tests. Ore and tailings also have similar neutralization potential ratio thresholds for ARD classification with PAG tailings estimated at 9.7% with a minimum ARD onset time for PAG tailings of 12 years based on laboratory neutralizing potential depletion rates. These rates are expected to be slower under field conditions and addressed through progressive rehabilitation and closure of the TMF.

In the TMF pond, concentrations of metals and total cyanide are predicted to meet MDMER criteria based on results of ageing tests. Unionized ammonia, cobalt, copper, arsenic, antimony, silver, and free cyanide were identified as parameters of potential concern during operation based on comparison with the PWQO.

Water from the TMF will not discharge directly to the environment and toe seepage will be collected and pumped back to the TMF pond during operation.

At closure, water and seepage collected in the TMF will be sent to the open pit to help expedite filling of the open pit. Once the pit lake is filled and water quality meets acceptable criteria for discharge, the TMF seepage collection facilities will be decommissioned and resulting flows will be directed overland to natural drainage features including the remaining portion of Goldfield Creek, the Goldfield Creek diversion, and Watercourse M.

20.2.4 Atmospheric Environment

The Project is located in a rural location of northern Ontario where air quality is primarily influenced by the Ward of Geraldton and traffic on Highway 11. Measured levels of nitrogen dioxide, sulphur dioxide, and inhalable particulate matter were below their applicable provincial criteria. The maximum measured concentrations of total suspended particles and all metals with Ministry of the Environment, Conservation and Parks (“MECP”) air quality criteria were well below their applicable criteria. The maximum measured concentrations of all volatile organic compounds with MECP air quality criteria were well below their applicable criteria except for benzene, which exceeded the MECP annual average ambient air quality criteria (“AAQC”). Maximum measured polycyclic hydrocarbons (“PAHs”) concentrations were below their applicable criteria except for benzo(a)pyrene (B(a)P), which exceeded the MECP 24-hour and annual average AAQCs. The methodology used to develop the background levels is expected to be conservative and over-estimate actual background levels in the Geraldton area.

20.2.5 Acoustic Environment

The major contributors to baseline acoustical environment were found to be the traffic noise from Highway 11, Michael Power Boulevard and the natural environment. Baseline sound levels were found to be dominated by traffic noise during the daytime and natural environment during the nighttime. No “non-traffic anthropogenic sources” were found to be major contributors to the acoustic environment, and no tonal or excessive low frequency noise was encountered during field studies. The field observations and measurements of baseline noise indicate that the receptors area closer to the roads are characteristic of a Class 2 acoustical environment and the rest of the receptors are with the characteristic of a Class 3 acoustical environment as defined in the MECP NPC-300 guideline.

20.2.6 Groundwater

Field activities to confirm hydrogeological conditions were completed from 2013 to 2019 and included borehole drilling and groundwater monitoring well installation, well development, hydraulic response testing, test pits, drive point piezometer and pressure transducer installation, water level monitoring and groundwater quality sampling.

The overburden and shallow bedrock are considered to be hydraulically connected. Groundwater levels are generally found at 1 m to 2 m below ground surface. Groundwater flow is strongly influenced by topography, which results in localized groundwater flow from topographic highs with groundwater discharge to wetland areas or surface water features. Overall, the regional groundwater flow within overburden is to the southeast toward Kenogamisis Lake. Significant water producing fractures or faults were not encountered during the drilling and testing completed, suggesting that significant water inflow issues from natural fractures or faults are not expected during open pit development. This is supported by the historical underground mining that did not identify significant water inflow issues.

Elevated concentrations of hardness, iron, manganese, and colour were consistently observed at the majority of background monitoring wells in the overburden and bedrock and are typical of groundwater in Ontario and are reflective of the natural mineralization and geochemical processes in the area. Overburden and bedrock water quality away from historical mining areas was generally of good quality with parameters occasionally above the Ontario Drinking Water Quality Standards reflective of location conditions.

There are several historical or existing land uses identified that have contributed to the degradation of water quality in the Project area. The historical MacLeod and Hardrock tailings contain elevated concentrations of arsenic and cobalt above the MECP Aquatic Protection Values. The historical Hardrock tailings water quality is generally similar to the historical MacLeod tailings with the exception of elevated concentrations of cyanide, cadmium, lead, nickel, and silver above the MECP Aquatic Protection Values and a small area of the tailings that is considered acid generating (referred to as the reactive tailings). Seepage from the historical tailings has been identified to affect water quality within Barton Bay and the Central Basin of Kenogamisis Lake, particularly arsenic concentrations, which are well above the PWQO for groundwater within the historical tailings.

The historical underground workings associated with the MacLeod-Mosher and Hardrock mines are currently flooded and will be dewatered prior to and during open pit mining. Water quality associated with Hardrock Shaft No. 1 had concentrations of cobalt that consistently exceeded the PWQO and concentrations of chloride, arsenic, copper, iron, and zinc that at times exceeded the PWQO or Interim PWQO. Water quality associated with the MacLeod-Mosher underground workings was characterized from samples collected at Mosher Shaft No. 1. They had concentrations of iron and zinc that consistently exceeded the PWQO and concentrations of arsenic that consistently exceeded the Interim PWQO. No discharge from the MacLeod-Mosher underground workings currently occurs.

20.2.7 Soil Quality

Soil investigations in the area of the former MacLeod-Mosher and Hardrock plant sites identified elevated arsenic concentrations at approximately 65% of the test pits/test holes and antimony and cobalt at approximately 25% of the test pits/test holes. Various other metals were found to exceed the applicable standards in approximately 10% of the test pits/test holes. Hydrocarbon impacts were identified associated with active and former fuelling stations that will be decommissioned as part of the Project as they are located within the open pit footprint. Hydrocarbon and metal affected soil exceeding applicable criteria will be taken offsite to an approved facility as applicable under the *Environmental Protection Act* and *Ontario Regulation 347*. Additional soil testing was completed in 2018 to support the development of a Soil Management Plan and in 2020 to characterize soil quality in areas of mine infrastructure prior to disturbing the soil as part of construction. A Soil Management Plan is being prepared to provide guidance on the management of excess soil generated during the development and operation of the Project, including soil contaminated by historical activities. A Historical Tailings Management and Relocation Plan is being developed that outlines the management required and sequencing for relocation of historical tailings.

20.2.8 Surface Water

20.2.8.1 Hydrology

The PDA is located in the Kenogamisis River watershed, adjacent to Kenogamisis Lake. The lake is long, narrow, shallow and consists of four main basins referred to as the Southwest Arm, Barton Bay Basin, the Central Basin and Outlet Basin. Water levels within Kenogamisis Lake are controlled by the Kenogamisis Lake Dam, which is operated under the guidance of the Aguasabon River System Water Management Plan. The normal operating water level range for Kenogamisis Lake is between 329.32 masl and 329.70 masl with two Cautionary Compliance Zones to provide flexibility during winter and spring freshet conditions.

The Kenogamisis River is the major river in the study area. Its watershed area upstream of the Southwest Arm of Kenogamisis Lake is 760 km^2 and provides approximately 92% of total inflow into the Southwest Arm of Kenogamisis Lake and 65% of total inflow into the Outlet Basin of Kenogamisis Lake. The flow regime of the Kenogamisis River is similar to other rivers in the area with high spring flows in April–May and low flows in summer (July–August) and winter (November–March).

The two primary permanent watercourses located in the PDA are the Southwest Arm Tributary and Goldfield Creek. The Southwest Arm Tributary is a second order tributary draining directly to Kenogamisis Lake. The main branch of this watercourse originates in a wetland that drains eastward for a distance of approximately 3.3 km before discharging into Kenogamisis Lake. Goldfield Creek is a larger watercourse with a watershed area of 32 km^2. The creek originates at Goldfield Lake and drains in an easterly direction towards Kenogamisis Lake. Goldfield Creek will be diverted to allow construction of the new TMF and connected to the Southwest Arm Tributary. Other areas of the PDA drain towards Mosher Lake and Barton Bay and the Central Basin of Kenogamisis Lake.

20.2.8.2 Surface Water Quality

Baseline water chemistry data were collected monthly or bimonthly from 2013 to 2018 and quarterly since 2018 and compared with historical results spanning almost 40 years of data. Surface water quality was generally moderately hard (moderately high mineral content), circumneutral in pH (mean values of 6.1 to 8.1), with mean total dissolved solids concentrations in the range of 107 mg/L to 131 mg/L and typical of northern Ontario lakes. Nutrient levels tended to be low, except for Barton Bay, which is affected by discharge from the municipal sewage treatment plant. With the exception of arsenic, copper, iron, and lead, metal concentrations in Kenogamisis Lake were present at levels below the Canadian Water Quality Guidelines for Freshwater Aquatic Life (“CWQG”) and PWQO. Seasonal and spatial trends were evident in the data with the lowest concentrations measured during the spring freshet, which increased gradually through the summer and fall. Barton Bay and the Central Basin of Kenogamisis Lake have the highest metal concentrations and are attributed to the effects of historical mine operations and sewage treatment plant discharges in Barton Bay. In lakes and creeks, sampled as unaffected background or reference lakes, most metals concentrations were below CWQGs and PWQOs.

Historical mining activities have contributed to the degradation of groundwater and surface water quality within the area of the PDA. An assessment of arsenic loading to Kenogamisis Lake was completed using a mass balance approach, which provides an accounting of the total arsenic loading on both an individual basin and overall lake perspective. The mass balance calculations indicate that while a small component of flow, the discharge of groundwater from historical tailings represents approximately 60% of the total arsenic load leaving the Outlet Basin, and about 55% of the total load leaving Kenogamisis Lake at the control dam. By the time water from Barton Basin mixes with water from the Central Basin and Southwest Arm, mean arsenic concentrations are at 9 µg/L, just above the Interim PWQO of 5 µg/L, with concentrations remaining similar through the Outlet Basin.

20.2.9 Fish and Fish Habitat

Characterizing fish and fish habitat in the study area included a review of pre-existing background information and the completion of field work in 2013, 2014, and 2015. Field work included the collection of fish habitat, fish community, fish tissue, sediment quality and benthic community data. Lakes within the study area provide cool water habitat with larger lakes such as Kenogamisis and Goldfield Lake providing a diversity of aquatic vegetation, cover and substrate types. Larger lakes also provided greater bathymetric structure (i.e., humps, shoals, flats, etc.).

There was an abundance of potential spawning habitat for Northern Pike and Yellow Perch throughout most lakes within the study area. Important spawning and feeding habitat for species like Walleye and Lake Whitefish was documented where the Kenogamisis River and Magnet Creek flow into Kenogamisis Lake. Important spawning habitat for these species may also be provided by rocky mid-lake shoals in Kenogamisis Lake and Goldfield Lake.

Moderate sized streams such as Goldfield Creek and its main tributary provided a variety of cover types and habitats, although riffle habitat was limited throughout the study area. These streams provided an abundance of potential Northern Pike spawning habitat in adjacent wetlands when they become inundated in the spring. Despite good cover, fish abundance and species diversity were considered low in the study area streams. The exception to this was large numbers of small bodied fish that may use lower stream reaches to spawn.

Shallow, isolated ponds and first order watercourses in the study area likely freeze to the bottom in winter, limiting fish use in these types of habitat. Highly organic substrates and ice cover may also create anoxic conditions in these areas, further limiting fish distribution.

More than 6,080 individual fish, consisting of 24 species were captured during baseline studies between September 2013 and October 2015. No species identified were listed as federal or provincial species at risk (“SAR”), nor are SAR expected to occur in the study area. Game and sustenance fish species, including Walleye, Lake Whitefish, Northern Pike, Yellow Perch and Burbot, were present in Kenogamisis and Goldfield Lakes.

Extensive data on metals in fish tissue from Kenogamisis Lake have been collected by the MECP for more than 30 years. These data were collected for large bodied fish, primarily sport fish. Mean total arsenic concentrations in forage fish were higher than in game fish. There is no standard provincial or federal consumption guideline for arsenic; however, sport fish from the study area did not exceed consumption guidelines published for other countries. Background concentrations of total mercury in Walleye were above the partial restriction guideline for human consumption (0.26 mg/kg). A bioavailability study was completed and concluded that, while the current elevated levels of arsenic and other metals in water and sediments of Barton Bay and the Central Basin may lead to bioaccumulation, the end points examined did not indicate adverse effects on phytoplankton, benthic invertebrates, or fish.

20.2.9.1 Sediment Quality

Sediment samples were collected throughout the study area in 2013 and 2014 to supplement sediment data collected from Kenogamisis Lake in 2011. Copper and arsenic commonly occur in sulphide-based minerals and the Geraldton area is rich in such minerals, so some naturally elevated levels of copper, arsenic and other metals are expected. Arsenic exceeded the MECP Lowest Effect Level ("LEL") and Severe Effect Level ("SEL") at most sampling stations including Lake A-322, Goldfield Lake and Mosher Lake. Within Kenogamisis Lake, the SEL for arsenic was only exceeded in Barton Bay and in the Central Basin. Common parameters that exceeded the LEL in the study area were cadmium, chromium, copper, lead and nickel. Exceedances of the LEL for zinc occurred in individual replicates from the Central Basin and Barton Bay. Additional sediment sampling was carried out in 2019 targeting a larger sample size and statistically unbiased sampling in all six sub-basins of Kenogamisis Lake (126 randomly located stations in all) to obtain a defensible baseline sediment quality dataset prior to construction.

20.2.10 Vegetation Communities

The Project is located along the southern boundary of the Boreal Forest Region, in northern Ontario. Typical forest cover is a mix between deciduous and coniferous forest cover as well as coniferous swamp; vegetation communities are predominantly coniferous with deciduous associates. White and black spruce, tamarack, balsam fir and jack pine are common throughout the area with frequent occurrences of deciduous vegetation communities and species, including white birch, trembling aspen and balsam poplar. Anthropogenic disturbances in the Project area have resulted in a variety of vegetation communities, ranging from open disturbed sites showing early successional growth to mature naturalized deciduous and coniferous forest communities. In the PDA, ecosites were approximately 40% conifer dominated upland forest, 10% hardwood dominated forest, 2% mixed forest, 35% swamp and <2% open wetland (marsh, bog, and fen) communities. The remaining <1% cover was shallow open water. Disturbed ecosite types made up 11% of the PDA.

A total of 253 species of vascular plants were recorded in the study area, of which 91% (230 species) were native and 9% (23 species) were not native species. No plant SAR or Species of Conservation Concern ("SOCC") were recorded in the study area during botanical inventories and are assumed to not be present in the study area. No known Provincially Significant Wetlands or provincially rare communities were identified in the study area. One sensitive, but not provincially designated as rare, wetland community was identified adjacent to the northeast limits of the proposed TMF. Although this ecosite community type (B136) is not listed as a provincially rare vegetation type, it could be considered a sensitive feature due to its size and potential to support habitat for Taiga alpine butterfly ("SOCC").

20.2.11 Wildlife and Wildlife Habitat

Field investigations identified a variety of wildlife species in the study area. Fifteen SAR and SOCC were recorded during baseline surveys for the Project. Of these, eight are confirmed to be either resident or breeding within the vicinity of the Project: barn swallow, bald eagle, Canada warbler, eastern wood-pewee, common nighthawk, northern myotis, little brown myotis and taiga alpine butterfly.

The analysis of results identified a number of terrestrial features and associated wildlife habitat for SAR and SOCC within the study area. The following habitat for SAR and SOCC were identified in the LAA during baseline field investigations:

- Breeding habitat for Canada warbler (SAR) and eastern wood-pewee (SOCC);
- Potential breeding habitat for common nighthawk (SAR);
- Barn swallow (SAR) nests and foraging habitat;
- Bald eagle (SOCC) nests and foraging habitat;
- American white pelican (SAR) stopover and foraging habitat on Kenogamisis Lake;
- Category 3 habitat (MNR 2013b) for woodland caribou (SAR);
- Potential natural bat (SAR) maternity roost habitat;
- Potential anthropogenic bat (SAR) maternity roost habitat;
- Significant Wildlife Habitat (SWH) for waterfowl nesting;
- SWH for moose late winter cover;
- SWH for turtle overwintering;
- SWH for amphibian breeding; and,
- SWH for Taiga alpine butterfly (SOCC).

20.2.12 Labour and Economy

Between 2006 and 2011, the population of Ontario increased by approximately 5% while the populations of the Thunder Bay District and the Municipality of Greenstone decreased by approximately 2% and 4%, respectively. Available population projections indicate that the municipality will continue to see population decline, with an estimated population of 4,618 residents in 2018 and 4,480 residents in 2023.

The Northwestern Ontario economic region includes the Districts of Thunder Bay, Rainy River and Kenora. Spatially, this is the largest economic region in the province, while also having the smallest population of all Ontario economic regions. Mining is a key component of the economy in Northwestern Ontario with over 80 active exploration projects during 2012, as well as six operational mines.

Key industries providing employment locally in the Municipality of Greenstone include trades; transport and equipment operations; processing, manufacturing and utilities; agriculture and resource-based industries, including mining and forestry. Baseline economic conditions indicate that the Greenstone economy has been in decline, with the number of people in the labour force decreasing by 11% between 2006 and 2011, and the unemployment rate increasing by two percentage points. In comparison, the size of the labour force in the District of Thunder Bay decreased by 4% over the same period, while the Ontario labour force increased by 4%. Within the Municipality of Greenstone, there are higher rates of unemployment in Indigenous communities than in non-Indigenous communities.

The Thunder Bay District is expected to experience a shortage of skilled workers for mining projects, primarily because there is a lack of younger people with appropriate skills coming into the regional labour market. Increased recruitment and retention challenges are also anticipated as competition for workers increases.

20.2.13 Community Services and Infrastructure

The Town of Geraldton, centrally located in the Municipality of Greenstone, is the service support centre for the surrounding region including government services (MNRF/Regional Fire Management), Medical Services (District Hospital), financial services and retail. Overall, the Project is located relatively close to existing municipal and provincial services, including water and wastewater, waste, transportation, power, recreational and emergency services. Key local community services and infrastructure in the study area include:

- Municipal features, including a park, public boat launches and public beaches, among other urban land uses;

- Kenogamisis Golf Club;
- MacLeod and Hard Rock townsites;
- Hydro One infrastructure, including a substation and power lines;
- Discover Geraldton Interpretive Centre;
- Highway 11 and Michael Power Boulevard;
- Gas station;
- MacLeod-Cockshutt Mining Headframe; and,
- Ontario Provincial Police station.

Some municipal services and infrastructure have been reported to be at or near capacity, including wastewater systems and solid waste facilities. Greenstone is designated as an underserviced area by the Ministry of Health and Long-term Care, which allows the community to access incentive funds for the recruitment and retention of family physicians. Primarily though, it means that the existing community is underserviced by health care professionals. Meanwhile, due to population decline, there has been a surplus of housing in some communities in the Municipality of Greenstone and there are some underdeveloped designated residential areas to accommodate larger-scale future growth in the Project vicinity, including in Beardmore, Longlac, Nakina and Geraldton.

20.2.14 Land and Resource Use

Land and resource use has been shaped by mining and forestry activity. In the early 1930s, the region became known for gold mining; however, extraction ceased during the 1970s leaving forestry as the main industry and land use in the region. Today, the most extensive land uses in the area are forestry, mineral exploration, hunting, trapping and fishing. Land and resource use areas and facilities in the local area include:

- MacLeod Provincial Park, which includes a campground, walking trail, cross-country skiing trails and public beach;
- Ward of Geraldton, which includes a municipal park, public boat launches and public beaches, among other urban land uses;
- Barton Bay Wildlife Trail;
- Public access points from Lahtis Road;
- Crown land campsite;

- One guide outfitter within the LAA (Kenogamisis Lake Resort);
- Snowmobile trails;
- Navigation routes;
- Planned forest harvest areas and forest access roads;
- Trapline areas (two in the PDA: GE021, GE022);
- Bear Management Areas (two in the PDA: GE-21A-032, GE-21A-027); and,
- Bait Harvesting Areas (four in the PDA: NI5035, NI5036, NI5027, NI5028).

20.2.15 Heritage Resources

20.2.15.1 Archaeology Resources

A Stage 1 Archaeological Assessment was completed for the Project to compile all available information about the known and potential archaeological heritage resources within the study area and to provide specific direction for the protection, management or recovery of these resources. A Stage 2 assessment was subsequently completed for areas of archaeological potential, including areas near water sources, transportation routes, and townsites. The Stage 2 assessment concluded that no archaeological resources were found in the PDA with no further archaeological assessments recommended.

20.2.15.2 Architectural/Historical Resources

A Cultural Heritage Evaluation Report has been completed to screen for resources of potential cultural heritage value or interest ("CHVI"), as defined by Ontario Regulation 9/06. Twenty-nine heritage resources were identified on properties which may be affected by the Project, the vast majority in residential developments constructed by mining companies. Of these, 18 were determined to be situated within the PDA.

20.2.16 Traditional Land and Resource Use (TLRU)

TLRU includes traditional activities, sites, and resources identified by Indigenous communities. Project engagement activities and the review of Project-specific traditional knowledge ("TK") studies, land use survey results, and existing literature have confirmed the potential for Project effects on TLRU. One gathering site (a former family settlement) was identified immediately north of the PDA. Campsites, cabins, and sacred sites were also identified in the regional assessment area but not within the PDA. Traditional activities (e.g. hunting, fishing and trapping) also occur in the PDA, LAA, and RAA.

Project-specific TK studies were completed by Métis Nation of Ontario, Eabametoong First Nation, Animbiigoo Zaagi'igan Anishinaabek, Ginoogaming First Nation, LL #58 First Nation, and Aroland First Nation. These studies provide information on the community values associated with the PDA and inform decision-making on the Project. A review of the information provided by these Indigenous communities, as well as secondary sources and consultation, informed the identification of potential Project effects on TLRU.

20.3 Environmental Constraints

The Project is located within an area bounded by Kenogamisis Lake to the north, south and east, with wetland and low-lying areas and associated surface water features to the west. These constraints have been incorporated into the design of the Project, which has focused on minimizing the environmental footprint of the Project while respecting environmental features and required setbacks.

GGM currently owns or has purchase agreements or purchase arrangements in principal with all property owners in the Project Development Area. These include provincial infrastructure related to the MTO patrol yard, Hydro One transmission and distribution power lines and associated substation, OPP station, the Discover Geraldton Interpretive Centre, properties within the MacLeod and Hard Rock townsites and Dan's General Store (Husky Gas Station). An environmental screening report has been completed by TBTE (TBTE, 2015) for the proposed Highway 11 realignment, which included the MTO patrol yard, Mosher portal area, historical MacLeod tailings and the MacLeod Mine landfill. A soil management plan is available and will be implemented during construction. The potential for soil and groundwater impacts were identified at the MTO patrol yard based on typical land use; however, comprehensive investigations have not been completed at this time. Modified Phase 1/2 Environmental Site Assessments (TBTE, 2015) were completed for a former gas station property (former Larry's Esso) and the current gas station property located at the intersections of Highway 11 and Michael Power Boulevard. Soil impacts associated with petroleum hydrocarbons and arsenic and groundwater impacts associated with petroleum hydrocarbons were identified. A Soil Management Plan is being prepared to provide guidance on the management of excess soil generated during the development and operation of the Project.

Historical mining activities have contributed to the degradation of groundwater and surface water quality within the area of the PDA, particularly with respect to the historical tailings. As discussed in Subsection 20.4.1.3.4, it is anticipated that the Project will result in an improvement in water quality within Kenogamisis Lake, having a positive effect on arsenic and iron concentrations due to the reduction in groundwater discharge associated with the historical MacLeod and Hardrock tailings. This will be achieved through removal of a portion of the historical tailings for placement within the newly constructed TMF, installation of seepage collection around the historical tailings as part of the berm and buttress construction to address long term physical stability, improved cover design for the remaining historical tailings and changes in groundwater flow during operations that will allow impacted groundwater to be captured within the open pit and treated prior to discharge.

Historical mine openings exist within the PDA and are currently capped or secure. The condition of the caps and security of the existing mine openings have been evaluated with respect to the requirements of O. Reg. 240/00 during preparation and upgrades will be completed as required during closure. For the majority of the mine openings, they will be removed during development of the Project and, as a result, a limited number of openings will remain at closure.

Nine provincial SAR or their habitats have the potential to occur on site: American white pelican (*Pelecanus erythrorhynchos*), bald eagle (*Haliaeetus leucocephalus*), bank swallow (*Riparia riparia*), barn swallow (*Hirundo rustica*), common nighthawk (*Chordeiles minor*), eastern whip-poor-will (*Caprimulgus vociferous*), little brown myotis (*Myotis lucifugus*), northern myotis (*Myotis septentrionalis*), and woodland caribou (*Rangifer tarandus caribou*). These species and their habitats are protected by the *Endangered Species Act*, 2007 with authorizations being provided by MNRF as required during permitting to develop these lands. GGM will submit applications for the appropriate authorizations to the MECP prior to Project development (refer to Subsection 20.4.2).

Development of the Project will alter existing activities and facilities within the PDA, including the MacLeod-Cockshutt Mining Headframe, the Discover Geraldton Interpretive Centre and the Kenogamisis Golf Club. An agreement has been signed between the Municipality and GGM to support the Municipality's future plans with respect to the removal of these facilities.

MacLeod Provincial Park is located 350 m east of the PDA. There are no other provincially or federally protected areas such as national parks, protected areas, ecological reserves, or conservation reserves near the Project. There are no Areas of Natural and Scientific Interest, or evaluated Provincially Significant Wetlands within or near to the Project site. One sensitive, but not provincially designated, rare, fen community was identified immediately adjacent to the PDA. There are no areas of archaeological resources identified through baseline studies at the Project site.

20.4 Environmental Approval Requirements

20.4.1 Environmental Assessment

20.4.1.1 Overview

Federal EA is regulated under the *Canadian Environmental Assessment Act 2012* ("CEAA 2012") and is administered by the CEA Agency. Under CEAA 2012, "designated" projects included in the *Regulations Designating Physical Activities* require a federal EA. The Hardrock Project has been confirmed as a Designated Project and a federal EA was implemented in accordance with the approved EIS Guidelines issued to GGM by the CEA Agency on August 5, 2014, with subsequent amendments on February 11, 2016 to include consideration of greenhouse gas emissions and February 12, 2016 related to changes in the list of Indigenous communities with which GGM is expected to engage.

Under Ontario's *Environmental Assessment Act* ("EAA"), mining development projects are not subject to provincial individual EA requirements because they are carried out by private sector proponents. GGM entered into a Voluntary Agreement with the MECP to make the entire Project subject to a single individual EA process in accordance with the approved Terms of Reference ("ToR") received from the province. A final ToR was submitted to the MECP on January 2, 2015, and an editorial amendment submitted on March 31, 2015 for completion of the provincial individual EA under the EAA. The final ToR was approved with amendments on June 24, 2015; it provided the framework for the individual EA and outlined key steps and requirements to undertake an EA process and prepare an EA report compliant with the EAA.

GGM completed a coordinated EA to address both federal and provincial EA requirements through a single process, which resulted in the filling of a single body of information (EIS/EA document) that addressed both provincial and federal EA processes. The final EIS/EA was submitted to the CEA Agency, MECP, Indigenous communities and public in July 2017. GGM completed consultation events with the regulatory agencies as well as the Indigenous communities and local community to present the final EIS/EA and solicit input and comments. Following receipt of all comments on the final EIS/EA, a supplemental information package was published in August 2018. Federal approval of the EIS/EA was received in December 2018 and provincial approval was received in March 2019. GGM is now advancing other environmental approvals required for the Project as described in Subsection 20.4.2.

20.4.1.2 Consultation

Consultation is a key component of both federal and provincial EA processes to engage interested parties to identify and address concerns with Project planning and implementation. Consultation with government, Indigenous communities and the public has been ongoing since before the formal start of the EA processes, and has included opportunities to review Project information and provide input at key stages in EA development. GGM's consultation program reflects the requirements of the federal EIS Guidelines and approved provincial ToR. Refer to Subsection 20.5 for further details regarding consultation and engagement activities undertaken in support of the Project.

20.4.1.3 Preliminary Effects Assessment

The methods that were used to conduct the environmental effects assessment were designed to meet the combined requirements of CEAA 2012 and the EAA. These methods are based on a structured approach that, particularly:

- Considers the federal and provincial regulatory requirements for the assessment of environmental effects as defined by CEAA 2012 and the EAA, with specific consideration of the requirements of the ToR and EIS Guidelines;
- Considers the issues raised by the public, Indigenous communities and other stakeholders during consultation and engagement activities conducted to date;
- Focuses on issues of greatest concern that arise from the above considerations;
- Considers existing environmental conditions of the area, particularly historical activities and resulting environmental effects that might have affected baseline conditions;
- Integrates engineering design and programs for mitigation and monitoring into a comprehensive environmental planning and management process that will be applied during the design and implementation of the Project;
- Considers the Project in a careful and precautionary manner, to avoid significant adverse environmental effects.

The environmental effects assessment methods addressed both Project-related and cumulative environmental effects based on the Project description at the completion of the final EIS/EA. Project-related environmental effects and cumulative environmental effects are assessed using a standardized methodological framework for each VC. Two conditions must be met to initiate an assessment of cumulative effects on a VC: firstly, the Project is assessed as having adverse residual environmental effect on a VC; and secondly, the adverse residual effects from the Project overlap spatially or temporally with residual effects of other physical activities on a VC. Cumulative environmental effects are assessed to determine whether they could be significant, and to consider the contribution of the Project to them.

The following sections present a summary of the environmental effects assessment, proposed mitigations and determination of significance for each VC from the final EIS/EA.

20.4.1.3.1 Atmospheric Environment

The potential environmental effects of the Project on the atmospheric environment include change in ambient air quality, climate change and change in lighting. As part of the assessment, mitigation measures were identified that will be applied to the Project to avoid or reduce effects.

For construction, operation and closure, mitigation measures include: implementing a fugitive dust best management plan; using dust suppressants; maintaining vehicles, implementing a ‘no idling’ policy to reduce emissions; applying speed limits to reduce dust from vehicles travelling on gravel roads; minimizing of haul routes to reduce vehicles use, locating portable lighting equipment where, to the extent feasible, it is not visible at nearby receptors; and the use of directional light fixtures to avoid the transmission of light outside of the PDA.

During operation, additional mitigation measures will include: equipping primary and secondary crushers with a dust collection system; enclosing the mill feed storage area; using scrubbers on grinding operations and the induction furnace; managing fugitive dust emissions; limiting off-site light effects through the use of downlighting; and implementing a greenhouse gas (“GHG”) management plan to minimize and track GHG emissions. In addition, new mobile equipment onsite will meet applicable Transport Canada off-road vehicle emission requirements.

With mitigation in place, air quality emissions resulting from construction and closure are expected to be temporary and within applicable regulatory objectives, standards and guidelines. Overall, the Project’s contribution to total Canadian annual GHG emissions would be up to 0.04% (based on 2014 GHG emission levels). Short-term GHG emissions from equipment are expected during construction and closure. During operation, the Project is expected to emit no more than 264 kt of CO_2e per year.

The Project baseline conditions were characterized as being rural (characterized by low district brightness) regarding lighting. The change in ambient lighting outside the Project boundary during operation is expected to be within guidelines for rural areas.

Residual adverse environmental effects on the atmospheric environment were determined to be not significant.

20.4.1.3.2 Acoustic Environment

The potential environmental effects of the Project on the acoustic environment include change in noise and change in vibration levels. As part of the assessment, mitigation measures were identified that will be applied to the Project to mitigate effects.

During construction and closure, major construction activities will be scheduled during daytime hours (e.g. 07:00 to 19:00), where possible, to avoid impact during sensitive nighttime periods. Other noise mitigation measures include installing muffler systems for the combustion exhausts, properly maintaining equipment, and implementing a complaint procedure to address any noise complaints should they arise. Preliminary blast design will meet the MECP's criteria for noise and vibration and all blasting will occur during the daytime.

During the operation phase, mitigation measures include selecting quieter equipment and/or designing acoustical enclosures and louvres to limit overall noise emissions and equipping generator inlets, radiator exhausts and combustion and exhaust stacks in the powerhouse with silencers.

With mitigation measures in place, predicted sound levels are expected to meet regulatory requirements at all Points of Reception and Points of Interest. The magnitude of vibration effects from Project-related activities is not predicted to exceed applicable guideline criteria and thresholds.

Residual environmental effects on the acoustic environment were determined to be not significant.

20.4.1.3.3 Groundwater

The potential environmental effects of the Project on the groundwater include change in groundwater levels and/or flow and change in groundwater quality. As part of the assessment, mitigation measures were identified that will be applied to the Project to avoid or reduce effects.

Mitigation measures for groundwater quantity and flow include: using standard management practices throughout the Project, including drainage control and excavation and open pit dewatering; limiting the construction footprint (i.e., PDA) to the extent possible to reduce the potential for reductions in groundwater recharge and limit the number of watersheds overprinted by the PDA; Use standard construction methods, such as seepage cut-off collars, where trenches extend below the water table to mitigate preferential flow paths; return water generated from historical underground dewatering (with treatment at the ETP as required) to Kenogamisis Lake during operation to offset a reduction in groundwater discharge and considering accelerating open pit filling at closure to re-establish groundwater levels to near pre-mining conditions as quickly as possible.

Mitigation measures for groundwater levels and flow include: implementing progressive rehabilitation (placement of vegetated soil cover) to reduce infiltration into the WRSAs and TMF; designing the WRSAs to increase the amount of runoff and reduce the amount of infiltration through the WRSAs; removing approximately 22% of the historical MacLeod tailings and 77% of the historical Hardrock tailings as well as contaminated soil from historical process plant areas; installation of a subsurface seepage collection system around a portion of the base of the historical MacLeod high tailings to collect seepage and groundwater recharge from the tailings; enhancing the cover over the remaining historical MacLeod high tailings; constructing runoff and seepage collection ditches and ponds around the overburden storage area, ore stockpile, WRSAs and the TMF; and implementation of cyanide detoxification technology to reduce cyanide concentrations and precipitate metals at the process plant, resulting in an improvement in water quality within the TMF.

Regarding groundwater levels and flow, the water table will be lowered in the local area due to dewatering of the open pit, however, there are no groundwater supply users within the area affected and the lands are owned or under lease by GGM. Groundwater quality is predicted to meet regulatory criteria at the point of discharge. In addition, there will be an overall reduction in loading to surface water features as a result of the removal of a portion of the historical tailings and changes in groundwater flow resulting in a positive change. Arsenic loading from groundwater discharge to surface water bodies is predicted to decrease by 99% during operations and 59% during closure.

Residual adverse environmental effects on groundwater were determined to be not significant.

20.4.1.3.4 Surface Water

The potential environmental effects of the Project on the surface water include change in surface water quantity and change in surface water quality. As part of the assessment, mitigation measures were identified that will be applied to the Project to avoid or reduce effects.

Mitigation measures related to surface water quantity and quality include, but are not limited to: limiting the Project footprint to the extent practicable to reduce contact water volumes and management requirements; maintaining existing drainage patterns with the use of culverts; maintaining access roads to improve water flow, reduce erosion and manage vegetation growth; inspecting culverts periodically and removing accumulated material and debris; designing the Goldfield Creek diversion channel to convey the peak flow; implementing progressive rehabilitation to reduce infiltration into the WRSAs and TMF; improving water quality within the TMF through cyanide detoxification; designing water management and storage infrastructure to control peak discharges to surface water; reusing contact water to reduce freshwater intake, effluent treatment and discharge requirements and treating effluent prior to discharge; and implementing progressive water management over the life of the mine including development of drainage controls for areas only prior to the development and expansion of these features.

Regarding water quantity, changes in drainage patterns, including from the Goldfield Creek diversion, will be contained within the LAA with flow continuing to the Southwest Arm, there will be limited changes to flows in Kenogamisis Lake and the flow will be within the range of background variability. With the design and mitigation for the Goldfield Creek diversion, a significant effect on water quantity is not predicted from the Project.

Regarding water quality, mine effluent discharge is predicted to meet baseline concentrations or PWQO within a relatively small mixing zone that does not extend beyond the Southwest Arm of Kenogamisis Lake. The removal and capping of historical MacLeod and Hardrock tailings and the subsequent reductions in groundwater discharge due to the Project are predicted to result in a decrease in arsenic concentrations in Barton Bay, Central Basin and Kenogamisis Lake Outlet Basin during operation. Overall, the Project is anticipated to result in an improvement in water quality within Kenogamisis Lake, having a positive effect on arsenic, sulphate and iron concentrations in Barton Bay, and a positive effect on arsenic in Central Basin and Outlet Basin due to the reduction in groundwater discharge associated with the historical MacLeod and Hardrock tailings.

Residual adverse environmental effects on surface water were determined to be not significant.

20.4.1.3.5 Fish and Fish Habitat

The potential environmental effects of the Project on fish and fish habitat include lethal and sub-lethal effects on fish, permanent alteration of fish habitat and loss of fish habitat. As part of the assessment, mitigation measures were identified that will be applied to the Project to avoid or reduce effects.

Mitigation measures for fish and fish habitat include, but are not limited to: project design (i.e. avoiding sensitive fish habitats where reasonably feasible); managing construction effects on fish by working “in the dry” by isolating work areas; performing fish salvages to transfer fish from work areas, and complying with in-water timing restrictions; implementing an offsetting plan for impacts to fish that cannot be fully mitigated; developing and implementing effluent discharge criteria; and designing water intake and effluent outfalls to prevent entrainment or impingement of fish. GGM will implement a Spill Prevention and Response Plan, Aquatics Effects Management and Monitoring Plan and follow DFO guidelines for the use of explosives near water.

Fish mortality can be avoided during all phases of the Project such that there is no substantive residual effect on fish mortality. The Project has been designed to reduce the potential for causing fish mortality through avoidance and mitigation measures. Alteration of fish habitat will result from changes to flow and drainage. Project designs have reduced effects on local waterbodies such that significant effects on fish and fish habitat are not anticipated. Effects on sustainability and productivity of fish habitat within the local area are not anticipated. Approximately 6.58 ha fish habitat will be lost or permanently altered, much of which consist of marginal or degraded habitat (e.g. ephemeral watercourses, artificial ponds and roadside ditches). The creation of new fish habitat in conjunction with the diversion of Goldfield Creek will offset the potential effects on fish and fish habitat.

Residual adverse environmental effects on fish and fish habitat were determined to be not significant.

20.4.1.3.6 Vegetation Communities

The potential environmental effects of the Project on vegetation communities include change in abundance of vegetation communities, change in function, connectivity and quality of vegetation communities and change in abundance of plant species of interest. As part of the assessment, mitigation measures were identified that will be applied to the Project to avoid or reduce effects.

The primary mitigation for vegetation communities is progressive rehabilitation of the PDA which will commence at the end of construction. In addition, GGM will implement a Biodiversity Management and Monitoring Plan designed to mitigate adverse effects on vegetation and wetlands during construction and operation, including timely restoration of affected vegetation communities, control of invasive species and protection of sensitive species. Other mitigation measures include those implemented to reduce effects from dust and sedimentation and effects on groundwater drawdown or surface water supply to mitigate effects on wetlands. In addition, where there is interest, GGM will provide opportunities to local communities for harvesting of plants for traditional purposes prior to construction.

With regard to change in abundance of vegetation communities although it was estimate that the removal of approximately 1,133 ha of upland vegetation communities and 810 ha of wetland vegetation communities in the PDA will be required, given that the community types that will be removed are generally common and widespread in the RAA, the loss of the community types in the PDA is not predicted to jeopardize the long-term viability of the community types.

With regard to change in function, connectivity and quality of vegetation communities, changes in surface water flow/drainage, dust deposition, the introduction of invasive species, fragmentation and groundwater drawdown may affect vegetation and wetland communities; however it is not expected to threaten the long-term viability of a vegetation community type in the regional area.

With regard to change in the abundance of plant species of interest, clearing of vegetation during construction will result in the removal of plant species of interest to Indigenous communities; however, plant species in the PDA are common species throughout the region and there is potential to incorporate plant species of interest to Indigenous communities during rehabilitation, where use and establishment of these species is appropriate and technically feasible. No plant SAR or SOCC were recorded in the PDA.

Residual adverse environmental effects on vegetation communities were determined to be not significant.

20.4.1.3.7 Wildlife and Wildlife Habitat

The potential environmental effects of the Project on wildlife and wildlife habitat include change in wildlife habitat, change in mortality risk and change in movement. As part of the assessment, mitigation measures were identified that will be applied to the Project to avoid or reduce effects.

Mitigation of potential Project effects on wildlife and wildlife habitat will be accomplished through the implementation of a Biodiversity Management and Monitoring Plan and the progressive restoration of vegetation communities and wildlife habitat. GGM will obtain any required authorizations under the *Endangered Species Act* and adhere to applicable timing windows. Additionally, mitigation measures proposed for other VCs (e.g. vegetation communities) or as part of other environmental management and monitoring plans (e.g. noise) will either directly or indirectly reduce effects on wildlife.

Effects on SAR and significant wildlife habitat are not predicted to affect the sustainability of wildlife within the region and will be partially reversible following closure. In addition, indirect effects from habitat avoidance due to sensory disturbance will be reversed following the completion of active closure activities. Project effects will not result in the irreversible loss of critical habitat for a species listed on Schedule 1 of the *Species at Risk Act.*

With regard to change in wildlife mortality risk, with the implementation of mitigation measures, the residual adverse effect on wildlife mortality is predicted to be within the normal variability of baseline conditions and is not expected to affect the long-term persistence or viability of wildlife within the region Project effects will not result in the permanent, irreversible loss of a species listed on Schedule 1 of the *Species at Risk Act* or listed as threatened or endangered under *Endangered Species Act.*

While the Project will affect existing wildlife movement in the local area, the effects will be limited spatially and temporally and new wildlife movement patterns are predicted to be established in response to rehabilitation within the PDA.

Residual adverse environmental effects on wildlife and wildlife habitat were determined to be not significant.

20.4.1.3.8 Labour and Economy

The potential environmental effects of the Project on labour and economy include changes in both. As part of the assessment, mitigation measures were identified that will be applied to the Project to avoid or reduce effects.

It is expected that the Project will result in positive effects on labour by employing local and Indigenous workers and reducing the unemployment rate in the local area. These positive effects do not require mitigation, but GGM commits to implementing various mechanisms for enhancing Project benefits through construction and operation such as: posting job qualifications and identifying available training programs and providers so that local and Indigenous residents can acquire the necessary skills and qualify for potential employment; working with local and Indigenous businesses to enhance the opportunity to participate in the supply of goods and services for construction and operation; working with the affected local communities to develop training programs oriented to operational needs; and implement the Project's labour and training framework, which includes partnerships with Indigenous communities and education institutes, information sharing (e.g. skills databases) and employment preparation and training. During closure, GGM will establish a skills inventory to be retained for active closure, support re-training to establish transferable skills, provide opportunities for voluntary redundancies during ramp-down (e.g. early retirement), provide redundancy payments, and provide job search assistance. Standard mitigation measures related to the loss of timber by salvaging merchantable timber in accordance with provincial requirements will be implemented. GGM will continue to communicate with the enhanced Forest Resource Licence holder to obtain an Overlapping Agreement and to harvest the trees under their pulp mill license. GGM has consulted with the municipality and developed an agreement to mitigate potential adverse effects on tourism resulting from the removal of existing structures, in particular the Kenogamisis Golf Club, MacLeod-Cockshutt Mining Headframe and the Discover Geraldton Interpretive Centre.

The overall Project effect on labour and economy is positive given the direct, indirect and induced benefits of Project expenditures. The Project will result an increase in the size of the labour force and reductions in the unemployment rate. The Project is also anticipated to result in increases in household incomes, increased opportunities for local and Indigenous businesses and contributions to municipal taxes.

Residual adverse environmental effects on labour and economy were determined to be not significant.

20.4.1.3.9 Community Services and Infrastructure

The potential environmental effects of the Project on community services and infrastructure include change in capacity of housing and accommodations, change in capacity of municipal and provincial services and infrastructure, and change in capacity of transportation services and infrastructure. As part of the assessment, mitigation measures were identified that will be applied to the Project to avoid or reduce effects.

The mitigation measure for housing and accommodation is the use of a temporary camp to accommodate the peak number of construction workers. Mitigation measures for municipal and provincial services and infrastructure include maintaining communication with relevant agencies and organizations to provide information and identify and address potential Project-related implications for services and infrastructure, and to support responsible organizations in planning for, adapting to, or benefitting from Project-related changes in demand. GGM will offer its employees an Employee Assistance Program and require pre-employment physicals. Project workforce education to encourage healthy lifestyle choices, sensitivity training and strict enforcement of GGM’s health and safety policies will also help mitigate potential adverse social effects. Mine rescue vehicles and trained First Responders will be available at the Project site and new employees will be required to take mandatory safety orientations. Employees will be trained in fuel handling, equipment maintenance, fire prevention and response measures. The Project site will be controlled through security measures.

Mitigation for recreation and entertainment services and infrastructure includes providing the temporary camp with dining services and a basic recreational area to accommodate the peak number of construction workers. GGM will maintain the Kenogamisis Golf Club clubhouse and the front nine holes and act in accordance with the agreement developed with the municipality regarding future plans for the MacLeod-Cockshutt Mining Headframe, the Discover Geraldton Interpretive Centre and the golf course. Further mitigation measures for provincial and municipal services and infrastructure include providing notice to the local school board regarding scheduling and human resources planning for the school board to prepare for the enrollment of additional students. To mitigate effects on local infrastructure and utilities, GGM will bus construction workers to and from the temporary camp to limit Project-related traffic, use an on-site natural gas-fueled power plant and electrical/recovered heat distribution system to supply heat and power for Project operation and have Project-dedicated sewage treatment facilities. Throughout the Project, GGM will schedule arrivals/departures of employee traffic to occur earlier than the existing observed a.m. peak hour for local traffic and later than the existing observed p.m. peak hour as well as schedule alternating work shifts so that all workers do not arrive in and leave the area at the same time to limit Project-related demands on both highway and air services and infrastructure. A third-party sewage disposal contractor will provide portable washroom facilities during early construction until the STP and sewage discharge line is set up and during active closure when facilities are decommissioned.

Residual adverse environmental effects on community services and infrastructure were determined to be not significant.

20.4.1.3.10 Land and Resource Use

The potential environmental effects of the Project on land and resource use include change in recreational land and resource use, change in commercially based land and resource use and change in navigation. As part of the assessment, mitigation measures were identified that will be applied to the Project to avoid or reduce effects.

Mitigation measures to reduce adverse effects on recreational land and resource use include: initiating revegetation as soon as practical after Project components are no longer needed; providing in-kind support to assist Greenstone Snowmobile Club in improving the existing trail to Longlac; where possible in accessible areas (e.g. along cleared rights-of-way), leaving trees and other vegetation in place to buffer the view of Project components, reducing the change in viewshed and muffling nuisance noise; siting the majority of Project components so as to achieve a 120 m setback for the surface rights reservation area on claim to lease lands and a 30 m high water mark setback for patent lands (existing vegetation will remain in these areas); removing construction-related buildings, access roads and laydown areas following construction; and implementing progressive rehabilitation works, including stabilization and rehabilitation of aggregate source areas, the north cell of the TMF, plateaus and benches of WRSAs A, B, and C and the overburden storage areas. Rehabilitation will be designed to meet desired end land uses, end land uses will be identified in the Closure Plan, in consultation with agencies, stakeholders and Indigenous communities, as the Project progresses. Mitigation measures related to the atmospheric environment (Subsection 20.4.1.3.1, acoustic environment (Subsection 20.4.1.3.2, fish and fish habitat (Subsection 20.4.1.3.5) and wildlife and wildlife habitat (Subsection 20.4.1.3.7) are also considered related to land and resource use.

In addition to the mitigation measures listed above for recreational land and resource use, to mitigate potential effects on commercially based land and resource use, GGM will maintain access to mining claims located on the peninsula east of the PDA.

To mitigate potential adverse effects on navigation, GGM will use established watercourse crossings and avoid obstructions to navigation; undertake construction activities in a way to prevent debris from flowing into a navigable waterbody; and implement the mitigation measures related to surface water quantity (refer to Subsection 20.4.1.3.4).

GGM will continue to meet with affected tenure holders on a regular basis (i.e. semi-annually) to discuss issues and concerns and to provide Project updates as well as continue discussions regarding accommodation for lost trapping areas with trapline licence holders. GGM will continue to consult with MNRF and the eFRL holder to address, to the extent possible, access to the PDA and the harvest of Crown timber that will be removed as part of site preparation. Timber removal will be completed in accordance with the *Crown Forest Sustainability Act* and *Crown Timber Act* and GGM will seek a Release of Tree Reservation under the Public Lands Act to remove trees on patent lands which have timber rights reserved to the Crown. The Project will be designed to use established watercourse crossings and avoid obstructions to navigation, and signs will be posted at locations around the perimeter of the PDA to alert local land and resource users of the presence of the Project and its components to alert boaters of the treated effluent discharge location. GGM is committed to maintaining alternate access within the PDA to the Southwest Arm of Kenogamisis Lake during construction and operation. Rehabilitation will be designed to meet desired end land uses, end land uses will be identified in the Closure Plan, in consultation with agencies, stakeholders and Indigenous communities, as the Project progresses.

The area where residual effects will occur has been disturbed by previous mining and forestry activities; however, there will be access restrictions to the PDA. Navigation between Kenogamisis Lake and Goldfield Lake will be maintained and land and resource use are expected to continue at current levels in the regional area where there is an abundance of trails, and wildlife resources for hunting, trapping, fishing, guide outfitting and bait harvesting.

Residual adverse environmental effects on land and resource use were determined to be not significant.

20.4.1.3.11 Heritage Resources

The potential environmental effects of the Project on heritage resources include change in archaeological resources and change to architectural and/or historical resources. As part of the assessment, mitigation measures were identified that will be applied to the Project to avoid or reduce effects.

Effects on archaeological resources will be avoided since archaeological assessment programs were conducted in areas of archaeological potential prior to ground disturbance activities. In the event of the unexpected discovery of additional archaeological resources, mitigation measures include: ceasing construction or operation within a 20 m radius and contacting relevant authorities prior to the implementation of procedures and mitigation if an archaeological resource is discovered; a licensed archaeologist will be retained by GGM and further archaeological assessment will be conducted with the involvement of Indigenous communities; key construction and operation staff will be trained in the recognition of basic archaeological artifacts such as Aboriginal material culture (e.g., clay ceramics, lithic artifacts, and faunal remains), and Euro-Canadian material culture (e.g. refined ceramics, glassware, construction debris, and personal effects) in case a potential archaeological resource is found during Project construction and operation; and GGM will work collaboratively with Indigenous communities to develop a protocol for communications should previously undocumented archaeological resources be discovered. If human remains are encountered, GGM will stop work immediately and contact the police or coroner, Registrar or Deputy Registrar of the Cemeteries Regulation section of the Ontario Ministry of Government and Consumer Services, as well as the Archaeology Program Unit of the Ministry of Tourism, Culture and Sport.

The mitigation strategies to be used for architectural/historical resources include implementing a 60 m buffer zone to isolate Cultural Heritage Resource 1 from Project activities; commemorating past occupation and past mining activity, involving the associated architectural and/or historical resources, by creating a record of activities/resources; and detailed documentation (i.e. creating a public record of the structure or structures, which provides researchers and the general public with a land use history, construction details and photographic record of the resource) and salvage (i.e. recovering architectural or historical resources) where retention or relocation are not feasible.

No residual effects on archaeological resources and Euro-Canadian architectural and/or historical resources for all phases of the Project are anticipated. Consequently, no residual adverse effects carried forward for the determination of significance.

20.4.1.3.12 Traditional Land and Resource Use ("TLRU")

The potential environmental effects of the Project on TLRU include change to availability of plant species and access to plant harvesting sites and activities; change to availability of fish species and access to fishing areas and activities; change to availability of hunted and trapped species and access to hunting and trapping areas and activities; and change to cultural or spiritual practices, sites or areas. As part of the assessment, mitigation measures were identified that will be applied to the Project to avoid or reduce effects.

Potential environmental effects on TLRU were determined based on the Project specific TK studies, Project engagement activities, past project experience and literature review. Other valued component assessments provided additional relevant information regarding effects on resources, and aspects of the biophysical and socio-economic environment that may affect TLRU.

To mitigate potential adverse effects, the mitigation measures identified under groundwater, surface water, wildlife and wildlife habitat, land and resource use, fish and fish habitat, and vegetation communities will be applied to avoid or limit effects on components of the environment related to TLRU. Where there is interest, GGM will provide opportunities to local communities for harvesting of plants for traditional purposes prior to construction. GGM will work with Indigenous community representatives in detailed recording and mapping of spiritual or cultural sites, a decision is then made about the relative importance of the site and, if warranted, how to maintain and control access. Through Project design the length and location of roads have been considered in order to reduce potential access restrictions. A Pipe Ceremony will be held prior to commencement of construction under the direction of local Indigenous communities.

It is predicted that residual effects on TLRU are limited to reduced access to the PDA for the pursuit of traditional activities. However, with the historical impacts through much of the PDA, the reduced access is not anticipated to be an issue and while access to the PDA will be limited for the lifetime of the Project, TLRU sites and areas within the local assessment area will continue to be accessible.

Based on the findings of the biophysical and socio-economic assessments related to TLRU (i.e., vegetation communities, fish and fish habitat, wildlife and wildlife habitat, heritage resources, land and resource use, and human and ecological health) and the characterization of effects to known and assumed TLRU sites and areas, it is predicted that the ability of Indigenous communities to maintain current use of lands and resources for traditional purposes outside of the PDA will be retained.

Residual adverse environmental effects on TLRU are determined to be not significant.

20.4.1.3.13 Human and Ecological Health

The potential environmental effects of the Project on human and ecological health include change in human health and change in ecological health. Project emissions include releases into the terrestrial, aquatic and atmospheric environment. As part of the assessment, mitigation measures were identified that will be applied to the Project to avoid or reduce effects.

Several mitigation measures incorporated in the Project to eliminate or reduce environmental effects of the Project will also serve to address human and ecological health effects. These mitigation measures include, but are not limited to, the use of dust suppressants, dust collectors and protective covers, implementation of a Water Management and Monitoring Plan, industrial health and hygiene programs and progressive rehabilitation that address pathways related to water.

The human health and ecological risk assessments identified negligible risks from exposure (i.e., inhalation and ingestion) of Project-related emissions. With the implementation of the planned mitigation measures for air and surface water, the potential increase in health risk as a result of the Project is negligible. As such, adverse health effects are not expected and, correspondingly, a change to human or ecological health is not expected.

20.4.1.4 Cumulative Effects Assessment

Based on the characterization of the residual cumulative effects (i.e., after mitigation has been applied) of the Project, in combination with the effects associated with other future projects in the regional assessment area, no significant residual adverse cumulative effects are predicted as a result of the Project.

20.4.2 Permits or Approvals to Obtain

A range of other permits and approvals are required for mining activities and operations through numerous federal, provincial and municipal authorities. Many of these approvals are in the process of being advanced in consultation with regulatory agencies. A list of federal, provincial and municipal permits, licences and/or authorizations required for the Project is provided below in Table 20.1.

Table 20.1: Potential Permits / Approvals

Permits / Approvals	Associated Activities
Federal Permits / Approval	
Authorization for Works Affecting Fish Habitat Legislation: *Fisheries Act* Responsible Agency: Department of Fisheries and Oceans ("DFO") (with some provisions administered by Environment and Climate Change Canada)	• Work that may result in serious harm to fish that are part of a commercial, recreational or Aboriginal fishery, or to fish that support such a fishery.
MDMER Schedule 2 Listing Legislation: *Fisheries Act* Responsible Agency: Environment and Climate Change Canada	• Use of fish bearing waters to deposit mine waste. Environmental effects monitoring program.
License for an Explosives Factory Legislation: *Explosives Act* Responsible Agency: Natural Resources Canada	• Manufacturing, use/storage of blasting explosives.
Transportation of Dangerous Goods Legislation: *Transportation of Dangerous Goods Act* Responsible Agency: Transport Canada	• Transportation of hazardous materials.
Provincial Permits / Approvals	
Mine Closure Plan Legislation: *Mining Act* Responsible Agency: MENDM	• Closure Plan for the Project.
Permit to Take Water Legislation: *Ontario Water Resources Act*, Ontario Regulation 387/04 Responsible Agency: MECP	• Surface water and groundwater taking and dewatering activities.
Environmental Compliance Approval – Air/Noise Legislation: *Environmental Protection Act*, Ontario Regulation 419/05, and Guideline A-7 Responsible Agency: MECP	• Air and noise emissions from Project components and activities.
Environmental Compliance Approval – Industrial Sewage Works Legislation: *Ontario Water Resources Act* Responsible Agency: MECP	• Mine process water; • Sewage treatment plants and discharge; • Discharge of construction dewatering.
Environmental Compliance Approval – Waste Disposal Site Legislation: *Environmental Protection Act*, Ontario Regulation 232/98 Responsible Agency: MECP	• Disposal of construction and/or operation waste materials at an on-site location.
Ozone Depleting Substance Registration Legislation: *Environmental Protection Act*, Ontario Regulation 463/10	• Discharge of a class 1 ozone depleting substance or any thing that contains a Class 1 ozone depleting substance.

Permits / Approvals	Associated Activities
Responsible agency: MECP	
Waste Generator Registration Legislation: *Environmental Protection Act*, Ontario Regulation 347 Responsible Agency: MECP	• On-site storage of materials such as oils, greases (or any other types of waste defined as hazardous or liquid industrial under Ontario Regulation 347).
Work Permit Legislation: *Public Lands Act* Responsible Agency: MNRF	• Permits for any activities or tenure on Crown land, if required.
Aggregate Licence Legislation: *Aggregate Resources Act* Responsible Agency: MNRF	• Extraction of aggregate for construction activities.
Permits and Licences (various) Legislation: *Fish and Wildlife Conservation Act* Responsible Agency: MNRF	• Pre-development fish/wildlife studies; • Initial fish and wildlife relocation; • Destruction of beaver dams, furbearer/bear dens, and nests/eggs of birds wild by nature.
Various Approvals Legislation: *Lakes and Rivers Improvement Act* Responsible Agency: MNRF	• Location Approval, and Plans and Specifications Approval for the polishing pond dam and tailings dams. • Approvals for diversions, channelizations and culverts.
Registration of Notice of Activity and/or Overall Benefit Permit Legislation: *Endangered Species Act, Ontario Regulation 242/08* Responsible Agency: MNRF	• Activities with potential to contravene Sections 9 (Species Protection) or 10 (Habitat Protection) of the *ESA*.
License to Harvest Forest Resources and/or Release of Reservation Legislation: *Crown Forest Sustainability Act* Responsible Agency: MNRF	• Release of Reservation required for Crown timber on private or patented land. • Forestry Resource Licence for Crown timber on Crown land.
Encroachment Permits Legislation: *Public Transportation and Highway Improvement Act* Responsible Agency: Ontario Ministry of Transportation (“MTO”)	• Any work upon, over or under provincial highway right-of-way (except entrances).
Entrance Permits Legislation: *Public Transportation and Highway Improvement Act* Responsible Agency: MTO	• Change in use of an existing entrance, construction of a new entrance or temporary entrance (for construction).
Sign Permits Legislation: *Public Transportation and Highway Improvement Act* Responsible Agency: MTO	• New signs for highway right-of-way.
Building and Land Use Permits Legislation: *Public Transportation and*	• Construction of buildings or facilities close to or adjacent to a provincial highway.

Permits / Approvals	Associated Activities
Highway Improvement Act Responsible Agency: MTO	
Order-in-Council - Legal Highway Transfer Process Legislation: *Public Transportation and Highway Improvement Act* Responsible Agency: MTO	• Transfer of ownership of new highway by-pass to the province and transfer of the existing section to private from province.
Letter of Compliance for Archaeology Legislation*: Ontario Heritage Act* Responsible Agency: MTCS	• Disturbance of any potential archaeological sites.
Official Plan Amendment Legislation: *Planning Act* Responsible Agency: MMAH	• Change to existing land use designation(s) in the Municipality of Greenstone and within the Thunder Bay North District Unorganized Territory.
Municipal Permits / Approvals	
Official Plan and Zoning By-Law Amendment Legislation: *Planning Act* Responsible Agency: Municipality of Greenstone	• Change to existing zoning provision(s).
Building Permit Legislation: *Building Code Act* and Building By-law 01-58 Responsible Agency: Municipality of Greenstone	• Construction of buildings.
Demolition Permit Legislation: *Building Code Act* and Building By-law 01-58 Responsible Agency: Municipality of Greenstone	• Demolition of buildings.

20.5 Consultation Activities

GGM has undertaken active participation through consultation during the planning and preparation of the draft and final EIS/EA which is continuing through the permitting phase of the Project. GGM's consultation program reflects the requirements of the consultation guidelines set out in the Code of Practice for Consultation in Ontario's Environmental Assessment Process (MOECC, 2014). In addition, the consultation program was designed to follow the federal EIS Guidelines and approved provincial ToR for the Project.

During the preparation of the draft and final EIS/EA, GGM consulted with a wide range of stakeholders, Indigenous communities and government agencies through various stages of the Project approval process and is currently completing consultation on required permit applications and EMMPs.

20.5.1 Indigenous Engagement

Through the federal EIS Guidelines and subsequent correspondence with the CEA Agency, GGM was provided direction to consult and engage with: Aroland First Nation (AFN), Ginoogaming First Nation (GFN), Long Lake #58 First Nation (LLFN), the Métis Nation of Ontario (MNO) and Animbiigoo Zaagi'igan Anishinaabek (AZA) as part of the EA.

Provincially, the MECP identified that three communities hold or claim Aboriginal or treaty rights that may be adversely impacted by the Project (AFN, GFN and LLFN), and that it was delegating aspects of consultation to GGM. MECP also indicated that in addition to GGM’s consultation obligations and delegation of procedural aspects with the Indigenous communities identified above, MECP also requires engagement with people or groups who may have an interest in the Project. These communities included:

- AZA;
- Biigtigong Nishnaabeg;
- Biinjitiwaabik Zaaging Anishinaabek (BZA);
- Bingwi Neyaashi Anishinaabek (BNA);
- Constance Lake First Nation;
- Eabametoong First Nation;
- Greenstone Métis Council;
- Marten Falls First Nation;
- Pays Plat First Nation;
- Red Sky Métis First Nation.

Indigenous Environmental Review Teams were formed during the EA process and numerous meetings took place with review teams as well as individual community meetings. Comments from communities were received during the EA process on environmental baseline, alternative methods, comparative analysis results and effects/mitigation and were incorporated into the final EIS/EA. Concerns/issues identified by each community will continue to be discussed and addressed as the Project progresses. GGM is working with Indigenous communities, and their technical review consultants, to address comments on various permit applications and EMMPs.

20.5.2 Summary of Influence of Consultation and Engagement on the Project

Since the initiation of the EA process, consultation has been carried out related to key aspects of the Project, including baseline studies, the identification and evaluation of alternatives, assessment of environmental effects and the overall design of the Project. GGM has considered the interests and questions of stakeholders, government agencies and Indigenous communities that were identified through consultation and incorporated this information as appropriate into the EA documentation to better reflect existing conditions, regulatory mandates, the selection of Project alternatives, the assessment of environmental effects, the identification of mitigation measures and other components of the assessment. A summary of the key changes/refinements made to the final EIS/EA based on the results of consultation are provided in Table 20.2. GGM will continue its ongoing engagement with interested parties throughout the permitting process and into construction, operation and closure of the Project.

Table 20.2: Influence of Consultation on the EIS/EA

EIS/EA Item	Influence of Consultation
Baseline Studies	• The scope of baseline studies included key data sources and comments identified for consideration in the EIS/EA. This information was used throughout the EA process to inform the evaluation of alternatives and the assessment of Project effects. • 2015 baseline results were incorporated into the EIS/EA. Fieldwork continues to be completed as the Project progresses and will be used to inform permitting/long-term monitoring phases. • Baseline studies and the analysis in the EIS/EA were refined to include additional information or clarification to respond to questions or concerns. • TK and TLRU information provided by communities was incorporated into relevant Baseline Reports to characterize existing conditions linked to Indigenous use of the area.
Identification of Alternatives	• The "long list" of alternatives for the initial screening was expanded to account for additional alternatives identified for consideration. • The results of the initial screening were refined to include consideration of new alternatives in the comparative analysis. • The rationale for the screening of alternatives was refined to include additional environmental rationale.
Alternatives Assessment Methodology and Results	• The list of criteria and indicators was revised to include additional consideration of key aspects of the environment. • The climate change indicator specifically identified related to greenhouse gases, and source water protection was considered in the rationale for the selection of the groundwater indicators. • The results of the comparative analysis were updated to consider the range of environmental effects identified as key areas of interest. • Further detail was included in the description of the comparative analysis results to address environmental concerns in the decision-making process, and provide additional rationale for the selection of preferred alternatives. • TK and TLRU information provided by communities was incorporated into the assessment of each alternative to identify the potential for effects on Indigenous communities in selecting preferred Project components. Where location-specific information was not provided, the alternatives assessment took a conservative approach that assumed that communities would potentially use areas anticipated to be disturbed by the Project for traditional purposes.
EA Methodology and Results	• Key areas of interest were identified and considered in the detailed assessment of effects for each VC. Further information provided through the completion of supplemental baseline reports was incorporated into the assessment for each VC. • The effects of historical mining activities were characterized through baseline studies and considered as part of the environmental effects assessment and cumulative effects assessment to clearly delineate between existing contributions to water quality and Project effects. • New data sources were identified and considered through the effects assessment process for VCs. • The rationale for the selection of measurable parameters used in the effects assessment was updated for VCs. • As part of the information sharing throughout the consultation process, Project-related information was provided by Indigenous communities in the form of TK and TLRU studies and other forms of information sharing.

EIS/EA Item	Influence of Consultation
	• TK/TLRU information provided by communities was incorporated into the assessment of effects on TLRU, including linkages to the environmental effects assessment for each other valued component as appropriate.
Project Design	• Setbacks from waterbodies - Removed infrastructure, where possible, from the 120 m reserve for surveyed claim to lease areas. Increased setbacks from Southwest Arm Tributary (including the Goldfield Creek diversion) for Project infrastructure including WRSA D. • Refinements to the size and configuration of the TMF – Avoided unnecessary disturbance of watercourses and fish habitat, including avoiding the infilling of Lake A-322. Shifted location of TMF reclaim pond away from the dams and Kenogamisis Lake. • Modifications to the TMF construction and operation sequence – Addressed regulatory closure objectives and requests from Indigenous communities to plan for closure and carry out progressive rehabilitation by developing the TMF in two separate cells. Incorporated seeding of the surface of the TMF to develop a vegetated cover to improve aesthetics, reduce the potential for surface erosion, and reduce the interaction between runoff and the tailings surface. • Addition of long-term geotechnical stability measures for historical tailings - Identification of stability measures (i.e., buttress, berms and subsurface seepage collection) for the historical MacLeod tailings and logistics for handling and transfer of historical tailings to the TMF. • Confirmation of the need and location of the temporary camp - Refinement of the location to the south side of Old Arena Road, farther from an existing residential area. • Removal of temporary STP effluent discharge to Barton Bay - The temporary camp will be connected to the municipal wastewater system. • Siting of water management facilities - Siting of the seepage, contact water, and subsurface seepage collection systems (i.e., ponds and ditching) to collect seepage and runoff from Project components. • Identification of contingency WRSAs - Identified in the event that the foundation conditions of primary WRSAs are deemed not suitable for anticipated capacities. May also be used in the event that waste rock volumes are higher than expected or increase based on refinements to ore processing as mining advances. • Refinements to the WRSA deposition schedule - Changed the waste rock deposition schedule to maintain the front nine holes and clubhouse of the existing public golf course for as long as possible. • Refinements to the WRSAs - Increased setbacks from the Goldfield Creek diversion and Kenogamisis Lake. Redesigned a portion of WRSA A located over the historical MacLeod tailings for use as overburden storage and provided an enhanced cover to increase runoff and reduce infiltration through the historical MacLeod tailings thereby reducing potential effects on water quality from the historical tailings. • Identifications of aggregate sources - Identification of aggregate sources in proximity to Project activities to provide aggregate and till materials for construction and ongoing maintenance activities (i.e., reducing haul distance). • Refinements made to watercourse diversion - Maintained existing flow through the northern portion of Goldfield Lake. Reduced the overall environmental effects on flow regimes, water transfer between subwatersheds, fish and fish habitat. Created a diversion pond (over 7 ha), over 2.5 km of channel and two backwater areas to manage flow and sediment control. Fluvial geomorphology studies were conducted as a result of consultation input, and contributed to the design of the backwater areas.

EIS/EA Item	Influence of Consultation
	• Identification of additional and backup power sources for the Project during both construction and operation - Power for construction activities was confirmed, and will be from a temporary grid connection and/or temporary diesel generators. • Change in location of the temporary construction treated effluent discharge location - The location of the temporary construction treated effluent discharge was moved from the Central Basin to the Southwest Arm of Kenogamisis Lake. • Confirmation of preferred freshwater intake and treated effluent discharge locations - The location for the freshwater intake and treated effluent discharge was confirmed. • The future locations of provincial facilities have been refined - Adjustments to the location of the MTO Patrol Yard were made in consultation with the MTO. Adjustments to the locations of the Hydro One access road and transmission/distribution line avoided a wetland. • Modification of the Highway 11 realignment - Identified a 500 m safety offset buffer surrounding the open pit associated with blasting. • Refinements to the size and configuration of the open pit - Refinements to the open pit were made based on ongoing Project planning and engineering design. • Modification to the number and location of ore stockpiles - Reduced the number of ore stockpiles from two to one and moved the stockpile closer to the process plant. • Refinement to the number of years required to fill the open pit during closure - Refinements were made as a result of ongoing Project planning and engineering design. Maximized the use of contact water to fill the open pit. Groundwater levels recover more quickly (i.e., with reduced timelines for open pit filling) during closure as the open pit fills to form a pit lake.
Mitigation and Monitoring	• Identification of additional mitigation measures or refinement of identified mitigation measures to address local stakeholder, Indigenous communities, and government agencies concerns. • Updates to mitigation measures also resulted in revisiting the residual effects conclusions to confirm if updates were appropriate. • EMMPs were identified and were developed to a conceptual level of detail in the final EIS/EA.

20.6 Follow-up Environmental Monitoring and Management Plans

As part of the EA process, a monitoring framework was advanced for all subsequent phases of the Project and conceptual EMMPs were developed. The framework includes monitoring related to both compliance monitoring and effects monitoring during construction, operation, and closure and to fulfill anticipated compliance monitoring requirements. EMMPs will outline the proposed environmental protection measures and commitments to be carried out by GGM and their contractor and subcontractors, during construction and operation, respectively to avoid or reduce potential effects. These EMMPs will outline adaptive management and contingency measures to respond to exceedances of regulatory standards related to environmental discharges or other adverse effects of the Project. Contingency measures specific to each EMMP will be implemented in the event that regular environmental and compliance monitoring programs detect deviations from standard operating conditions that result in, or may lead to, adverse effects on worker safety or the environment.

GGM is currently advancing the development of the EMMPs and expects the following list of plans to be required:

- Air Quality Management and Monitoring Plan;
- Aquatic Management and Monitoring Plan;
- Archaeological and Heritage Resources Management Plan;
- Biodiversity Management and Monitoring Plan;
- Construction Environmental Management Plan;
- Communications Plan;
- Current Use of Land for Traditional Peoples Plan;
- Emergency Response Plan;
- Spill Response and Contingency Plan;
- Erosion and Sediment Control Plan;
- Explosive and Blasting Management Plan;
- Health and Socio-Economic Conditions of Indigenous Peoples Monitoring Plan;
- Historical Tailings Management and Relocation Plan;
- Noise and Vibration Management and Monitoring Plan;
- Sanitary Sewage Management & Contingency Plan;
- Soil Management Plan;
- Waste Management Plan;
- Waste Rock Management Plan;
- Water Management and Monitoring Plan.

Upon completion of permitting, refinements to the follow-up and monitoring programs will incorporate outcomes of the approval processes, and refinements will be considered throughout the Project. Program plans are iterative by nature and the monitoring activities associated with the Project will be used to inform adaptive management, which is a process for continuously improving environmental management practices.

20.7 Closure, Decommissioning and Reclamation

Before mining operations can begin, MENDM requires that a Closure Plan with Financial Assurance be submitted and approved under the *Mining Act R.S.O. 1990*, Chapter M.14 (amended by S.O. 2010, 18. 23); Part VII under the Act, O. Reg. 240/00 as amended, and Schedule 1 and 2, Mine Rehabilitation Code of Ontario.

A Closure Plan has been developed and includes details on closure including progressive rehabilitation, rehabilitation measures, monitoring and expected site conditions following closure. At the end of mining operations, the main features requiring closure will include the main open pit, water management and drainage systems, WRSAs, TMF, site access roads and buildings and associated infrastructure. After the closure activities have been carried out, a post-closure monitoring program will be carried out to verify that the closure objectives and criteria have been met and confirm that the Project can proceed to final close out under the *Mining Act*.

The main elements of the Closure Plan include progressive rehabilitation during Project operation for certain components, and final closure measures following the end of mine operations. When practical, areas that are no longer required may be rehabilitated during mining operations. These activities, known as progressive rehabilitation, contribute to the overall rehabilitation efforts that would otherwise be carried out at closure, or efforts carried out in support of the closure activities (e.g. field trials). Once the mine advances from the development stage to the operational stage, progressive rehabilitation activities can commence, as applicable. Progressive rehabilitation opportunities may include:

- Removal of construction-related buildings, laydown areas, and access roads;
- Stabilization and re-vegetation of WRSAs, where practical;
- Rehabilitation of the north cell of the TMF, upon completion of deposition anticipated after Year 11, consisting of a vegetated soil cover with runoff from the cell directed to the south cell, or to the environment once water quality meets acceptable regulatory requirements;
- Rehabilitation of aggregate sources once they are no longer required;
- Backfilling the eastern extension of the open pit with waste rock;
- Removal of hazardous and non-hazardous waste materials from the site on a regular basis, where possible.

While progressive rehabilitation activities will be carried out throughout the mine life, the majority of rehabilitation work will take place once mining has been completed. The following list summarizes the main activities associated with closure:

- Infrastructure, equipment and mining materials (including buildings, pipelines, site lighting and security, service water supply, water management facilities and petroleum products) will be removed.

- Some facilities (e.g. access roads and the effluent treatment plant) may be required for the proper care and maintenance of the site during closure and will be removed/rehabilitated once they are no longer required during closure.
- The open pit will be filled with water, creating a pit lake.
- The WRSAs and TMF will be stabilized (chemically and physically) and the top surfaces and benches covered with a soil cover to facilitate vegetation growth. The TMF will be fully revegetated.
- In preparation for revegetation efforts, the ground surface will be prepared by scarification or ripping of compact surfaces, amending soil to support vegetative growth, and implementing erosion protection measures to protect the soil cover until vegetation is established.
- The majority of the closure measures will be implemented over a five-year period after the cessation of mining and ore processing activities; however, rehabilitation of the open pit will take significantly longer due to the time required to fill the open pit with water. To expedite the filling of the open pit, water will be pumped into the open pit from the TMF, contact water collection ponds and from the Southwest Arm of Kenogamisis Lake. The site can be considered to be in a state of post-closure when the site has been shown to be stable and able to meet the closure criteria.

The overall objective of closure is to return the site to a chemically and physically stable state which is self-sustaining and supports the desired future land uses. The landscape will be revegetated using locally available, non-invasive plant species to encourage the return of wildlife and fish species to the area. Most access restrictions will be lifted after closure; however, a boulder fence will be erected around the open pit to restrict access for safety purposes. It is anticipated that recreational activities such as hunting, hiking, snowmobiling and other passive activities, as well as economic uses such as forestry, would be permitted.

Monitoring will be completed to assess the physical, chemical and biological stability of the Project and confirm that closure objectives have been met and when the Project has reached a condition suitable for moving to closed out status as defined under the *Mining Act*. The site will be monitored by GGM according to a set schedule to verify the site is performing as expected and that effluent criteria are being met. This includes the monitoring of effluent water quality (surface and groundwater), aquatic and terrestrial environments, and physical stability (embankments and the open pit slopes).

The Closure Plan and all technical details, including the cost estimate for Financial Assurance have been approved by MENDM in January 2020. As the Project progresses, it is anticipated that subsequent Closure Plan Amendments will be required to capture any proposed changes to the Project and rehabilitation methods.

21. CAPITAL AND OPERATING COSTS

21.1 Capital Expenditures

21.1.1 Responsibility Matrix

Greenstone Gold Mines Limited ("GGM") is responsible for the overall coordination, compilation, documentation and quality control of the initial capital cost ("CAPEX") estimate. Estimating responsibilities were assigned at the Work Breakdown Schedule ("WBS") level to various contributors. Responsibility for providing inputs were as follows:

- Ausenco Engineering Canada Inc. ("Ausenco") – quantities for the process plant and supporting infrastructure (except as otherwise noted); firm and budgetary Request for Proposal ("RFP") processes for major equipment and certain supply/install contracts and equipment, material and labor costs for piping, HVAC, underground utilities and fire protection;
- Wood Canada Limited ("Wood") – quantities for the TMF and Goldfield Creek diversion, and the estimate for the fish habitat compensation;
- TBT Engineering Limited ("TBT") – quantities and firm price RFP processes for the re-alignment of Highway 11 and the relocation of the MTO Patrol Station;
- SLR Consulting ("SLR") – scope and quantities for mine waste stockpiles water management ponds and ditches and Macleod high tailings seepage collection subdrain;
- Enbridge Gas Inc. ("Enbridge") – quantities and cost estimates for the natural gas pipeline from the TransCanada Pipeline Mainline to the power plant and site natural gas distribution header;
- FNX Innov ("FNX") – quantities and cost estimates for the Hydro One Geraldton Transmission Station and 115 kV power lines;
- G Mining Services Inc. ("GMS") – quantities and cost estimates for the mine mobile equipment and mining cost and overall review of the CAPEX;
- Greenstone Gold Mines ("GGM") – quantities for earthworks other than noted above, direct labor rates, labor hours for concrete, structural and mechanical installations, site communications and IT costs, quantities and cost estimates for the construction indirects, infrastructure relocation budgets (except as otherwise noted), and Owner's general and administration cost. GGM also maintains the overall estimate database and libraries.

21.1.2 Basis of Estimate

The base date of the CAPEX estimate is Q3 2019.

The CAPEX period is assumed to start on January 1, Year -3. The pre-production CAPEX period is planned over 39 months (start of CAPEX period to commercial production), which includes 8 months for construction readiness and pre-production activities, and an additional 27 months for major construction.

The CAPEX estimate is aligned with an owner-managed project delivery model.

21.1.3 CAPEX Summary

The CAPEX estimate is summarized in Table 21.1. WBS Areas 100 to 600 include the Project's direct costs, while WBS Areas 700 to 900 cover indirect costs, Owner's cost and mine-preproduction. The CAPEX for Project construction, equipment purchases and pre-production activities is estimated to be CAD 1,301M, as shown in Table 1.4. The CAPEX includes a contingency of CAD 108M, which is 9.0% of the total before contingency. The total hours for the CAPEX phase are 3.9M, including Project construction, Owner's cost and mine pre-production hours.

Table 21.1: Capital Expenditures Summary

Work Breakdown Structure	Total (CAD M)
100 - Infrastructure	79.4
200 - Power & Electrical	70.4
300 - Water & Tailings Management	94.2
400 - Mobile Equipment	155.1
500 - Infrastructure Repositioning	61.3
600 - Process Plant General	313.8
700 - Construction Indirect Cost	242.1
800 - General Services - Owner's Cost	44.8
900 - 980 - Preproduction, Start-up, Commissioning	131.2
990 - Contingency	108.4
Grand Total	**1,300.7**

21.1.3.1 Direct Costs

The engineering consultants who completed the detailed engineering (or advanced basic engineering in certain cases) were responsible for calculating the material take off estimates for their respective areas. Appropriate waste allowances were included. Firm price RFP processes were completed for the majority of process plant equipment, the power plant equipment, the temporary and permanent effluent treatment plants, the sewage treatment plant and the mine mobile equipment fleet. Where vendor data was required for detailed engineering, a Letter of Intent has been issued. The remaining equipment and material costs were based on budgetary bid processes, quotes, consultant's historical data and in-house databases, or benchmarked from a previous project.

Labour unit rates were developed from first principles based on budgetary quotations from fourteen general contractors located in Northern Ontario or Western Quebec, consistent with the type and size of contractors that will form the bidders list for these installation activities. The direct installation hours were based on a combination of firm price proposals, budgetary quotes and updated feasibility study estimates, benchmarked against previous projects and reviewed by experienced construction personnel. Firm price RFP processes were completed for the Tailings Management Facility ("TMF"), Goldfield Creek and Highway 11 relocation. A productivity factor adjustment to account for items that are not incorporated in the base installation hours was developed and applied to the relevant trades.

Estimates related to certain offsite infrastructure or infrastructure relocations were either provided by the relevant agency or developed by specialized engineering consultants. This includes the Enbridge natural gas pipelines, the Hydro One Substation relocation and the connection to the Municipal network for potable water.

The CAPEX estimate for WBS 100 - Infrastructure is summarized in Table 21.2. The CAPEX for the temporary camp WBS Area 140 represents the cost of site preparation only, as the rental for the camp buildings is captured in WBS Area 740. Support facilities include the site administration building, the explosives reagents storage, and the temporary explosives storage. The predominant cost in WBS Area 170 is the natural gas tap and pipeline to the power plant, and also includes the diesel fuel storage and distribution systems for the mine fleet refueling. WBS Area 180 includes the recycling and waste management facilities.

Table 21.2: Infrastructure Capital Expenditures

Work Breakdown Structure	Total (CAD M)
110 - General Site Preparation	16.2
120 - Workshops/Storage	28.9
130 - Support Facilities	10.1
140 – Camp	2.9
170 - Fuel Systems	21.1
180 - Other Facilities	0.2
Grand Total	**79.4**

The CAPEX estimate for WBS Area 200 - Power Supply and Communications is summarized in Table 21.3. The power plant WBS Area 210 includes all material, equipment and construction costs related to the power plant. The detailed engineering and commissioning costs for the power plant are in WBS Areas 710 and 950 respectively. The IT estimate reflects the infrastructure and systems required for the Project construction phase. Systems required for operations including G&A (finance, accounting, purchasing, and inventory), mining operations (slope stability monitoring fleet and maintenance management) and the geology and mining engineering departments are included in WBS 800 and 900.

Table 21.3: Power Supply and Communications Capital Expenditures

Work Breakdown Structure	Total (CAD M)
210 - High Voltage	62.6
240 - Site Power Distribution	4.9
260 - IT and Site Communications	3.0
Grand Total	**70.4**

The CAPEX estimate for WBS 300 - Water and Tailings Management is presented in Potable water includes the connections to municipal systems from the Project site. Reclaim water consists primarily of the pump station at the TMF and reclaim return water pipeline to the process plant. Effluent and surface water management consists primarily of the mine water effluent treatment plant, collection ditches and ponds. Fire water in the WBS 370 represents only the pump station, fire piping site distribution is within WBS 100 with underground services and sprinklers and other related device are costed within each building. The plant site sewage system in WBS 380 is a modular unit and the site sewage piping network is costed in WBS 100 with underground services.

Table 21.4. The TMF estimate includes the scope that will be completed during the pre-production phase. Detailed designed is substantially complete and was used as the basis for the RFP process and the estimate for this area is based on the bids received from contractors. Mine pre-production will be responsible for the transport and placement of mine rock fill for the TMF main dams and TMF access road.

Potable water includes the connections to municipal systems from the Project site. Reclaim water consists primarily of the pump station at the TMF and reclaim return water pipeline to the process plant. Effluent and surface water management consists primarily of the mine water effluent treatment plant, collection ditches and ponds. Fire water in the WBS 370 represents only the pump station, fire piping site distribution is within WBS 100 with underground services and sprinklers and other related device are costed within each building. The plant site sewage system in WBS 380 is a modular unit and the site sewage piping network is costed in WBS 100 with underground services.

Table 21.4: Water and Tailings Management Capital Expenditures

Work Breakdown Structure	Total (CAD M)
310 - Potable Water	1.5
320 - Reclaim Water	3.6
340 - Tailings Management Facility (TMF)	64.4
350 - Surface Water Management	11.9
360 - Effluent Water Management	11.6
370 - Fire Water	0.5
380 - Domestic Sewage	0.7
Grand Total	**94.2**

The CAPEX estimate for WBS 400 - Mobile Equipment is summarized in Table 21.5. A firm price RFP process was completed for the mine mobile equipment fleet. The equipment pricing includes tires, fire suppression, transport to the Project site, assembly and commissioning.

Table 21.5: Mobile Equipment Capital Expenditures

Work Breakdown Structure	Total (CAD M)
410 - Mine Equipment	153.8
430 - Plant and Surface Mobile Equipment	1.3
Grand Total	**155.1**

The CAPEX estimate for WBS Area 500 - Infrastructure repositioning totals CAD 61.3M. Detailed designed for the Highway 11 and MTO Patrol Yard are substantially complete. RFP processes were completed for these scopes and used as the basis of the estimate update. The CAPEX estimate for the relocation of the existing substation was based on advanced basic engineering and developed by a Hydro One approved engineering consultant.

The CAPEX estimate for the process plant is summarized in Table 21.6. The estimate includes all direct costs for the processing facilities. The processing facilities are further described in Section 17.

Table 21.6: Process Plant Capital Costs

Work Breakdown Structure	Total (CAD M)
610 - Crushing and Ore Handling	61.0
620 - Grinding & Gravity	103.4
630 - Pre-Leach / Leach / CIP	54.3
640 - CN Detox & Final Tails	11.3
650 - Acid Wash, Elution, Carbon Regeneration	6.7
660 - Refinery	2.2
670 - Electrical Process Plant	8.4
680 - Plant Reagent & Services	9.0
690 - Mill Building	57.5
Grand Total	**313.8**

21.1.4 Construction Indirect Costs and Owner's Costs

Indirect costs have been developed primarily from detailed estimates:

- Construction indirect costs were developed by GGM based on the execution strategy and budgetary quotes where applicable, and include all temporary site facilities, construction services, contractor management and indirects, and temporary camp costs;
- Freight and logistics were estimated based on budgetary quotations;
- General & Administration Owner's costs were developed by GGM;
- Pre-production mining costs were developed by GMS and are consistent with the basis of the OPEX costs;
- Commissioning costs were based on a ramp-up schedule and are consistent with the basis of the OPEX costs.

The CAPEX estimate for indirect costs is summarized in Table 21.7.

Table 21.7: Indirect Costs

Work Breakdown Structure	Total (CAD M)
700 - Construction Indirect Costs	**242.1**
710 - Engineering, CM, PM	77.5
720 - Construction Facilities & Services	44.6
730 - Contractor Mobilization/Demobilization and Indirects	56.3
740 - Construction Camp Facilities & Operation	40.0
750 - Freight & Logistics	23.7
800 - General Services - Owner's Cost	**44.8**
810 - Departments	39.2
820 – Logistics / Taxes / Insurance	4.5
830 - Operations Accommodations	1.1
900 - Preproduction, Startup, Commissioning	**239.6**
910 - Mining Preproduction / Commissioning	93.8
940 - Spares & First Fills	10.6
950 - Process Plant Preproduction / Commissioning	24.4
960 - Operational Readiness Support	2.5
990 - Contingency	108.4
Grand Total	**526.5**

21.1.4.1 Allowances, Contingency and Escalation

The contingency amount was established through a facilitated quantitative risk assessment ("QRA") process, using a Monte Carlo simulation at a P80 confidence level. The total contingency provision is CAD 108M, which represents 9% of the total CAPEX, and is comprised of the following:

- A design/quantity growth allowance for all equipment, material and labour (except WBS 400 Mine Mobile Equipment) for CAD 10.7M;
- A contingency provision of CAD 81.4M on project costs (WBS 100 - 300 and 500 - 700), representing 9.5% of those costs;

- A contingency provision of CAD 3.9M on mine mobile equipment (WBS 100 400), representing 2.5% of those costs;
- A contingency provision of CAD 12.4M on Owner's Cost and Mine Pre-production (WBS 800 and 900), representing 7% of those costs.

There is no provision for escalation in the CAPEX.

21.1.5 Sustaining Capital

Sustaining capital is presented in Table 21.8

Sustaining capital for the mine includes additional equipment purchases for a total of CAD 156.8M. Major equipment repairs are capitalized which represent CAD 165.8M over the LOM. The sustaining capital estimate also includes the remaining mining construction civil works for ditches, ponds and dump shear keys totalling CAD 2.8M.

Major dam raises for the TMF are accounted for in sustaining capital, a total of CAD 85.2M over two major work periods: Year 2-3 and Year 7-9 of mine operations. This work would be done by the mine operations team.

Table 21.8: Sustaining Capital Costs

Sustaining Capital Costs	Total (CAD M)
Mine Equipment Capital Repairs	165.8
Mine Equipment Purchases	156.8
TMF Dam Construction	85.2
Mine Civil Works	2.8
Geraldton Heritage Interpretive Center	4.5
Headframe Relocation	1.4
Golf Course Expansion	3.1
Exploration	-
Total	**419.7**

21.2 Operating Costs

21.2.1 Operating Costs Summary

Operating costs are summarized in Table 21.9. The operating costs include mining, processing, G&A, transportation and refining, royalties and other costs. The average operating cost is CAD 708/oz of gold or CAD 26.51./t milled over the LOM.

Table 21.9: Operating Costs Summary

Category	Total Costs (CAD M)	Unit Cost (CAD/t milled)	Cost per oz (CAD/oz)
Mining	1,890	14.13	378
Processing	968	7.23	193
G&A	401	3.00	80
Transp. & Refining	15	0.11	3
Royalties	273	2.04	55
Total Operating Costs	**3,547**	**26.51**	**708**

The operating organization consists of three departments: mine, including mine operations, geology, engineering and maintenance; process and power plant; and G&A including human resources, environment, health and safety, site services and accounting. The peak total operating workforce is 521 employees (reached in Year 6).

Table 21.10: Peak Operations Workforce

Operations Department	Peak Workforce
Mine	371
Process Plant	103
G&A	47
Number of Employees	**521**

A summary of the total operating costs, by year, is presented in Table 21.11 and Figure 21.1.

Figure 21.1: Operating Cost by Year

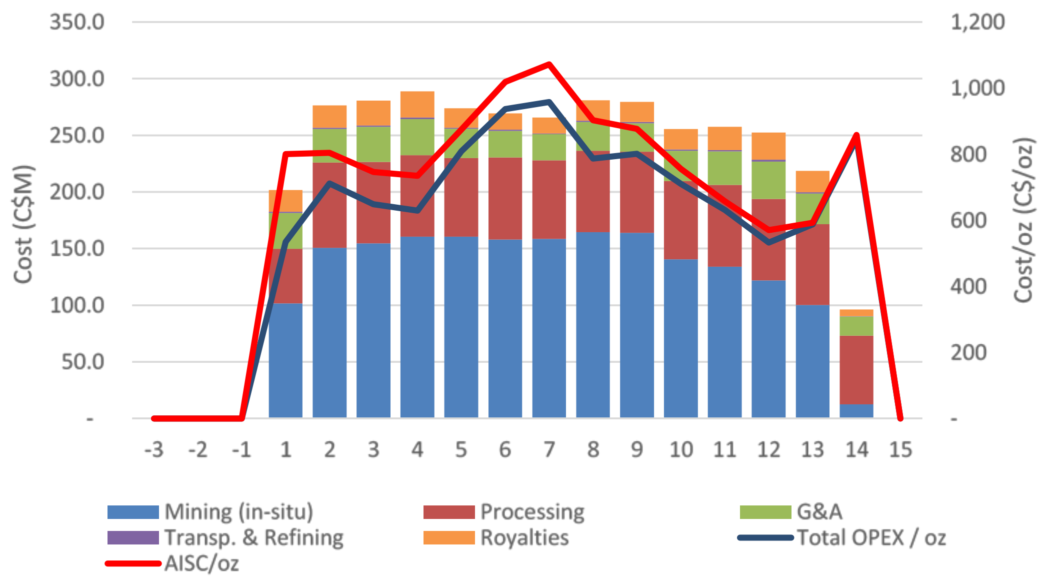


Table 21.11: Total Operating Costs Summary

Commercial Production	**Total**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**
Tonnage Milled (Mt)	**133.79**	-	6.71	9.88	9.86	9.86	9.86	9.88	9.86	9.86	9.86	9.88	9.86	9.86	9.86	8.73
Tonnage Mined (Mt)	**784.72**	-	49.83	69.94	70.10	69.89	70.05	70.06	69.89	70.08	69.92	61.89	48.95	35.01	26.29	2.81
Gold Sales Ops (kozs)	**5,007**	-	379	389	433	460	339	289	278	357	349	360	408	474	373	118
Operating Costs																
Mining (in-situ)	**1,882**	-	101.5	150.7	154.5	160.5	160.7	158.0	158.4	164.6	163.9	140.4	134.0	122.0	100.1	12.3
Mining (rehandling)	**8**	-	0.9	0.6	0.1	0.1	0.5	1.3	0.4	-	-	0.0	-	0.0	0.6	3.8
Processing	**968**	-	48.3	75.2	72.0	71.9	69.3	72.4	69.6	71.7	71.6	69.3	72.3	71.8	71.4	60.7
G&A	**401**	-	33.3	31.2	32.8	34.0	27.2	24.4	23.6	27.1	26.8	28.3	31.1	35.2	28.8	17.0
Transp. & Refining	**15**	-	1.1	1.2	1.3	1.4	1.0	0.9	0.8	1.1	1.0	1.1	1.2	1.4	1.1	0.4
Royalties	**273**	-	20.7	21.2	23.6	25.1	18.5	15.8	15.1	19.5	19.0	19.6	22.3	25.8	20.3	6.4
Total Operating Cost	**3,547**	**-**	205.8	280.1	284.3	292.9	277.2	272.8	268.1	284.0	282.4	258.7	261.0	256.3	222.4	100.7
Closure & Reclamation	**54**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.5
Sustaining Capital	**420**	0.4	100.7	35.9	42.3	48.4	22.3	23.7	31.7	41.4	26.6	16.5	10.6	17.8	1.5	-
AISC	**4,020**	0.4	306.5	316.0	326.6	341.3	299.5	296.5	299.8	325.4	308.9	275.2	271.6	274.1	223.9	102.1
Total OPEX / oz	**708**	-	543	720	657	637	817	943	965	795	809	719	639	541	596	852
Total OPEX / t milled	**26.51**	-	30.68	28.34	28.85	29.72	28.13	27.60	27.20	28.81	28.65	26.17	26.48	26.00	22.56	11.53
Mining Cost / t mined	**2.41**	-	2.05	2.16	2.21	2.30	2.30	2.27	2.27	2.35	2.34	2.27	2.74	3.49	3.83	5.72
AISC/oz	**803**	-	809	812	755	742	883	1,025	1,079	911	885	764	665	578	600	865

21.3 Mining Costs

Table 21.12 presents the breakdown of mining costs, by department, while Table 21.13 presents the major cost drivers for the mine department.

The mine operating costs are estimated from first principles for all mine activities. Equipment hours required to meet production needs of the LOM plan are based on productivity factors or equipment simulations. Each piece of equipment has an hourly operating cost which includes operating and maintenance labour, fuel and lube, maintenance parts, tires (if required) and ground engaging tools (if required). A formal RFP process has been completed for the mine equipment and associated operating costs, fuel, tires, explosives and accessories, etc.

The average mining cost during operations is estimated at CAD 2.41/t mined including re-handling costs. The mining costs are lower than average during the early years and increase with increased haulage distances and pit deepening, in the later years. This operating cost estimate excludes capital repairs which treated as sustaining capital.

Haulage is the major mining cost activity representing 41% of total costs followed by blasting (12%), loading (9%) and drilling (7%). Some haulage costs have been back charged to the TMF dam construction as this represents incremental haulage. Loading and haulage for stockpile re-handling is also captured as a separate activity cost.

Fuel is the dominant cost, by element, representing 31% of total costs, followed by salaries (27%), maintenance parts (14%) and bulk explosives (12%).

Table 21.12: Mining Cost Summary Total

Mining Costs (CAD M)	Total	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Mine Operations	**34.8**	2.4	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.7	2.6	0.2
Mine Maintenance Admin.	**97.7**	8.5	11.1	10.7	8.9	6.4	6.4	6.4	6.4	6.4	6.1	6.0	6.0	5.8	2.8
Mine Geology	**14.4**	0.9	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.0	0.1
Mine Engineering	**29.6**	1.7	2.2	2.3	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.4	2.3	1.9	0.2
Grade Control	**41.5**	3.3	3.9	3.5	3.5	3.3	3.1	3.3	3.5	3.5	3.2	2.9	2.4	2.0	0.3
Voids Management	**22.1**	0.7	0.9	0.9	0.9	0.9	0.9	0.9	4.6	5.7	2.1	1.3	1.2	1.2	0.1
Topo Drilling Contract	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Drilling	**139.4**	8.3	12.5	12.2	12.5	12.4	12.4	12.5	12.4	12.5	11.2	8.6	6.5	5.1	0.5
Blasting	**229.1**	13.8	20.5	20.1	20.4	19.8	19.8	20.0	19.9	20.0	17.6	14.6	11.7	9.8	1.1
Pre-Split D&B	**54.8**	4.5	4.6	3.6	4.2	4.0	4.0	4.0	4.9	5.1	5.0	4.5	2.8	2.8	0.6
Loading	**164.1**	11.8	14.4	14.6	14.4	14.5	14.4	14.3	14.5	14.4	13.0	10.4	7.5	5.3	0.6
Hauling	**776.5**	29.0	54.3	59.8	66.8	70.5	68.1	68.2	69.6	67.4	53.3	58.6	59.5	48.7	2.8
Dump Maintenance	**81.6**	5.2	7.0	7.0	7.0	7.0	7.0	7.0	7.0	7.0	7.0	5.8	4.7	2.5	0.4
Road Maintenance	**72.2**	4.3	5.6	5.6	5.6	5.6	5.6	5.6	5.6	5.6	5.6	5.6	5.5	5.4	0.8
Dewatering	**17.7**	0.9	1.2	1.5	1.3	1.4	1.4	1.4	1.4	1.4	1.4	1.5	1.5	1.4	0.2
Overburden Mining Contract	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Support Equipment	**106.4**	6.4	8.8	8.8	8.8	8.8	8.8	8.8	8.8	8.8	8.7	7.9	6.7	4.8	1.7
Sub-Total In-Situ Mining	**1,881.8**	**101.5**	**150.7**	**154.5**	**160.5**	**160.7**	**158.0**	**158.5**	**164.6**	**163.9**	**140.4**	**134.0**	**122.0**	**100.1**	**12.4**
Rehandling	**8.4**	**0.9**	**0.6**	**0.1**	**0.2**	**0.5**	**1.3**	**0.4**	**-**	**-**	**0.0**	**-**	**0.0**	**0.6**	**3.8**
Total Mining	**1,890.2**	**102.4**	**151.3**	**154.6**	**160.7**	**161.2**	**159.4**	**158.9**	**164.6**	**163.9**	**140.5**	**134.0**	**122.0**	**100.7**	**16.1**
Total Mining / t mined	**2.41**	**2.05**	**2.16**	**2.21**	**2.30**	**2.30**	**2.27**	**2.27**	**2.35**	**2.34**	**2.27**	**2.74**	**3.49**	**3.83**	**5.72**

Table 21.13: Top Three Mining Costs by Cost Type

Top Mining Costs (CAD K)	Total	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Diesel/Fuel	**580,191**	26,386	42,624	45,921	49,971	52,227	49,079	49,663	51,474	49,206	41,178	43,239	41,192	33,101	4,930
Salaries (incl Maintenance Labor)	**506,928**	26,764	39,560	40,237	41,305	42,420	42,531	42,069	41,994	42,027	38,884	37,053	35,045	29,992	7,048
Maintenance Parts	**257,029**	13,907	20,318	20,802	21,583	22,329	22,513	22,400	22,190	22,345	19,367	18,081	15,906	12,817	2,471
Sub-Total Top Three	**1,344,149**	**67,056**	**102,502**	**106,959**	**112,859**	**116,977**	**114,123**	**114,133**	**115,658**	**113,578**	**99,429**	**98,373**	**92,143**	**75,910**	**14,449**
% of Total	**71%**	66%	68%	69%	70%	73%	72%	72%	70%	69%	71%	73%	76%	75%	90%

21.3.1 Processing Costs

The process plant operating costs were evaluated based on metallurgical test work, recent supplier quotations, recent salary survey and standard industry practice. The process costs are divided into eight categories: workforce, electrical power, wear parts, maintenance parts, grinding media, reagents including water treatment plant, metallurgical and geochemical laboratories and mill general.

The total process plant LOM average operating costs were estimated at CAD 7.23/t milled at a nameplate capacity of 27,000 t/d following ramp-up The average process plant operating cost is summarized in Table 21.14.

Table 21.14: Process Operating Costs Summary

OPEX Cost Category	Total OPEX (CAD M/y)	% of Total	Unit Cost (CAD/t milled)
Labour (excl. Power Plant)	140.3	14.5%	1.05
Power (including Ancillary Equipment NG)	207.1	21.4%	1.55
Wear Parts	133.2	13.8%	1.00
Maintenance Parts	91.7	9.5%	0.69
Grinding Media	103.3	10.7%	0.77
Reagents (incl. WTP)	272.8	28.2%	2.04
Metallurgy & Geochem Lab	4.1	0.4%	0.03
Mill General	14.9	1.5%	0.11
Total Mill Costs	**967.5**	**100%**	**7.23**

The processing plant electrical power requirements are based on the electrical demands specified in the load list which take into account the installed power, the utilization factor, the mechanical load factor and the process availability. The installed power for all the major equipment and most of the minor equipment have been validated with suppliers during the equipment selection process during the detailed engineering phase.

The wear parts cost category includes all the major equipment replacement parts (crusher liners, ball mill liners, HPGR rolls, etc.) and are generally based on equipment vendor recommendation and / or contractors have provided cost estimates required to execute these replacements. The life cycle estimation and replacement parts costs are based on data provided by the selected manufacturer for each major type of equipment and in some cases a lower estimate has been adopted.

The maintenance parts cost category includes all the minor normal operation replacement parts such as pump casings, screen decks, chute liners, conveyor belts, etc. Year 1 operational spares have been included, where possible, based on selected equipment packages to obtain a higher level of accuracy. For equipment that has not been selected and general maintenance requirements, these costs are calculated to represent 3.5% of the total mechanical equipment costs.

Grinding media consumption is based on the ore abrasion index and is calculated at CAD 0.039 kg/kWh. The ball mill power consumption and grinding media costs are used to evaluate an annual grinding media cost.

Most reagents consumption data is derived from test work. For some low consumption reagents, such as antiscalant and refining flux, the requirements have been estimated based on similar projects. For all reagents (except low consumption) a formal RFP process has been completed and the selected vendor pricing is included in the plant cost model.

Oxygen is produced on site by a vacuum swing absorption (“VSA”) plant with liquid storage in tanks as back-up. The plant is built, owned, and operated by a third party. A fixed monthly fee is associated with this service. Where possible, reagents will arrive in bulk containers with appropriate on-site unloading and storage facilities

Sample preparation for mining and plant samples for gold assaying will be carried out by a third party facility. The geochemical lab will carry out pulp digestions and analysis for all other geochemical requirements except certified water analysis. The metallurgical lab will consist of conducting bottle roll test work and other investigations by the metallurgical department.

A mill general category is included to cover miscellaneous costs such tool purchase, dozer usage, equipment rental, consulting and other costs.

The power cost of site generated power was derived from three major components: 1) forecasted energy price (natural gas); 2) workforce required to operate and maintain the power plant; and 3) maintenance costs over the LOM. The total unit power cost is estimated at CAD 0.049/kWh.

The natural gas price used for power requirements was evaluated at CAD 3.89/GJ. With annualized power demands, this represents CAD 0.033/kWh.

21.3.2 General and Administration Costs

The G&A costs, by year, are summarized in Table 21.15 and peak at CAD 18.3M per year. The labour costs for G&A represent 31% of the total G&A budget. The G&A costs reflect the operating model which assumes a locally sourced workforce with no camp related costs.

The average G&A costs over the LOM were estimated at CAD 3.00/t milled.

Table 21.15: General and Administration Operating Costs Summary

G&A Costs (M CAD)	Total	1	2	3	4	5	6	7	8	9	10	11	12	13	14
General Management	**26.6**	1.9	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	1.7	1.7	1.7	1.6
Accounting / Finance	**11.9**	0.6	0.8	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.6
Supply Chain	**31.7**	1.7	2.3	2.3	2.5	2.5	2.5	2.5	2.5	2.5	2.3	2.3	2.3	2.0	1.8
Information Technology	**14.6**	1.0	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.0	0.6
Human Resources	**24.4**	1.8	2.1	2.0	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.1	0.9
Health and Safety	**8.7**	0.5	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.5	0.3
Surface Support	**22.5**	1.3	1.8	1.7	1.7	1.7	1.7	1.7	1.7	1.7	1.7	1.7	1.7	1.4	1.3
Environment	**34.1**	1.9	2.4	2.4	2.5	2.6	2.6	2.4	2.6	2.5	2.4	2.5	2.5	2.4	2.4
Security	**7.2**	0.4	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
Corporate	**1.1**	1.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Customs, Taxes, Duties	**24.7**	1.3	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8	1.8
Insurance & Banking Fees	**35.8**	2.8	2.8	2.8	2.7	2.7	2.6	2.6	2.5	2.5	2.5	2.4	2.4	2.3	2.3
Other	**157.6**	17.2	12.9	14.7	15.8	9.0	6.2	5.7	9.0	8.9	10.6	13.8	18.0	13.1	2.8
Total G&A	**400.9**	**33.3**	**31.2**	**32.8**	**34.0**	**27.2**	**24.4**	**23.6**	**27.1**	**26.8**	**28.3**	**31.1**	**35.2**	**28.8**	**17.0**
Total G&A (CAD/t milled)	**3.00**	4.97	3.16	3.33	3.44	2.76	2.46	2.40	2.75	2.72	2.86	3.16	3.58	2.92	1.95

22. ECONOMIC ANALYSIS

This section presents all elements of the economic model which principally consist of metal production and revenues, royalty agreements, operating costs, capital costs, sustaining capital, salvage value, closure and reclamation costs, taxation and net Project cash flow.

The economic analysis is carried out in real terms (i.e. without inflation factors) in Q4 2019 Canadian dollars without any project or equipment financing assumptions. The economic results are calculated as of the beginning of Year -3, which corresponds to the start of the 39 months pre-production CAPEX phase, including engineering and procurement, with all prior costs treated as sunk costs but considered for the purposes of taxation calculations. The economic results such as the net present value ("NPV") and internal rate of return ("IRR") are calculated on an annual basis.

22.1 Assumptions

The key assumptions influencing the economics of the Project include:

- Gold price in USD/oz;
- Canadian dollar to United States dollar exchange rate ("CAD/USD");
- Diesel price in CAD/L;
- Natural gas price for power generation.

22.1.1 Gold Price

The base case gold price selected for the economic evaluation is USD 1,400/oz. This price assumption is supported by independent forecasts and consensus pricing.

22.1.2 Exchange Rates

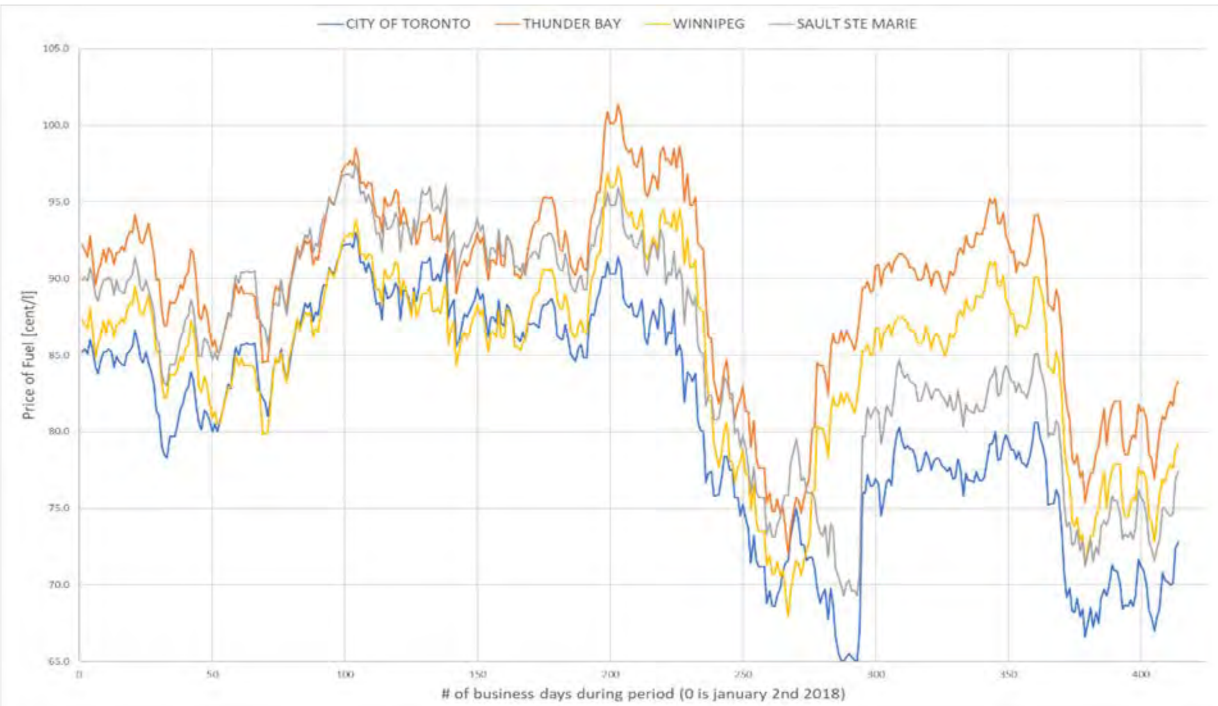
The base case Canadian dollar exchange rate for economic evaluation is CAD/USD 1.30. Most operating costs are estimated in Canadian dollars with the US dollar denominated gold revenue converted to Canadian dollars.

The Euro exchange rate assumption is CAD/EUR 1.47, which is relevant in purchasing some mining and power plant equipment sourced from Europe.

22.1.3 Fuel

The reference diesel fuel price used for estimating operating costs is CAD 0.89/L, which is an estimated delivered price to site for coloured diesel destined for off-road vehicles. It is exclusive of provincial road taxes and sales taxes which are reimbursable but includes the federal excise tax. The reference price is benchmarked from Toronto, Ontario rack price for ultra-low sulfur diesel no. 1. The price assumption is based on a 3-yr average price and includes a carbon tax cost assumption based on the 2019 rate.

Figure 22.1: Daily Rack Pricing Data for Wholesale Diesel from January 2nd, 2018 until July 31st, 2019



22.1.4 Natural Gas

The long-term natural gas price used in the power costs is the 2021 monthly figures from the forward prices of the Alberta Energy Company ("AECO") and increased thereafter by 3.5% to ensure some flexibility to absorb any potential market fluctuations. The assumption for the Report is an average life-of-mine ("LOM") price of CAD 3.90/GJ including transportation and charges to site. The power generation plant is fueled by natural gas and is therefore an important consumable for the processing cost.

22.2 Metal Production and Revenues

Gold production over the Project life is 5,051 koz based on an average recovery of 91.2%. Gold sales during pre-production is 45 koz, generating estimated revenue of CAD 78.9M (net of transportation, refining and royalty costs) which offsets pre-production CAPEX. Gold sales during operations are 5,007 koz (which include 4 koz of inventory sold at end of operations) and gross revenue is CAD 9,112M.

During the commissioning and start-up phase there is a build-up of 4 koz of inventory in the process circuit which is recovered at the end of operations. The commissioning and ramp-up schedule is presented in Beginning in Year 1, small tonnages are fed into the crushing circuit for cold commissioning, with mill processing starting at an average of 8 kt/d and ramping up to 18 kt/d over a 4-month period. At this point, commercial production is achieved with the plant processing 63% of nameplate throughput for 30 days, which meets the commercial production definition of at least 60% of nameplate throughput over 30 days.

Table 22.1. Beginning in Year 1, small tonnages are fed into the crushing circuit for cold commissioning, with mill processing starting at an average of 8 kt/d and ramping up to 18 kt/d over a 4-month period. At this point, commercial production is achieved with the plant processing 63% of nameplate throughput for 30 days, which meets the commercial production definition of at least 60% of nameplate throughput over 30 days.

Table 22.1: Mill Commissioning and Ramp-up

Mill Commissioning and Ramp-up	Tonnage (kt/month)	kt/d (avg.)	% Nameplate	% Gold Recovery
Pre-Prod month 1	242	8	29%	90.3%
Pre-Prod month 2	357	12	43%	91.1%
Pre-Prod month 3	400	14	53%	91.6%
Pre-Prod month 4	530	17	63%	91.6%
Commercial Production Year 1	745	25	91%	92.4%
Operations	819	27	100%[1]	91.2%

Note: Represents 100% of full nameplate capacity at 27,000 t/d

The annual mine and mill production is summarized in Table 22.2 and Figure 22.3. The gold production profile is presented in Figure 22.2. Commercial production is reached in Year 1 and ends in Year 14 for a 14-year mine life. Additional details on the production schedule are presented in Subsection 16.3.

Figure 22.2: Annual Gold Sales Profile

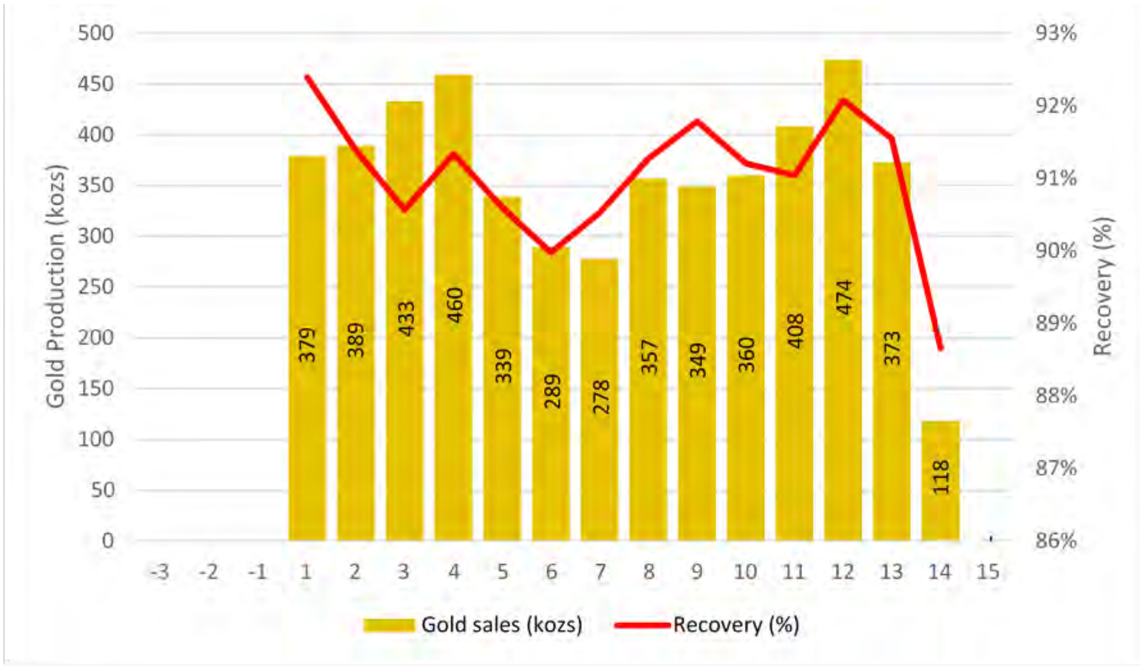


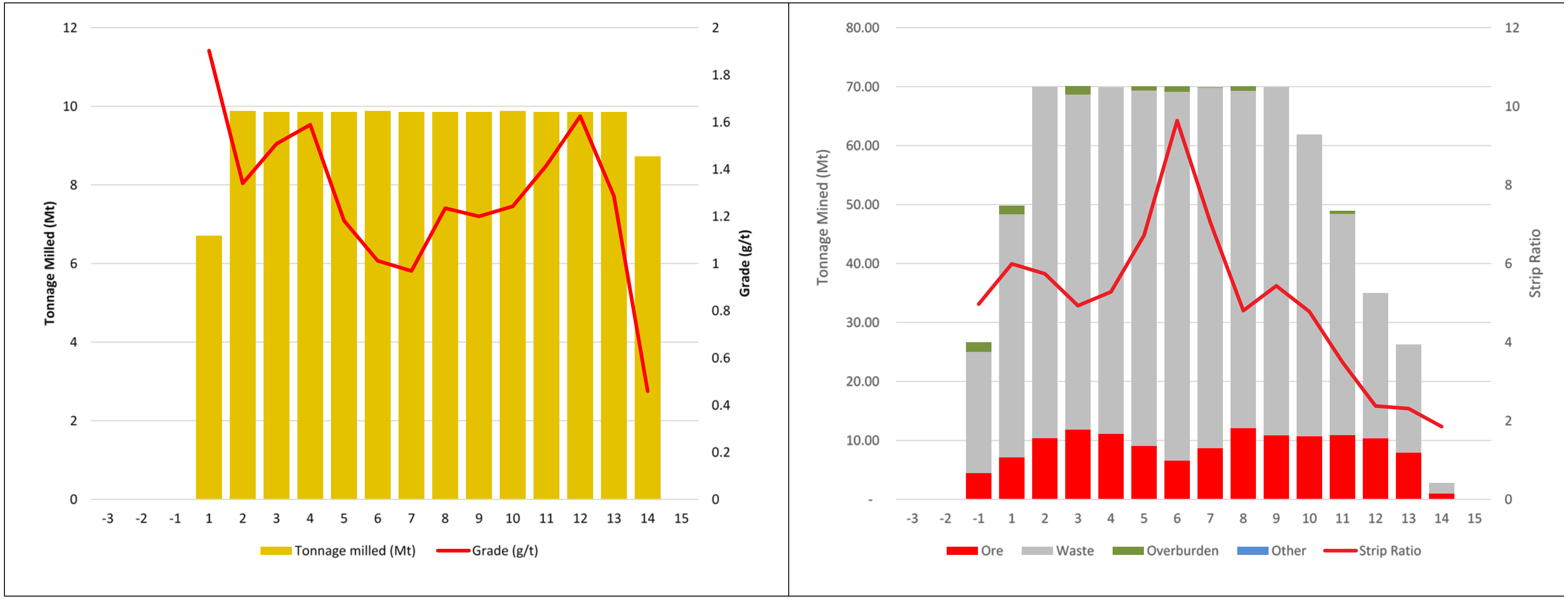
Table 22.2: Annual Mine and Mill Production Summary

Production Summary	Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Mill Production																					
Tonnage Milled (Mt)	**133.79**				6.71	9.88	9.86	9.86	9.86	9.88	9.86	9.86	9.86	9.88	9.86	9.86	9.86	8.73	-	-	-
Gold Processed (kozs)	**5,486**				410	426	478	503	374	321	307	391	380	395	449	515	407	129			
Head Grade (g/t)	**1.28**				1.90	1.34	1.51	1.59	1.18	1.01	0.97	1.23	1.20	1.24	1.42	1.62	1.29	0.46			
Gold Production (kozs)	**5,003**				379	389	433	460	339	289	278	357	349	360	408	474	373	114			
Recovery	**91.2%**				92.4%	91.4%	90.6%	91.3%	90.6%	90.0%	90.5%	91.3%	91.8%	91.2%	91.0%	92.1%	91.5%	88.7%			
Gold Sales (kozs)[1]	**5,007**				379	389	433	460	339	289	278	357	349	360	408	474	373	118	-	-	-
Mine Production																					
Waste	**670.51**	-	-	20.53	41.22	59.55	56.83	58.76	60.26	62.52	61.11	57.20	59.05	51.17	37.50	24.64	18.34	1.83	-	-	-
Overburden	**7.73**	-	-	1.66	1.48	0.01	1.45	-	0.72	0.95	0.11	0.80	-	-	0.55	-	0.00	-	-	-	-
Ore	**133.13**	-	-	4.47	7.13	10.38	11.82	11.13	9.08	6.59	8.68	12.08	10.87	10.71	10.90	10.37	7.95	0.99	-	-	-
Total Mined (Mt)	**811.38**	-	-	26.66	49.83	69.94	70.10	69.89	70.05	70.06	69.89	70.08	69.92	61.89	48.95	35.01	26.29	2.81	-	-	-
Strip Ratio	**5.09**			4.97	5.99	5.74	4.93	5.28	6.72	9.64	7.06	4.80	5.43	4.78	3.49	2.38	2.31	1.85			

Note: 1. includes sale of gold in inventory

2. excludes pre-production

Figure 22.3: Mine and Mill Production Profile



22.3 Royalties

Certain mining claims in the Hardrock deposit are subject to a 3% net smelter royalty ("NSR") payable to Franco Nevada Corporation. Over the course of the LOM, payments under this royalty are expected to total CAD 273M.

22.4 Operating Cost Summary

Operating costs include mining, processing, G&A services (including estimated payments to Indigenous communities), transportation and refining of gold. The operating cost summary is presented in Detailed operating cost budgets have been estimated from first principles based on detailed wage scales, consumable prices, fuel prices and productivities. The transportation and refining cost used in the economic model is CAD 3.00/oz and is based on indicative pricing from a Canadian refiner.

Table 22.3.

Detailed operating cost budgets have been estimated from first principles based on detailed wage scales, consumable prices, fuel prices and productivities. The transportation and refining cost used in the economic model is CAD 3.00/oz and is based on indicative pricing from a Canadian refiner.

Table 22.3: Operating Cost Summary

Category	Total Costs (M CAD)	Unit Cost (CAD/t milled)	Cost per oz (CAD/oz)
Mining	1,890	14.13	378
Processing	968	7.23	193
G&A	401	3.00	80
Transportation & Refining	15	0.11	3
Royalties	273	2.04	55
Total Operating Cost	**3,547**	**26.51**	**708**
Closure & Reclamation	54	0.40	11
Sustaining Capital	420	3.14	84
All-in Sustaining Cost (AISC)	**4,020**	**30.05**	**803**

The average operating cost for the LOM is CAD 708/oz and is lower at CAD 640/oz for the first four years of commercial production. The costs increase in Year 5 to Year 7 when feed grade is lower and increases for the last few years when low grade stockpile ore is processed. The all-in sustaining cost ("AISC") which includes closure, reclamation and sustaining capital costs averages CAD 803/oz over the mine life.

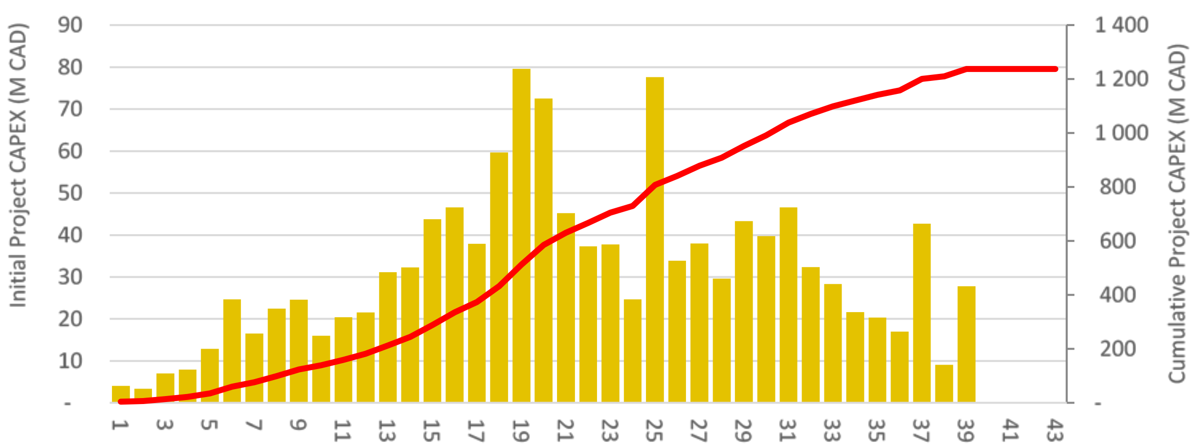
22.5 Capital Expenditures

The capital expenditures include initial capital ("CAPEX") as well as sustaining capital to be spent after commencement of commercial operations.

22.5.1 Initial Capital

The CAPEX for Project construction, including processing, mine equipment purchases, pre-production activities, infrastructures and other direct and indirect costs is estimated to be CAD 1,301M. The total initial Project capital includes a contingency of CAD 108M which is 9% of the total CAPEX. The initial capital including pre-production revenue and IBA payments during construction is estimated at CAD 1,226. The monthly CAPEX is presented in Figure 22.4.

Figure 22.4: Initial CAPEX by Month



The native currency assumptions of the CAPEX are 78% in Canadian dollars, 17% in US dollars and 5% in Euros.

22.5.2 Sustaining Capital Expenditures

Sustaining capital is required during operations for additional equipment purchases, mine equipment capital repairs, mine civil works and additional infrastructure relocation. The sustaining capital is estimated at CAD 420M (Table 22.4).

Table 22.4: Sustaining Capital Summary

Sustaining Capital Costs (M CAD)	
Mine Equipment Capital Repairs	165.9
Mine Equipment Purchases	156.8
TMF Dam Construction	85.2
Mine Civil Works	2.8
Geraldton Heritage Interpretive Center	4.5
Headframe Relocation	1.4
Golf Course Expansion	3.1
Total	**419.7**

22.5.3 Salvage Value

A salvage value is estimated for some mining equipment purchased during operations that will not have been utilized to its useful life. A residual value is estimated for some of the major process plant equipment such as grinding mills, crushers and tank agitators. The power plant will have a residual value as the units will have a remaining useful life of 10 to 15 years at the end of operations. The salvage value is summarized in Table 22.5.

Table 22.5: Salvage Value

Salvage Value	(M CAD)
Process Plant Equipment	15.8
Power Plant Equipment	20.0
Mine Major Equipment	9.4
Total Salvage Value	**45.2**

22.6 Working Capital

Working capital is required to finance supplies in inventory. Given the accessibility of the site, the working capital requirements are considered low compared to remote operations.

22.7 Reclamation and Closure Costs

Reclamation and closure costs include infrastructure decommissioning, site preparation and revegetation, maintenance and post closure monitoring. The reclamation cost is spent over three years at the end of operations and is guaranteed through issuance of a surety bond throughout the operation at a LOM cost of CAD 10.7M. The total reclamation and closure cost is estimated at CAD 54.1M, as summarized in Table 22.6. In addition to the actual closure costs are CAD 10.7M in bonding costs that are part of the general and administration operating costs.

Table 22.6: Reclamation and Closure Cost

Reclamation and Closure	(M CAD)
Progressive Rehabilitation	7.9
Linear Infrastructure	2.0
Plant Site Area	6.4
Tailings Management Facility (TMF)	7.8
Waste Rock, Overburden and Stockpiles	4.5
Water Management	3.8
Mine Hazards	0.4
Monitoring and Studies	9.0
Other	12.5
Total Reclamation & Closure	**54.1**

22.8 Project Financing

The economic model excludes any Project debt or equipment financing and is therefore 100% financed through equity for the purposes of the Report. The uses and sources of funds is summarized in Table 22.7. The funding requirement is CAD 1,238M.

Table 22.7: Funding Summary

Funding Summary	(M CAD)
Uses of Funds	
Construction Costs (incl. pre-prod revenue & IBA payments)	1,226.0
Working Capital Adjustments	11.3
Total	**1,237.9**
Sources of Funds	
Equity	1,237.9
Total	**1,237.9**

22.9 Taxation

Partnerships are not legal tax paying entities under the Canadian *Income Tax Act* as the income or loss is calculated at the partnership level and allocated to the partners. Centerra Gold Mines Inc. ("Centerra") and Premier Gold Mines ("Premier") will bear the responsibility for paying tax on profits generated by the Partnership. The after-tax results are based on the assumption that the Partnership is a taxable Canadian entity and tax is calculated based on the tax rules in Ontario. The calculations do not reflect the benefit of any historical tax positions held by either Centerra or Premier. Losses have been included from the date of the Partnership agreement. The Ontario mining tax, federal income tax and provincial income tax during the LOM totals CAD 1,195M.

22.9.1 Ontario Mining Tax

Ontario mining tax is levied at a rate of 10% on taxable profit in excess of CAD 0.5M derived from a mining operation in Ontario. There are specific guidelines for the calculation of profit and depreciation for the purpose of the Ontario mining tax. A mining tax exemption on up to CAD 10M of profit during a 3-year period is available to each new non-remote mine, of which Hardrock does not qualify. The total Ontario mining taxes are CAD 338M over the Project life.

22.9.2 Income Taxes

The federal and provincial income taxes have both been estimated from an identical taxable income which is arrived at by deducting the Ontario mining tax and various tax depreciations allowances. The federal income tax rate is 15% while the Ontario income tax rate is 10%. The total federal income tax is estimated at CAD 515M and the provincial income tax at CAD 343M.

22.10 Economic Results

The main economic metrics used to evaluate the Project consist of net undiscounted after-tax cash flow, net discounted after-tax cash flow or NPV, IRR and payback period. The discount rate used to evaluate the present value of the Project corresponds to the weighted average cost of capital. The discount rate represents the required rate of return that an investor would expect based on the risks inherent in achieving the expected future cash flows.

A 5% discount rate is commonly used for gold projects located in a developed and stable mining jurisdiction. The relative country and project risk are assessed as low for the Hardrock Project. Sensitivities have been presented at various discount rates ranging from 5 to 8% (Table 22.9).

A summary of the Project economic results is presented in Table 22.8 and the annual Project cash flows are presented in Table 22.11. The total after-tax cash flow over the Project life is CAD 2,716M and after-tax NPV 5% is CAD 1,364M. The after-tax Project cash flow results in a 3.2-year payback period from the commencement of commercial operations with an after-tax IRR of 20.1%.

Table 22.8: Project Economic Results Summary

Project Economics		Base Case Results
Production Summary		
Tonnage Mined	Mt	824.9
Ore Milled	Mt	135.3
Head Grade	g Au/t	1.27
Gold Processed	k ozs	5,539
Recovery	%	91.2%
Gold Production	k ozs	5,051
Cash Flow Summary		
Gross Revenue	M C$	9,112
Mining Costs (incl. rehandle)	M C$	(1,890)
Processing Costs	M C$	(968)
G&A Costs (incl. transport & refining)	M C$	(416)
Royalty Costs	M C$	(273)
Total Operating Costs	M C$	(3,547)
Operating Cash Flow Before Taxes	M C$	5,566
Initial CAPEX	M C$	(1,226)
Sustaining CAPEX	M C$	(420)
Total CAPEX	M C$	(1,646)
Salvage Value	M C$	45
Closure Costs	M C$	(54)
Interest and Financing Expenses	M C$	-
Taxes (mining, prov. & fed.)	M C$	(1,195)
Before-Tax Results		
Before-Tax Undiscounted Cash Flow	M C$	**3,911**
NPV 5% Before-Tax	M C$	**2,054**
Project Before-Tax Payback Period	years	**2.8**
Project Before-Tax IRR	%	**24.7%**
After-Tax Results		
After-Tax Undiscounted Cash Flow	M C$	**2,716**
NPV 5% After-Tax	M C$	**1,364**
Project After-Tax Payback Period	years	**3.2**
Project After-Tax IRR	%	**20.1%**

Table 22.9: Project Net Present Values at Various Discount Rates

Discount Rate	Before-Tax Project NPV (M CAD)	After-Tax Project NPV (M CAD)
5%	2,054	1,364
6%	1,804	1,183
7%	1,584	1,022
8%	1,389	879.9

22.11 Sensitivity Analysis

A sensitivity analysis was performed for ±10% and ±15% variations for gold price, exchange rate, OPEX and CAPEX. Each parameter was calculated independent of any correlations that may exist between variables such as for gold price and exchange rate, which tend to be negatively correlated.

The Project is most sensitive to gold price followed by exchange rate, initial capital costs and operating costs. The Project is somewhat less sensitive to the CAD / USD exchange rate than the gold price in USD/oz as some of the CAPEX are in US dollars. The sensitivity on gold grade is identical to that of the gold price and is therefore not presented in the following figures.

The results of the sensitivity analysis on the NPV 5% and IRR is presented in in Figure 22.5 and Figure 22.6 respectively.

Table 22.10: Project After-Tax Sensitivities

	NPV 5%			IRR		
FS Variable	-15 % (M CAD)	FS (M CAD)	+15 % (M CAD)	-15 % (% IRR)	FS (% IRR)	+15 % (% IRR)
Operating Costs	1,553	1,364	1,175	21.8%	20.1%	18.4%
Capital Costs	1,486	1,364	1,240	23.6%	20.1%	17.4%
Exch. Rate (CAD/USD)	837	1,364	1,885	15.1%	20.1%	24.4%
Gold Price	816	1,364	1,905	14.7%	20.1%	24.9%

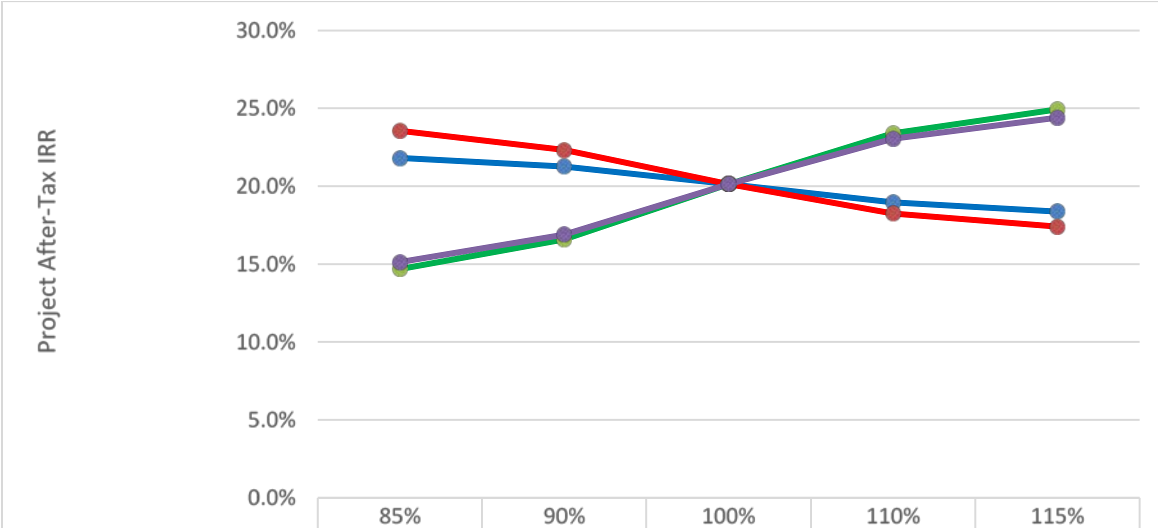
Table 22.11: Project Cash Flow Summary

Life-of-Mine Cash Flow	Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Sales and Revenue																					
Gold Price (USD/oz)	**1,400**	-	-	-	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400	1,400	-		
Exch. Rate (CAD/USD)	**1.30**	-	-	-	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30	-	-	-
Gold Sold (Koz)	**5,007**	-	-	-	379	389	433	460	339	289	278	357	349	360	408	474	373	118	-	-	-
Gold Revenue (M CAD)	**9,112**	**-**	**-**	**-**	**690**	**708**	**788**	**837**	**617**	**526**	**506**	**650**	**635**	**655**	**743**	**863**	**679**	**215**	**-**		
Operating Costs (M CAD)																					
Mining	**1,890**	-	-	-	102.4	151.3	154.6	160.7	161.2	159.4	158.9	164.6	163.9	140.5	134.0	122.0	100.7	16.1	-	-	-
Processing	**968**	-	-	-	48.3	75.2	72.0	71.9	69.3	72.4	69.6	71.7	71.6	69.3	72.3	71.8	71.4	60.7	-	-	-
G&A	**401**	-	-	-	33.3	31.2	32.8	34.0	27.2	24.4	23.6	27.1	26.8	28.3	31.1	35.2	28.8	17.0	-	-	-
Royalties	**273**	-	-	-	20.7	21.2	23.6	25.1	18.5	15.8	15.1	19.5	19.0	19.6	22.3	25.8	20.3	6.4	-	-	-
Refining & Other	**15**	-	-	-	1.1	1.2	1.3	1.4	1.0	0.9	0.8	1.1	1.0	1.1	1.2	1.4	1.1	0.4	-	-	-
Total Direct Costs	**3,547**	**-**	**-**	**-**	**205.8**	**280.1**	**284.3**	**292.9**	**277.2**	**272.8**	**268.1**	**284.0**	**282.4**	**258.7**	**261.0**	**256.3**	**222.4**	**100.7**	**-**	**-**	**-**
Capital and Other Costs (M CAD)																					
Construction Capital	**1,113**	179.0	524.2	374.7	35.6	-															
Contingency	**108**	1.3	16.4	57.2	33.4	-															
Other Capital	**4**	1.2	2.8	0.1	0.0	-															
Sustaining Capital	**420**	-	-	0.4	100.7	35.9	42.3	48.4	22.3	23.7	31.7	41.4	26.6	16.5	10.6	17.8	1.5	-	-	-	-
Working Capital	**(0)**	-	5.0	(3.7)	26.6	(5.8)	0.3	0.9	(4.2)	(3.7)	1.7	2.7	1.1	0.6	2.4	3.5	(0.4)	(27.0)	-	-	-
Interest & Fees	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Equip. Facility Funding	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Equip. Facility Repayment	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Reclamation Fund	**54**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1.5	17.5	17.5	17.5
Salvage Value	**(45)**	-	-	-	-	-	-	-	-	-	-	-	-	(3.6)	(0.8)	(0.2)	(0.5)	(4.0)	(36.1)	-	-
Total Capital & Other	**1,655**	181.5	548.4	428.8	196.4	30.0	42.6	49.2	18.0	20.1	33.4	44.1	27.7	13.5	12.2	21.1	0.6	(29.5)	(18.6)	17.5	17.5
Cash Flow (M CAD)																					
Pre-Tax Cash Flow	**3,911**	(181.5)	(548.4)	(428.8)	287.8	398.0	460.7	494.6	322.2	233.6	204.1	321.8	325.1	383.1	470.1	585.4	455.8	143.8	18.6	(17.5)	(17.5)
Cash Taxes	**1,195**	-	-	-	2.5	46.8	91.1	117.4	65.6	44.9	45.1	96.8	95.0	113.6	143.3	173.8	134.0	25.5	-	-	-
After-Tax Cash Flow	**2,716**	(181.5)	(548.4)	(428.8)	285.4	351.2	369.6	377.1	256.6	188.6	159.1	225.0	230.2	269.6	326.8	411.6	321.7	118.3	18.6	(17.5)	(17.5)

Notes:

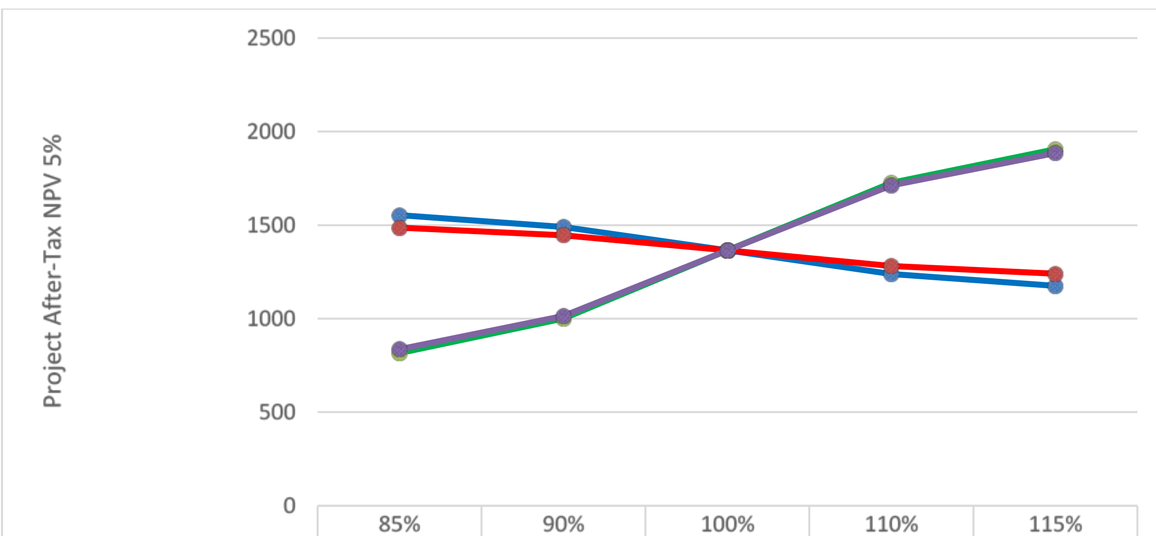
1. *Non-GAAP measure.*
2. *Pre-production gold sales treated as credit against pre-production costs in construction capital.*
3. *Numbers may not add due to rounding.*

Figure 22.5: Project After-Tax NPV 5% Sensitivity



	85%	90%	100%	110%	115%
OPEX	21.8%	21.3%	20.1%	19.0%	18.4%
Initial CAPEX	23.6%	22.3%	20.1%	18.2%	17.4%
Gold Price	14.7%	16.6%	20.1%	23.4%	24.9%
CAD/USD Exch. Rate	15.1%	16.9%	20.1%	23.0%	24.4%

Figure 22.6: Project After-Tax IRR Sensitivity



	85%	90%	100%	110%	115%
OPEX	1553	1490	1364	1238	1175
Initial CAPEX	1486	1446	1364	1282	1240
Gold Price	816	1000	1364	1725	1905
CAD/USD Exch. Rate	837	1014	1364	1712	1885

23. ADJACENT PROPERTIES

23.1 Overview

The qualified person responsible for this section was not able to verify the information pertaining to the Talmora and Tombill Mines, and the information presented in this section is not necessarily indicative of the mineralization on the Hardrock Property.

There are no adjacent properties that have any significant information relating to the Project. Greenstone Gold Mines ("GGM") maintains a significant land position in the Geraldton mining camp, and most of the camp's historical mineral deposits (Table 23.1 and Figure 23.1) are located within the boundaries of the GGM projects. Two exceptions are the historic Talmora and Tombill Mines that are held by others.

Table 23.1: Gold Production Statistics for the Bankfield, Little Long Lac, Magnet, Talmora Long Lac and Tombill Mines (from Ferguson et al., 1971; Mason and White 1986)

	Bankfield Mine	Little Long Lac Mine	Magnet Mine	Talmora Long Lac Mine	Tombill Mine	Total
Production Years	1937-1942, 1944-1947	1934-1954, 1956	1936-1943, 1946-1952	1942, 1947-1948	1938-1942, 1955	
Ore Milled (short tons)	229,009	1,782,516	359,912	9,570	190,622	2,571,629
Ore Milled (metric tonnes)	207,757	1,617,099	326,512	8,682	172,933	2,332,983
Au Grade (oz/t)	0.290	0.340	0.423	0.147	0.361	0.348
Au Grade (g/t)	9.94	11.65	14.49	5.04	12.36	11.92
Gold Ounces	66,416	605,449	152,089	1,415	69,120	894,489
Silver Ounces	7,590	52,750	16,879	66	8,595	85,881

23.1.1 Talmora Long Lac

This description was, for the most part, taken from Ferguson et al. (1971) except where otherwise noted.

The past-producing Talmora Long Lac Mine is located in Errington Township, on the south side of Barton Bay, Kenogamisis Lake, and 4 km southwest of the Town of Geraldton (Figure 23.1).

Between 1934 and 1936, an extensive surface trenching and diamond drilling program was performed by Longlac Lagoon Gold Mines, revealing three mineralized zones.

Between 1938 and 1940, a shaft was sunk to a depth of 544 ft (165.8 m) with levels at 195 ft (59.4 m), 315 ft (96.0 m) and 515 ft (157.0 m) on which 4,796 ft (1,461.8 m) of drifting and 1,038 ft (316.4 m) of cross-cutting were done. Diamond drilling done included 400 ft (121.9 m) from surface and 2,449 ft (746.5 m) in four underground holes. All the work was performed by Elmos Gold Mines Ltd.

Between 1940 and 1942, trenching, stripping and two underground diamond drill holes (“DDH”) totalling 234 ft were carried out by Tombill Gold Mines Ltd. A small 50 t mill was constructed on the mine site during winter of 1941-1942. Underground work resumed in March 1942, and during the summer, 1,017 oz of gold and 36.5 oz of silver were produced from 3,947 t of sorted material. Due to the unfavourable wartime conditions, operations were suspended in November of the same year.

Between 1947 and 1948, Talmora Longlac Gold Mines Ltd. completed 1,663 ft (506.9 m) of drifting and 670 ft (204.2 m) of crosscutting. Diamond drilling comprised of four surface holes totalling 139 ft (42.4 m) and 91 underground holes totalling 10,776 ft (3,284.5 m). From the start of milling on September 15, 1947 until the cessation of operations on March 31, 1948, a total of 398.5 oz of gold and 30 oz of silver were produced from 5,623 t of hoisted material, for an average grade of 0.07 oz Au/t.

At the time operations were suspended, it was estimated that about 12,000 t with an average grade of 0.37 oz Au/t remained in the mine (Pye, 1951). These “reserves” are historical in nature and should not be relied upon. It is unlikely they conform to current National Instrument 43-101 criteria or to CIM Definitions Standards and they have not been verified to determine their relevance or reliability.

In 1968, some geophysical work was carried out by Tombill Mines Ltd.

The geology of the mine consists of greywackes with interbeds of iron formation intruded by a diorite mass, folded into a westerly-plunging anticline (Pye, 1951). A felsic intrusive occurs as a sill-like mass on the south limb. Two steeply dipping diabase dykes up to 30 m wide cross the anticline in a northerly direction. Shear zones striking N060° to N080° and dipping 45° near the diorite-greywacke contact contain quartz lenses averaging less than 30 cm thicknesses. The main sulfides are pyrite and arsenopyrite.

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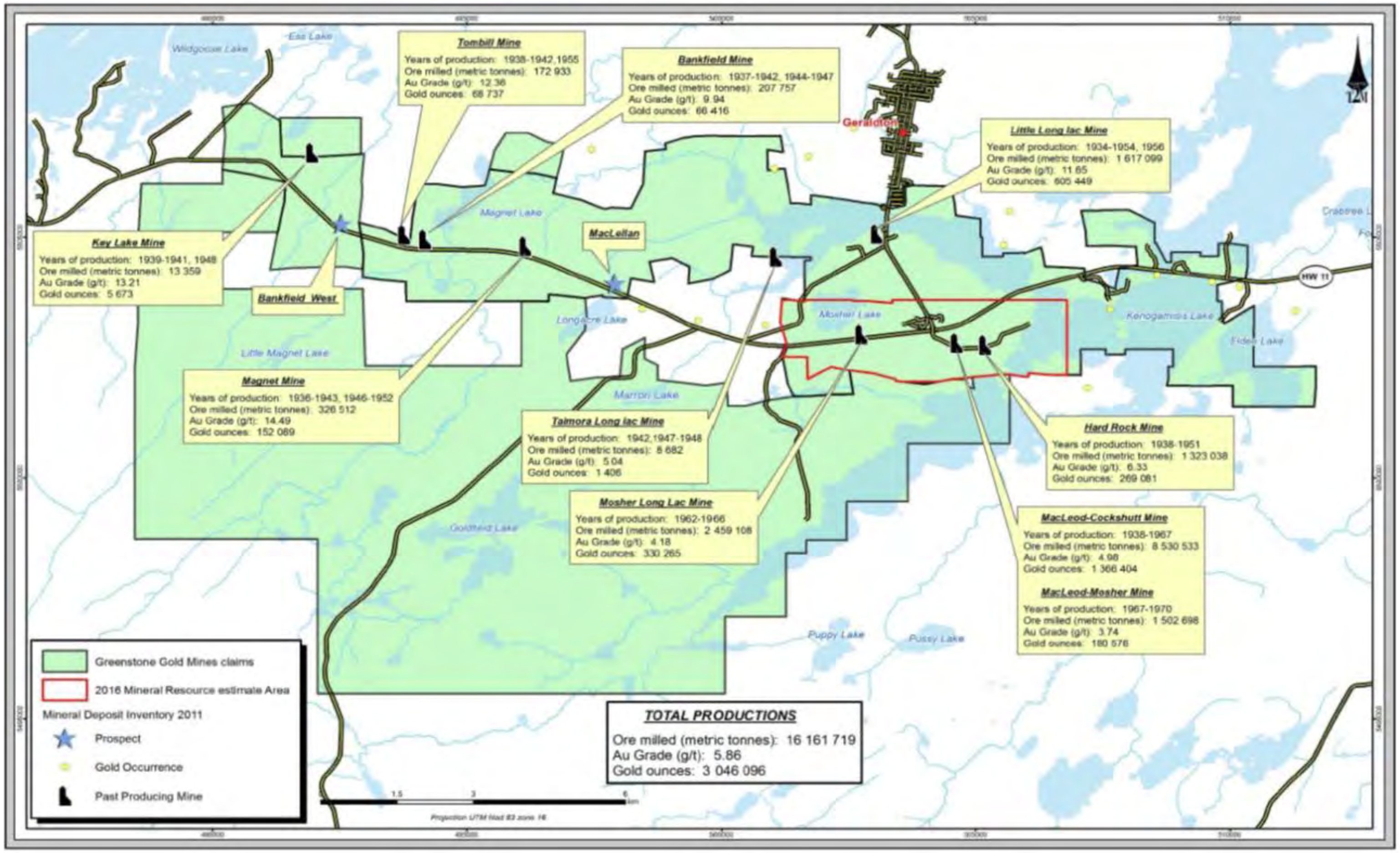
Figure 23.1: Past Gold Producers on the Hardrock Project



Source: Innovexplo, 2015

23.1.2 Little Long Lac Mine (Past Producer)

This description was, for the most part, taken from Ferguson et al. (1971) except where otherwise noted. The past-producing Little Long Lac Mine is located in the southeastern part of Errington Township, extends eastwards into Ashmore Township and is bounded to the north by Kenogamisis Lake. The Little Long Lac Mine is located about 2 km south of Geraldton (Figure 23.1).

Between 1933 and 1953, a shaft was sunk to a depth of 2,318 ft (706.5 m) with levels at 200 ft (61.0 m), 300 ft (91.4 m), 445 ft (135.6 m), 570 ft (173.7 m), 695 ft (211.8 m), 850 ft (259.1 m), 1,000 ft (304.8 m), 1,152 ft (351.1 m), 1,300 ft (396.2 m), 1,450 ft (442.0 m), 1,600 ft (487.7 m), 1,750 ft (533.4 m), 1,900 ft (579.1 m), 2,050 ft (624.8 m) and 2,200 ft (670.6 m). From level 2,200, a winze was sunk to a depth of 3,952 ft (1,204.6 m), with levels at 2,405 ft (733.0 m), 2,558 ft (779.7 m), 2,711 ft (826.3 m), 2,864 ft (872.9 m), 3,013 ft (918.4 m), 3,159 ft (962.9 m), 3,309 ft (1,008.6 m), 3,459 ft (1,054.3 m), 3,609 ft (1,100.0 m), 3,759 ft (1,145.7 m) and 3,920 ft (1,194.8 m). Drifting totalled 37,370 ft (11,390.4 m) and crosscutting 10,596 ft (3,229.7 m). Diamond drilling undertaken from surface totalled 105,626 ft (32,194.8 m), and underground drilling totalled 101,558 ft (30,954.9 m). A 150-ton mill was installed, and a small mill for scheelite production was added later. The work was performed by Little Long Lac Gold Mines Ltd.

From 1934 to 1954 and in 1956, a total of 605,409 oz of gold and 52,750 oz of silver were produced from 1,780,516 t of hoisted material. The average gold recovery was 0.34 oz/t of ore milled.

Between 1967 and 1968, Little Long Lac Gold Mines Ltd drilled a total of 5,000 ft (1,524 m) to test the iron formation.

The geology of the mine consists of arenaceous metasediments with interbeds of iron formation and some mafic intrusive rocks that have been folded into a synclinal structure striking N272° (Pye, 1951). The deposits occur within the fracture zones of massive quartz greywacke on the drag-folded north limb of the syncline. The Main vein zone is 3 to 4 ft wide (0.9 to 1.2 m), strikes approximately N075°, dips 80°, and consists of two parallel veins 2 to 6 in wide (5 to 15 cm). Some mineralization was also extracted from the lower grade 09 vein zone located about 600 ft (183 m) to the south of the Main zone; this zone is about 2 ft wide (60 cm), strikes N065°, dips 85°, and contains scheelite. The metallic constituents of quartz veins, which rarely make up more than 2 or 3% of the mineralization, include arsenopyrite, pyrite, pyrrhotite, sphalerite, chalcopyrite, galena and gold.

23.1.3 Magnet Consolidated Mine (Past-Producer)

This description was, for the most part, taken from Ferguson et al. (1971) except where otherwise noted.

The past producing Magnet Consolidated Mine is located in the southwest part of Errington Township, about 8 km southwest of the Town of Geraldton (Figure 23.1).

The discovery of native gold on a small island in the southern part of Magnet Lake in 1931 initiated an intensive search for gold in the area. Between 1934 and 1936, trenching was performed by Magnet Lake Gold Mines and 24,641 ft of diamond drilling were carried out by Wells Mines Ltd. Drilling uncovering three mineralized zones, two of which, now known as the Magnet and Wells vein zones, showed considerable promise. In order to explore these zones jointly underground, the two companies amalgamated in 1936 to form the present Magnet Consolidated Mines Limited (Mason and White, 1986).

Between 1936 and 1940, a shaft was sunk to a depth of 1,115 ft (339.9 m) with levels at 203 ft (61.9 m), 328 ft (100.0 m), 480 ft (146.3 m), 630 ft (192.0 m), 780 ft (237.7 m), 930 ft (283.5 m) and 1,080 ft (329.2 m) on which 11,181 ft (3,408.0 m) of drifting and 1,943 ft (592.2 m) of crosscutting was done. A total of 13 underground DDH totalling 1,665 ft (507.5 m) was completed. A 100-ton amalgamation- floatation mill was built.

Between 1940 and 1952, the shaft was continued to a depth of 1,772 ft (540.1 m), with additional levels at 1,230 ft (374.9 m), 1,380 ft (420.6 m), 1,555 ft (474.0 m) and 1,730 ft (527.3 m). An inclined winze 228 ft long (69.5 m) was constructed between levels 9, 10 and 11. A winze was sunk 931 ft (283.8 m) from the 1,730 ft level to a total depth of 2,640 ft (804.7 m) with levels at 1,884 ft (574.2 m), 2,037 ft (620.9 m), 2,160 ft (658.4 m), 2,312 ft (704.7 m), 2,460 ft (749.8 m) and 2,610 ft (795.5 m). Drifting totalled 19,585 ft (5,969.5 m) and crosscutting 2,944 ft (897.3 m). The company drilled seven surface DDH for a total of 4,029 ft (1,228.0 m) and 265 underground holes for a total of 43,054 ft (113,122.9 m).

From 1938 to 1943 and from 1946 to 1952, 152,089 oz of gold and 16,879 oz of silver were produced from 359,912 t of hoisted material. Average gold recovery was 0.42 oz/t.

The geology of the mine consists of metasediments, mostly greywacke with interbeds of iron formation and conglomerate, striking N290° and dipping 75 to 80°. Intrusive rocks consist of dykes and sill-like masses of diorite and porphyry and younger diabase dykes cutting across the formations (Pye, 1951). The two deposits, raking N300 to N315°, consist of lenticular quartz veins and accompanying veinlets predominantly in sheared greywacke. The Magnet vein zone, with an average strike of N285° and a near-vertical dip, was developed over a maximum length of about 1,300 ft (396.2 m). The leaner North zone, 50 to 100 ft (15.2 to 30.5 m) to the north, strikes N280° and dips vertically.

The deposits at the Magnet mine consist chiefly of quartz with small amounts of carbonate and subordinate sulfides. The metallic constituents, which seldom constitute more than 5% of the mineralization, are arsenopyrite, pyrite, pyrrhotite, chalcopyrite, sphalerite, galena and gold (Mason and White,1986).

23.1.4 Bankfield Mine (Past-Producer)

This description was, for the most part, taken from Ferguson et al. (1971) except where otherwise noted.

The past producing Bankfield Mine is located near the southwest part of Magnet Lake in the west-central part of the Errington Township and extends into Lindsley Township. This historical mine is situated about 10 km west-southwest of the Town of Geraldton (Figure 23.1).

The property was originally staked in October 1931 by T. A. Johnson and Robert Wells when they discovered gold-bearing quartz occupying a shear zone cutting a small reef in the southern part of Magnet Lake. Subsequent to this discovery, a mineralized zone was found by surface exploration about 1,000 ft (304.8 m) southwest of the lake. Surface-trenching and diamond drilling indicated sufficient material to merit development by underground methods.

Between 1934 and 1936, a shaft was sunk to a depth of 552 ft (168.2 m) with levels at 150 ft (45.7 m), 250 ft (83.8 m) and 525 ft (160.0 m). Drifting totalled 2,468 ft (752.2 m) and crosscutting 781 ft (240.6 m). Underground diamond drilling totalled 1,416 ft (431.6 m) and drilling from surface totalled 2,237 ft (431.6 m) during this period. Work was performed by Bankfield Gold Mines Ltd.

Between 1935 and 1942, a winze (located in Lindsley Township) was sunk from the 525 ft level to a depth of 1,297 ft (395.3 m) from the surface with levels at 779 ft (237.4 m), 900 ft (274.3 m), 1,025 ft (312.4 m), 1,150 ft (350.5 m) and 1,275 ft (388.6m). Sub-levels were established at 275, 400, 1,025 and 1,150 ft. Drifting totalled 14,516 ft (4,424.5 m) and crosscutting 7,832 ft (2,387.2 m). Diamond drilling included 132 underground holes totalling 21,628 ft (6,592.2 m), six surface holes totalling 2,328 ft (709.6m) and 10,145 ft (3,092.2 m) of unspecified drilling. A 100-ton cyanide mill was constructed. The work was performed by Bankfield Consolidated Mines Ltd.

From 1937 to 1942 and from 1944 to 1947, a total of 66,417 oz of gold and 7,590 oz of silver were produced from 231,009 t of hoisted material. The average gold recovery was 0.29 oz/t.

The geology of the mine consists of greywacke with bands of conglomerate, slate and iron formation striking N290 to 300° and dipping 75 to 80° (Pye, 1951). The rocks have been intruded by diorite and quartz porphyry, and ultimately by a 200 ft (61.0 m) wide diabase dyke which runs parallel to a strike fault near the mine workings. The main mineralized horizon, consisting of a sheared, brecciated and highly silicified zone, occurs near a contact between the sediments and a porphyry-diorite mass. It strikes N275 to N288°, dips 70 to 78°, has an average width of 7 ft (2.1 m) and is, including its extension into the adjacent Tombill Property, 2,000 ft long (609.6 m). The deposits at the Bankfield Mine consists mainly of sheared and silicified greywacke and porphyry, mineralized with sulfides and small amounts of gold, and are cut by numerous "opalescent" grey quartz veins. The reported metallic minerals are arsenopyrite, pyrite, pyrrhotite, sphalerite, chalcopyrite, galena and ilmenite.

23.1.5 Tombill Mine (Past-Producer)

This description was, for the most part, taken from Ferguson et al. (1971) except where otherwise noted.

The past producing Tombill Mine is located in the east-central part of Lindsley Township, about 10 km west-southwest of the Town of Geraldton (Figure 23.1).

Between 1935 and 1942, a shaft was sunk to a depth of 630 ft (192.0 m), with levels at 215 ft (65.5 m), 400 ft (121.9 m) and 600 ft (182.9 m) on which 3,762 ft (1,146.7 m) of drifting and 4,442 ft (135.9 m) of crosscutting were done. Diamond drilling comprised more than 12 surface holes totalling 15,570 ft (4,745.7 m) and 63 underground holes totalling 4,406 ft (1,342.9 m). A mill with a 100-ton capacity was erected, and was later increased to 150 t. All work was carried out by Tombill Gold Mines Ltd. In 1940, an agreement was reached allowing Bankfield Consolidated Mines Ltd to explore and develop a block below the 500-ft level.

From 1938 to 1942 and in 1955 (mill clean-up), a total of 69,120 oz of gold and 8,595 oz of silver were produced from 190,622 t of hoisted material. Average gold recovery was 0.36 oz/t. The geology of the mine consists of metasediments and felsic intrusive rocks along a sheared and fractured contact where mineralized zones developed. Associated minerals are pyrite, arsenopyrite and pyrrhotite.

23.1.6 Gold Potential of the Other Historical Mines

The information presented on historical gold mines near the Hardrock Project was obtained through the literature and not verified by GGM. The presence of significant mineralization on these adjacent historical mines is not necessarily indicative of similar mineralization at the Hardrock Project.

24. OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution and Organization

The Hardrock Project team is responsible to deliver all temporary and permanent infrastructure required to provide a fully operational mine capable of sustainably processing 27,000 t/d.

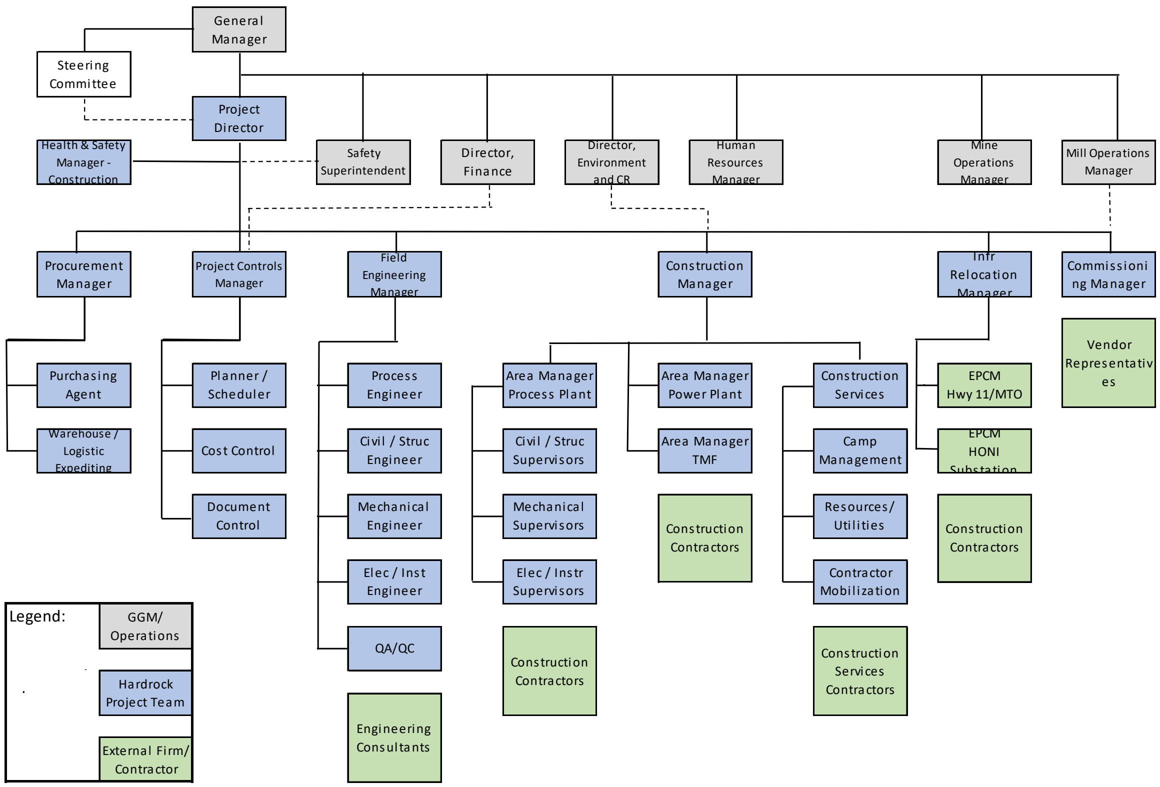
The Project will be executed using an "Owner-managed" project delivery model. Engineering and a portion of procurement will be outsourced, while construction and commissioning for the Project will be managed directly by the Owner. The execution strategy incorporates the key elements required for successful project development, including health and safety, environment, community relations, Indigenous relations, planning, project controls, document control, risk management, construction management and quality management.

The Hardrock Project team will manage health and safety, engineering, procurement and contracts, construction, and pre-commissioning, as well as overall Project management services during the Project. Greenstone Gold Mines ("GGM") operations will provide environmental management, Indigenous relations, community relations and certain site services to support to the Project team during construction. Overall responsibility for the site and facilities will transition to Operations at the start of hot commissioning of the mill.

An Independent Tailings Management Review Board ("ITRB") has been established for the Project. The purpose of the ITRB is to review and advise on the design, construction, operation, performance and closure planning for the TMF, and it will be in place through to closure.

The Project team will peak at 75 people during the construction execution phase. The Project organization chart is shown in Figure 24.1.

Figure 24.1: Hardrock Project Organization Chart



24.1.1 Health, Safety and Environment

GGM recognizes the protection of the health and safety of its employees, contractors, and the public as vital to the Company's existence and continued development. GGM is committed to conducting all its activities responsibly and providing a safe and healthy work environment.

GGM will leverage the existing Joint Venture Partner health and safety strategies and leadership programs. Health & Safety Management Plans, programs and procedures for the Construction phase will align with these strategies and leadership programs, will reflect industry practices, and will be established in such a way to be consistent with and facilitate a seamless transition to the Health and Safety Management Plans, programs and procedures for the Operations phase. All programs and standards will meet or exceed applicable laws and regulations, including but not limited to the Occupational Health and Safety Act of Ontario, and the Workplace Safety and Insurance Act.

All personnel working for the Project at site will be required to complete the Hardrock Project safety induction prior to starting work and must have all relevant mandatory training for specific tasks.

All environmental management and monitoring requirements during construction are the responsibility of the Operations Team, working in close collaboration with the Project Director and Construction Manager.

Environmental Management and Monitoring Plans (“EMMPs”) are being updated to support the permitting phase. The EMMPS include Project phase requirements, which will be consolidated into the Environmental Construction Management Plan and the relevant procedures. The plans and procedures will include management and monitoring, risk and hazard assessment and mitigation, contractor requirements, environmental incident reporting, investigation and follow-up, and reporting.

GGM’s Emergency Response Plan will evolve as the Project moves from the detailed engineering phase with limited site activities to full construction execution and operations. Ambulance services are available in the area and a hospital is located in Geraldton, a few kilometres from the site. During the Project phase, a nurse will be on-site to provide first aid. The Project will rely on local volunteer fire fighting services.

24.1.2 Indigenous Relations

The Operations team is accountable for all activities related to Indigenous relations and community relations during the Project phase. The Project team will support the Operations Team in fulfilling the commitments GGM has made with respect to Indigenous business opportunities, employment and training. This will help ensure all neighboring Indigenous and local communities will share in the opportunities and benefits of the Project.

In particular, the procurement and contracting processes for the Project phase will consider and reflect both the specific commitments in GGM’s Long terms Relationship Agreements (“LTRAs”) with Indigenous communities, as well as GGM’s overall objective to deliver long-lasting socio-economic benefits for Indigenous and local communities.

GGM will work to ensure that contractors and suppliers contribute to the positive socio-economic benefits of the Project. Prospective contractors and suppliers will be encouraged to consider how they can incorporate socio-economic considerations, including employment, workforce development and skills training within their proposals. Contractors will also be required to submit an Indigenous Participation plan as part of their proposal package. When evaluating proposals, GGM will consider the potential socio-economic benefits for Indigenous communities.

GGM will continue on-going consultation and engagement with Indigenous and local community members who reside within the geographical area influenced by the Project during all phases of the Project.

24.1.3 Engineering and Procurement Management

Detailed engineering has been outsourced to third party engineering providers. The mine design will also be outsourced during the initial phases of the Project and will be brought in-house once the operations mine department is established.

The detailed engineering scope has been divided into primary work packages. All detailed engineering packages have been awarded. In the case of the Hydro One relocation of the Power Lines and the Enbridge Natural Gas Pipeline, this engineering is being performed by the stakeholder/owner of the infrastructure. The engineering and installation of these facilities will be a cost to the Project.

Procurement and contracting activities include pre-qualification and selection of vendors, sourcing of equipment and bulk materials, expediting of goods and documentation deliverables, inspection surveillance of equipment and materials, transportation, logistics, and warehousing, field procurement, materials management during construction, as well as the contracting of all necessary engineering, consulting, construction or installation services.

24.1.4 Construction Management

The Project will be construction driven, and the construction management team will work with planning and engineering to influence the structure and timing of engineering deliverables, and to confirm the match of equipment and fabrication dates with required “on-site” dates. The master Project schedule will drive the work planning on site, and priority will generally be given to the critical path activities.

Labour relations strategies and plans will be implemented for construction. Construction will be on a 7-day / 10-hour schedule, except for start-up which will be on a 7-day / 24-hour schedule. The planned rotation is 14 days on site / 7 days leave for all Project team staff and contractors. A temporary camp will be installed a few kilometres outside of the mine site and close to Geraldton.

The dismantling, relocation and/or reconstruction of existing facilities will be managed by the construction management team in close collaboration with the local community and facilities owner. Certain other properties will be handled through compensation rather than being rebuilt.

Pre-commissioning activities will include final inspections and adjustment of equipment before it is tested and the testing of equipment and components of the plant in an energized state to ensure that the equipment can be handed over to the commissioning team in an acceptable operating condition for process commissioning. Operations representatives will be actively involved in the pre-commissioning process.

24.1.5 Operational Readiness, Commissioning and Ramp-up Strategy

The GGM Operations team will be accountable for all hot commissioning activities with the support and involvement of the Project team. A commissioning, handover and transition plan will be developed jointly with construction and operations teams for each facility.

The ramp-up period to commercial production is expected to take approximately six months (including two months of crushing commissioning activities). The pre-production phase of the Project is considered complete once commercial production is achieved which is defined as achieving 60% of the design throughput over a period of 30 days.

An operational readiness plan ("ORP") has been prepared for the entire mine site with a emphasis in the mine and plant areas given the large requirements for staffing and training in these areas. The ORP describes the philosophy and the additional short-term resources that are needed to quickly ramp-up to nameplate capacity. The ORP will be further elaborated to detail out the high-level plan that has been developed.

24.1.6 Risk Management

GGM's risk identification, assessment and mitigation processes will continue to be applied throughout the detailed engineering, construction, operation, and closure phases.

24.1.7 Quality Assurance and Quality Control

Quality Assurance and Quality Control ("QA/QC") will be applied to all levels of the Project. All designs will conform to the codes and standards applicable for the province, and applicable acts and regulations.

Engineering consultants and construction contractors will be responsible for their own quality control as specified in the terms and conditions of their contracts. Certain quality control work will be performed by third parties during execution.

The Project team will be responsible for ensuring that contractors follow their quality control programs, through the implementation of a comprehensive quality assurance program. Third parties will be used as required for on-site testing and laboratory analysis.

24.1.8 Project Controls

Project controls will be implemented to provide Project management and other stakeholders with transparent and timely information on project progress, performance and variances. This information will be used to assist management in setting priorities and making decisions that will support the delivery of the Project on time and on budget. Project controls includes planning and scheduling, progressing, cost control, change management, estimating, forecasting, earned value analysis and reporting. Robust change management processes will be in place to identify, mitigate and manage change. The schedule is structured to identify the critical path and link all activities. Measurement of physical progress, quantities and hours against the baseline will be a key component of the schedule updates.

24.2 Project Schedule

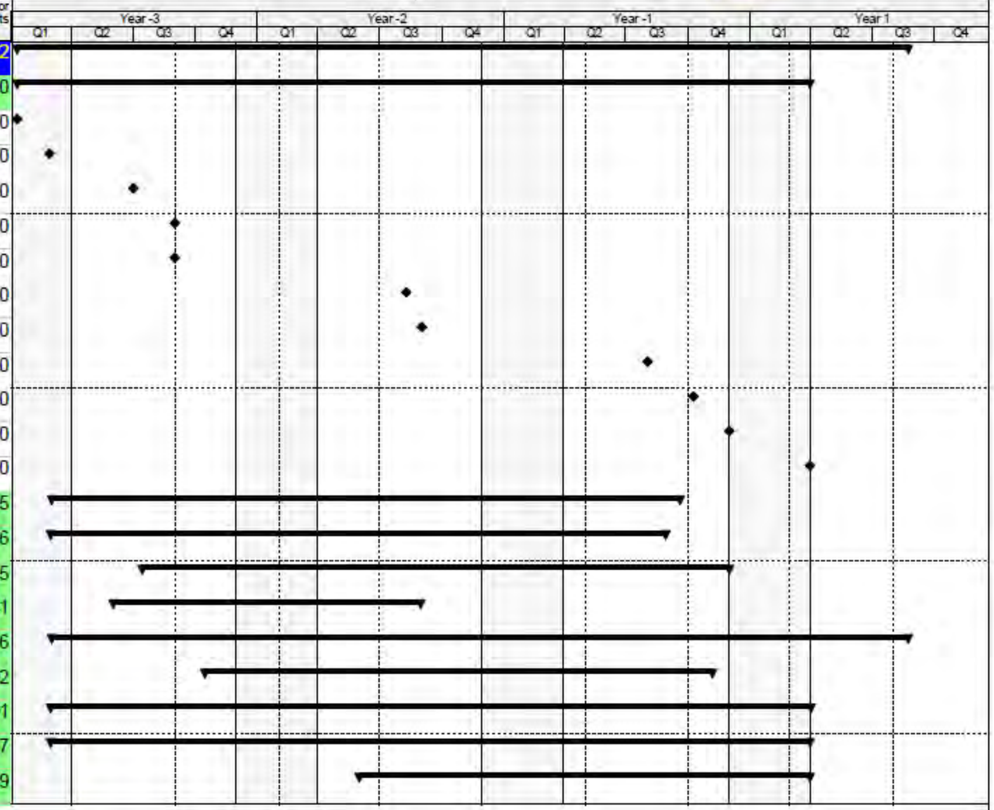
The Project executive summary schedule is shown in Figure 24.2. Detailed engineering will be completed ahead of construction ramp-up and is not on the critical path. Procurement is aligned with the required on-site dates and construction activities to meet schedule requirements, Concrete and structural steel have generally been scheduled for the non-winter months.

Figure 24.2: Hardrock Project Level 1 Schedule

GGM Hardrock - Level 1 Summary - Draft

GGM Hardrock Construction Schedule

Activity ID	Activity Name	Budgeted Labor Units
Total		3900722
000 General		0
MS-030	Commitments for Fabrication of Long Lead Equipment	0
MS-020	Construction Decision	0
MS-070	Commence Pre Construction	0
MS-090	Temporary Camp Operational	0
MS-100	Commence Construction	0
MS-180	Truck Shop Available for Use	0
MS-200	Commence Mine Pre Production	0
MS-230	Power Plant Commissioning Complete	0
MS-260	Process Plant Construction Complete	0
MS-280	Hot Commissioning First Ore	0
MS-290	Declare Commercial Production	0
100 Infrastructure		240305
200 Power & Electrical		160386
300 Water & Tailings Management		299345
400 Mobile Equipment		19241
500 Infrastructure Repositioning		184416
600 Process Plant General		843002
700 Construction Indirects		1197191
800 General Services - Owner's Cost		171487
900 Preproduction, Startup, Commissioning		785349

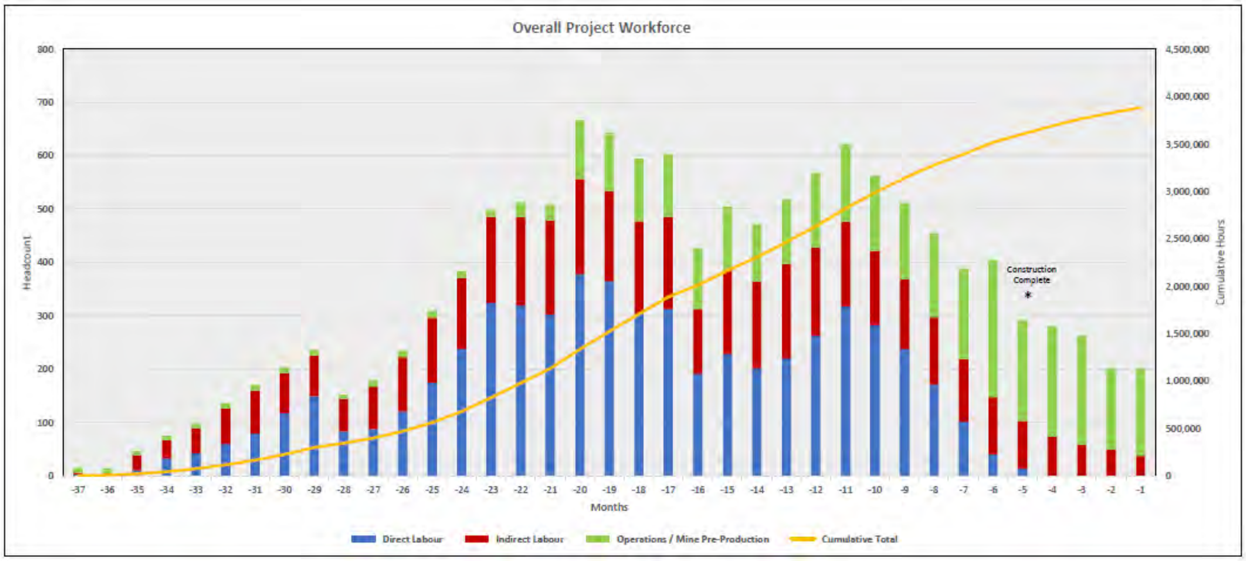


Actual Level of Effort — summary — Milestone

Page 1 of 1

The Project workforce requirements are summarized in Figure 24.3. The total direct and indirect construction and mine pre-production hours is estimated at 3.9 million hours and the on-site workforce peaks at approximately 660 people.

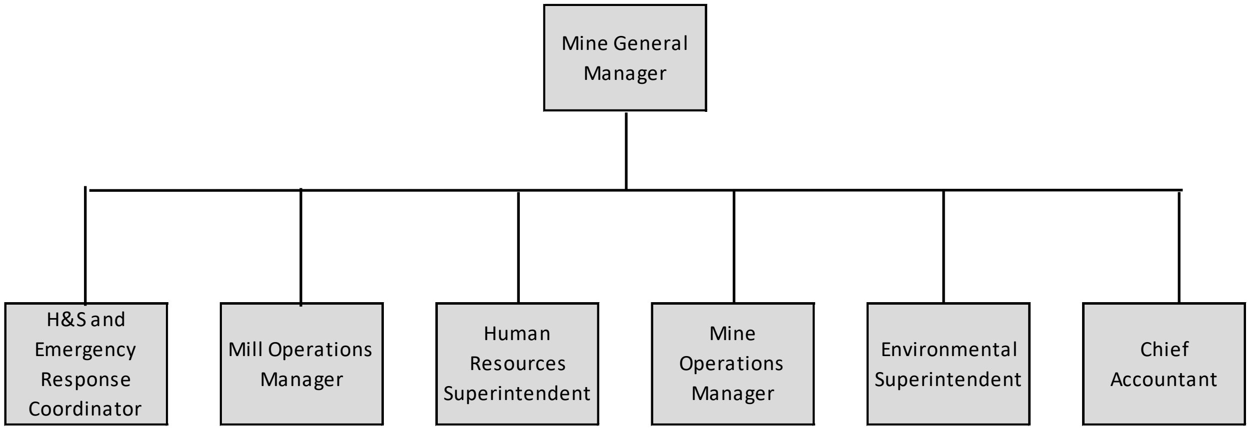
Figure 24.3: Hardrock Project Workforce



24.3 Operating Plan

The overall organization has three main areas: mine, process plant and administration. The mine department will include operations, maintenance, and mine engineering. The process plant department, which includes the power plant involves operations, maintenance and the metallurgical department. The operation organizational chart is presented in Figure 24.4.

Figure 24.4: Operation General Organizational Chart



The Hardrock General Manager has single point accountability for all aspects of the Hardrock site and business. The General Manager will be assisted by a core team of functional managers who will have accountability for front line and functional management of the various business areas. The operations workforce peaks at 521 in Year 6, based on a 24-hour, 7-day operation.

The management team members are accountable to deliver on the HSE, production, business objectives and to assure management systems are established and effective.

The Environmental Superintendent and H&S & Emergency Response Coordinator Safety will ensure that GGM meets or exceed the requirements of the environmental and occupational health and safety legislation respectively. This will be achieved by implementing and maintaining effective HSE management systems that drive continuous improvement.

The mine and process plant are planned to operate 24 hours/day, 365 days/year. A predictive and planned maintenance strategy will be implemented throughout the site and dedicated planners for each discipline will coordinate all maintenance procedures and work plans.

The mine is headed by a Mine Operations Manager who is responsible for the overall management of the mine. The chief mine engineer and chief geologist and superintendent positions in operations and maintenance report directly to the Mine Operations Manager.

The Technical Services group will consist of mine engineers, geologists, planners and technicians. The Technical Services group supports and services operations by ensuring that mine plans, systems, designs, records, budgets and schedules are in place and support the safe and efficient operation of the mine. The group will also provide engineering services, as required, for the process plant and related infrastructure.

The process plant, power plant, tailings area, and water management and treatment are the responsibility of the Mill Operations Manager. The mill operations superintendent, mill maintenance superintendent and chief metallurgist report directly to the Mill Operations Manager.

Primary and secondary crushing are ahead of the process plant stockpile area and will require dedicated operation and maintenance teams as the manipulations and overall maintenance aspects of the large machines are time intensive. A lower planned availability, as well as appropriate maintenance workforce, is planned for these systems.

To achieve industry standard plant on-line time, planned shutdowns will be a focus area as the high-pressure grind rolls, wet screens, grinding mills and miscellaneous conveying and feeding systems are interconnected. Plant shutdowns will be managed internally with external contracted assistance as required (workforce and specialized technical people).

The site accounting team reports to the chief accountant. The accounting department is responsible for budgeting, processing, measuring and reporting business results, in an efficient, accurate and timely manner.

A procurement system will be in place and will consist of market research, operation requirements planning, suppliers' management, purchasing and order controlling.

A warehouse management system and logistics system will be set-up that will ensure that each operating unit will be supplied with the required parts, products and consumables.

The information technology ("IT") department is primarily responsible for standardizing, operating and managing the IT systems, applications and infrastructure required by the business. This includes management of business information systems, communication infrastructure, data security and integrity.

Gold balance, bullion preparation, sale to clients as well as sales contract management will be under the supervision of the Mill Operations Manager. The Chief Accountant will be accountable for sales and revenue reporting, as well as gold security and general auditing practices.

25. INTERPRETATION AND CONCLUSIONS

25.1 Conclusions

Following the 2016 Feasibility Study ("FS"), the completion of the NI 43-101 Technical Report Feasibility Study Update has reconfirmed the technical feasibility and economic viability of the Project based on an open pit mining operation with average gold production at 366 koz per year over a 13.7-year life-of-mine ("LOM").

Table 25.1: Technical FS Update – LOM Results

Technical Report Feasibility Study Update Life-of-Mine Results	
Gold Price - Base Case (USD/oz)	1,400
Exchange Rate (CAD/USD)	1.30
Mine Life (operation yrs)	13.7
Strip Ratio (W:O)	5.10
Avg. Process Rate (kt/d)	26.8
Average Grade (g Au/t)	1.27
Average Gold Recovery (%)	91.2
Average Annual Gold Production (koz)	366
Total Recovered Gold (koz) Incl. Pre-prod.	5,051
Initial Capital Expenditures (CAD M) [1]	1,226
Sustaining Capital (CAD M	420
All-in Sustaining Cost (CAD/oz)	803
Project After-tax NPV5 (CAD M)	1,364
Project After-tax IRR (%)	20.1%

Note: 1. Including pre-production revenue credit and LTRA costs. Excludes working capital

The principal conclusions by area are detailed below.

25.1.1 Geology and Mineral Resources

- Since the 2016 MRE, there has been significant reverse circulation ("RC") and diamond drilling at the Hardrock Project. Drilling focused on de-risking the early years of production (RC grade control targeting the first 3 benches of production), in-filling gaps in the drill pattern and validating the new mineralisation interpretation.

- Cut-off grades of 0.30 g Au/t for the in-pit resource and 2.00 g Au/t for the underground resource are appropriate for reporting Mineral Resources for the Project.
- At a cut-off grade of 0.30 g Au/t, the in-pit Measured and Indicated Mineral Resources are estimated to be 137.7 Mt grading 1.33 g Au/t for 5.9 Moz of gold, inclusive of mineral reserves. In-pit Inferred Mineral Resources are estimated to be 0.9 Mt grading 1.19 g Au/t for 36 koz of gold.
- At a cut-off grade of 2.00 g Au/t, the underground Indicated Mineral Resources are estimated to be 9.8 Mt grading 3.93 g Au/t for 1.2 Moz of gold. Underground Inferred Mineral Resources are estimated to be 24.6 Mt grading 3.87 g Au/t for 3.1 Moz of gold.
- Definitions for Mineral Resource categories used in this report are consistent with the 2014 CIM definitions and prepared in accordance with NI 43-101.
- Since the 2012 Brookbank MRE by Micon International Ltd., more diamond drilling and resampling of drill core has occurred warranting an update to the MRE in 2020. Drilling focused on delineating the high-grade core of the deposit. No additional drilling has been undertaken at Key Lake or Kailey since 2012, however there has been significant advancements in the geological understanding of the orebodies since 2012.
- A cut-off grade of 0.40 g Au/t for the in-pit resource at Kailey and Key Lake was chosen, and 0.6 g Au/t cut-off was chosen for Brookbank, due to the distance from the envisaged Hardrock processing facility.
- A cut-off grade of 2.4 g Au/t was chosen for underground resources at Brookbank and is considered appropriate for reporting Mineral Resource for the Project.
- The Brookbank deposit consists of in-pit Indicated Mineral Resources of 1.147 Mt @ 2.24 g Au/t for 83 KOz of gold, and Inferred Mineral Resources of 0.045 Mt @ 2.07 g Au/t for 3 KOz of gold. The underground Indicated Mineral Resource is stated at 2.281 Mt @ 7.06 g Au/t for 517 KOz of gold, and Inferred Mineral Resources of 0.706 Mt @ 3.38 g Au/t for 77 KOz of gold.
- The Key Lake deposit consists of in-pit Indicated Mineral Resources of 3.761 Mt @ 1.16 g Au/t for 141 KOz of gold, and Inferred Mineral Resources of 1.836 Mt @ 1.39 g Au/t for 82 KOz of gold. No underground Mineral Resources are reported at Key Lake.
- The Kailey deposit consists of in-pit Indicated Mineral Resources of 11.276 Mt @ 0.96 g Au/t for 348 KOz of gold, and Inferred Mineral Resources of 4.858 Mt @ 0.87 g Au/t for 136 KOz of gold. No underground Mineral Resources are reported at Kailey.

25.1.2 Mining and Mineral Reserves

- The mine design and Mineral Reserve estimate have been completed.
- At a cut-off grade of 0.35 g Au/t, the Proven Mineral Reserves total 5.6 Mt at an average grade of 1.28 g Au/t for 232 k in-situ ounces of gold. The Probable Mineral Reserves total 129.7 Mt at an average grade of 1.27 g Au/t for 5,307 k in-situ ounces of gold. The total Proven and Probable reserve is 135.3 Mt at an average grade of 1.27 g Au/t for 5,539 k in-situ ounces of gold.
- The mining activities will occur over a period of 12.9-years (from start of commercial production to end of in-pit mining activities) excluding the pre-production period.
- The open pit generates 689.6 Mt of overburden and waste rock (inclusive of historic tailings and underground backfill) for a strip ratio of 5.1:1.
- The Mineral Reserve estimate stated herein is consistent with CIM definitions. The Mineral Reserves are based on Measured and Indicated Mineral Resources, and do not include any Inferred Mineral Resources.

25.1.3 Metallurgical Testing and Mineral Processing

- The process design criteria have been established based on test results, Owner and vendor recommendations and on industry practices.
- Processing options for the Project were selected based on the results of this testwork and are well known technologies currently used in the mining industry.
- Trade-off studies have determined that the ore processing plant should be sized at 27,000 t/d and using standard proven technology for crushing, grinding, thickening, leaching, CIP and gold recovery.
- The gold recovery process for the Project consists of a crushing circuit, a HPGR / ball mill grinding circuit, pre-leach thickening and cyanide leaching; CIP circuit, carbon elution and regeneration; electrowinning and gold refining; cyanide destruction and tailings disposal. The process plant is designed to operate at a throughput of 27,000 t/d.
- The overall gold recovery is 91.2% and is based upon metallurgical testing completed comprising of composite samples representing the full (global) deposit, early production years, lithological zones, low grade and near surface areas. The results demonstrate that the ore is amenable to gold recovery via cyanidation. Gold recovery is correlated to grind size, gold, sulphur and arsenic head grade. Block models have been created and each is assigned a gold recovery based upon the block attributes and the target grind size.

25.1.4 Infrastructure

- Existing infrastructure within the footprint of the property limits will need to be relocated or purchased and dismantled. The most significant relocation is that of the TransCanada Highway 11. All private properties within the project area have been purchased.
- Power availability from the existing grid is deemed insufficient and not reliable. Construction of a 65 MW natural gas-fired power plant is planned, with a designed capacity of 46.5 MW.
- As with the other main infrastructure, the administration building, truck shop, reagent storage and explosives plant and tailings management facility have been sized to support the mine and process operation.
- The historical seepage collection system and mine site collection ponds have been sized to handle normal flows with surface capacity for containing a 100-year return flood without discharge to the environment.
- Goldfield Creek ("GFC"), which currently traverses through the Tailings Management Facility ("TMF") footprint will be permanently diverted towards northeast to Kenogamisis Lake. The permanent GFC diversion channel design meets the fish offset guidelines. The GFC diversion dyke required for the diversion is designed in accordance with CDA and LRIA guidelines.
- The TMF has been designed in accordance with LRIA and CDA guidelines. The stability of the dams meets the target factors of safety required as per CDA. Tailings deposition plans have been developed in such a way that the wide tailings beaches abut the perimeter rock fill dams and pond being pushed to the west abutting natural ground.
- Seepage and runoff form TMF will be pumped back to the TMF. In case there is surplus water, which cannot be pumped into the TMF, water will be treated prior to discharging to the environment

25.1.5 Environmental Considerations

- The EIS/EA received Federal approval on December 13, 2018 and Provincial approval on March 12, 2019. The EIS/EA, including implementing the identified mitigation measures, supports the conclusion that the Project will not cause significant adverse environmental effects, including effects from accidents and malfunctions, effects of the environment on the Project and cumulative effects.
- Conceptual environmental management and monitoring plans ("EMMPs") were provided in the final EIS/EA and include measures related to both compliance and EIS/EA monitoring for all phases of the Project. The collective monitoring activities associated with the Project will also be used to inform

adaptive management for the Project, as required. The management and monitoring requirements have been incorporated into Project plans and budgets.

- Active consultation with stakeholders (community members, agencies and interested parties) and Indigenous communities has been undertaken throughout Project planning and will continue as the Project progresses.
- Greenstone Gold Mines (“GGM”) has established Long Term Relationship Agreements ("LTRAs") with the five local Indigenous communities. The agreements establish increased clarity regarding GGM’s ability to develop the Project and the Indigenous communities’ opportunity to benefit from future mining opportunities in the region, including the potential to extend the life of the Project.

25.1.6 Capital and Operating Costs

- The CAPEX estimate was based on material take-offs from advanced detailed engineering in most areas (50% to 90%), and firm price bids for the majority of process plant, power plant equipment, and the mine mobile equipment fleet. A QRA session was held to establish the contingency for the capex update.
- The initial CAPEX for Project construction, including processing, mine equipment purchases and pre-production activities, infrastructure and other direct and indirect costs, is estimated to be CAD 1,301M (before pre-production revenue and LTRA costs). The total initial capital includes a contingency of CAD 108M, which is 9.0% of the total CAPEX.
- Sustaining capital required during operations for additional equipment purchases, mine equipment capital repairs, mine civil works, TMF dam raises and additional infrastructure relocation is estimated at CAD 420M.
- A salvage value of CAD 45M is estimated for some mining and processing equipment and the power plant that will not have been utilized to their useful life.
- The total reclamation and closure cost are estimated to be CAD 54M.
- The average operating cost is CAD 708/oz Au or CAD 26.51/t milled over the LOM. The all-in sustaining cost (“AISC”) which includes closure, reclamation and sustaining capital costs average CAD 803/oz Au over the mine life.

25.2 Risks and Opportunities

25.2.1 Risks

GGM's risk identification and assessment process is iterative and has been applied throughout the FS and Detailed Engineering phases. Through a series of risk assessment sessions, risks are identified in relation to Project objectives and the internal and external context at the time of each assessment and are summarized into the Hardrock Project Corporate Risk Register. The risk evaluation process uses a 5 x 5 Impact and Likelihood matrix to rate risks at the enterprise level. A broad range of Project risk areas (technical, environmental, community, financial, health and safety, etc.) are assessed in order to provide a business or enterprise level perspective. Risk assessments by department or discipline are undertaken at the appropriate stages. Various standard engineering risk assessment processes, such as HAZOPs and Failure Mode and Effects Analysis ("FMEA") are undertaken during the detailed engineering phase. Standard health & safety risk assessment processes will be implemented for the construction phase.

Risk treatment plans are developed for each risk in order to reduce the risk's probability of occurring and/ or impact to an acceptable or practical level. Certain risk mitigation activities were completed as planned during the current project phase, while other actions are planned for construction, operations or closure phases as appropriate. These mitigation plans are incorporated in the project execution plans and where required in the CAPEX and OPEX budgets.

25.2.2 Discussion of Risks

The following is a discussion of the key risks for the Project with summaries of the related controls and risk mitigation strategies.

25.2.2.1 People

Ensuring that the right people are available at the right time, and the continuity of key positions is critical to ensure that the construction can be executed and delivered on schedule and budget to achieve the planned ramp-up. Comprehensive mitigation measures to ensure a smooth execution of the project have been considered and incorporated into Project plans. Initial work on developing and fostering the desired GGM culture is in progress. An overall project delivery strategy based on an "Owner Managed" model has been established and staffing plans have been established. Engineering and selected procurement activities will be outsourced. During construction, work will rely on the use of experienced contractors. Preliminary relocation, compensation and retention strategies and policies have been developed and are reflected in the capital and operating cost estimates. Work is underway to develop the training programs that will be required to ensure LTRA commitments will be met.

25.2.2.2 Gold Production

Gold production will depend on a number of factors such as accuracy of the resource and diluted models (tonnes and grade), mill throughput and recovery.

Significant drilling has occurred since the 2016 FS to improve the drill density of the early mining phases and to validate the updated resource model particularly within the 5 year pit extents. The recent drilling has confirmed that the block model is a good predictor of the expected gold grade. The resource model has been peer reviewed. The resource grade of the new resource model has been verified in two recent drilling campaigns; however, actual results may vary as the deposit is mined particularly beyond the 5 years pit extents. Similarly, although the 2019 diamond drilling campaign intersected very close to the 2018 modelled wireframes, resource tonnages may be different than expected. As the open pit is developed in the early years, an improved understanding of the geology, structures and reconciliations will lead to an updated and improved resource model.

The expected milling on-line time is 92% and is based upon on a good plant design and the execution of good operating and maintenance practices. An online simulation study was carried out following the FS and supported this figure. Some challenges may include the operation of the primary crusher, secondary screen and crusher as one system prior to feeding the stockpile storage area which may place additional demands and require additional flexibility on the mining operation to maintain an adequate coarse ore stockpile level to minimize ore shortages. The throughput may be mitigated from the HPGR process which will generate microcracks and reduce the grinding power necessary to achieve the targeted grind size and tonnage if the ball mills were the bottleneck.

Arsenic and sulphur models have been created and the results are available for each ore block within the block model to estimate the expected gold recovery from a multivariable regression analysis based on grind size, arsenic, sulphur, and gold head grades. The metallurgical regression analysis was based on the metallurgical testwork results obtained. The metallurgical testing and subsequent regression analysis does not cover the full spectrum of the ore expected to be processed particularly when low ratio material (gold to arsenic or gold to sulphur) is processed and metallurgical performance may be lower than expected. During operations, on-going optimization of the metallurgical performance will be carried out via leach testwork and throughput vs grind size tradeoffs will be evaluated on a regular basis in conjunction with anticipated gains from the HPGR circuit due to microcracking.

25.2.2.3 Permitting

With ongoing constraints in the public sector, GGM is monitoring the risk of agencies not meeting reasonable or the mandatory timeframe for permitting approvals. Consultation with Indigenous communities and agencies is undertaken on all key permit applications prior to submission to facilitate the approval timeframes.

25.2.2.4 Tailings Management Facility

Risks identified in relation to the TMF are reviewed for all phases of work including design, permitting, construction and operations. The TMF design is based on significant geotechnical drilling and hydrogeological field work, which were completed to support the design basis. Various dam construction risks assessments were made with a more conservative core upstream design chosen. A detailed risk assessment was undertaken by Wood on the preferred alternative to understand the impact of extreme weather resulting in too much or too little water in the TMF.

A detailed Tailings Facility Construction Management Plan, including a quality assurance/quality control program, will be developed and implemented for construction. A dam deposition plan and a dam raising schedule has been developed to ensure capacity for the mill tailings during operations.

An Independent Tailings Review Board ("ITRB") was established at the start of the detailed engineering phase to provide oversight during the lifecycle of the TMF. The purpose of the ITRB is to review and advise on the design, construction, operation, performance, and closure planning for the TMF. Recommendations from the ITRB have been incorporated into the detailed design of the TMF. Consultation with stakeholders has been occurring and will continue throughout permitting process.

25.2.2.5 Project Cost Estimate

The CAPEX estimate was updated in 2019 based on material take-offs from advanced detailed engineering (50 to 90%) for the process plant, TMF, Goldfield Creek and Highway 11 realignment. The CAPEX update for the majority of process plant, power plant equipment, effluent treatment plant, sewage treatment plant and mine mobile equipment fleet was based on firm price bid process, and in many cases Letters of Intent have been issued, pending a Construction Decision. Budgetary bids, quotes, estimating databases and historical data have been used for the remainder of the capex update. A Quantitative Risk Assessment ("QRA") session was held to establish the contingency for the CAPEX update. The Project execution philosophy and plans are aligned with the CAPEX, and strategies and plans are in place to ensure a highly experienced team is hired and retained to deliver on the plan. Robust planning, cost control, and change management processes are in place and will be scaled for the construction execution phase.

25.2.2.6 Accidents and Incidents

Ensuring the right systems and people are in place to manage health, safety and environmental risks is of paramount importance to GGM. The expectation is that everybody leaves the site the same way they arrived is one of GGM's core values. The Hardrock project will develop a health and safety management plan and will comply with the Occupational Health and Safety Act of Ontario (R.R.R 1990, Regulation 854) and all other health and safety regulations for mines operating in Ontario. All procedures and training will be carried out in compliance to the regulations. Area managers are accountable to ensure that procedures meet or exceed minimum standards for protecting the health and safety of all personnel. Compliance to the policies and procedures is the responsibility of every individual and not solely that of the health and safety department.

Draft environmental management plans (17) covering environmental management requirements have been prepared and are in the process of being finalized.

25.2.2.7 Pit Wall Failure

A sound comprehensive Pit Slope Management Program will be put in place by the Hardrock geotechnical engineering department to manage risks attributable to potential movement of the exposed rock faces. Rock mass failure is considered a low risk at Hardrock due to the high overall rock mass strength. Design elements have included a temporary wall slope profile that allows for wider catch benches to manage overbank hazards. The final design of the pit will evolve through the mine life considering information collected during the interim pit phases. Slope movement monitoring is also planned.

25.2.2.8 Stability of Historical Tailings

Geotechnical investigations and a stability analysis of the MacLeod High Tailings ("MHT") have been completed to assess the risk of overloading the historical tailings. Infrastructure already exists on these historical tailing with no incidents reported. As a result of these investigations and analysis, a stability buttress was added, in order to protect the integrity of the new highway and to prevent any further contamination to the lake. The buttress has been designed to hold the historical tailings in place in case of an earthquake that could potentially liquefy the historical tailing under the highway. The buttress will be constructed using mine rock along the north section of the MHT. The majority of the remaining perimeter of the MHT is adjacent to the waste rock dump or the mine haul road, providing stability. The final stability analysis based on the buttress design has concluded that required stability will be achieved.

Proper mining practices will be implemented when mining proximal to the historical tailings, especially focusing on controlling the vibrations attributable to blasting activities. An emphasis will be placed on avoiding the exposure of historical tailings faces for a long period of time. A rock dam will be put in place following the advancement of the excavation.

GGM is also pursuing the optimization of geotechnical concepts with specialized consultants aimed at improving future practices when dealing with historical tailings.

25.2.2.9 Relocation of Infrastructure

The Project requires a variety of existing infrastructure to be relocated in order to accommodate the proposed pit and plant site. Accordingly, there are risks associated with such proposed relocation. For example, there is a risk associated with delays in obtaining permits or a refusal in granting permits for the Highway 11 realignment work as a section of the realignment must be constructed over existing historical tailings. Detailed design and geotechnical investigations for this work have been completed. A mobile water treatment system will be in place during construction to treat any contact water. An engineering company familiar with the highway-related technical and regulatory processes in northwestern Ontario has been engaged to undertake the work and lead the permitting process.

Preliminary engineering has been completed by Enbridge for their segment of the natural gas pipeline. Discussions are well advanced with Hydro One for the relocation of their transformer station.

25.2.2.10 Water Management

The Project is surrounded on three sides by lakes and is cross-cut by small streams. There are several risks associated with construction activities and the use, treatment and discharge of water during operations and closure. These risks and associated treatment plans are as follows:

- Groundwater modelling and laboratory testing have been undertaken to understand the risk of unacceptable contaminants such as arsenic seeping from the TMF, historical tailings, and waste rock storage areas. Design elements include seepage collection ditches, temporary water treatment during construction and collections ponds that allow for water to be recycled to the plant during operations to ensure the required water quality objectives are met. Arsenic loading has been modelled.

- Federal permits for Schedule 2 of the MMER, and Section 35 for the Department of Fisheries and Oceans Canada and Environment Canada ("DFO") have been submitted to federal agencies. The DFO has accepted the Goldfield Creek compensation strategy prior to the application submission. Environment Canada has indicated that they will recommend a streamlined approval approach, which will reduce the Schedule 2 approval process by six to eight months. The DFO Section 35 Offsetting for Goldfield Creek authorization is expected to be received in Q4 2019.

- The risks related to water ingress into the open pit is deemed to be manageable as the historical dewatering rate was reasonable and the permeability of the rock is low as determined by geotechnical work. Pit dewatering is planned to be a minimum 20 m below the lowest mining bench elevation.

- The operation will be relying on water collected from underground workings, open pit and collection from surface drainage as its main source of fresh water which given the overall requirement to discharge water from the permanent water treatment plant should not pose a risk to a water deficit situation. The water balance will be managed closely and the operation has sufficient flexibility to minimize water related impacts.

- The treated water from the effluent treatment plant is required to meet certain water discharge criteria established for the Project which includes ammonia. The ammonia losses in the pit, due to the use of explosives, could exceed the expected levels and require an ammonia removal step in the water treatment plant. During the mining pre-production period, ammonia levels will be monitored closely.

25.2.3 Opportunities

There are a few remaining opportunities to improve overall Project economics and sustainability.

- Revenue-related potential opportunities:
  - The use of the Hardrock process plant and TMF for the future processing of gold from other GGM properties including the Hardrock underground resource and the regional exploration to improve the LOM average grade and/or extend the LOM. The Indigenous Agreements include agreed financial benefits for the Hardrock Underground.
  - The Project is permitted for 30,000 tpd providing the opportunity to increase throughput, post ramp-up through optimization. Additional throughput may be achieved through the milling circuit as a result of microcracks generated from the high-pressure grinding rolls; not accounted for in the throughput rate estimate. There is also a potential opportunity to use the Hardrock process plant and TMF to process some portion of the existing surface historic tailings in order to recover

gold, generate revenue, and also potentially mitigate environmental liabilities related to sulphides, arsenic and other contaminants.

- Connecting the natural gas power plant to the grid and selling spare power generation to the grid during times of shutdowns or excess capacity.
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- OPEX related potential opportunities:
  - Consider the possibility of a potential blend of LNG and diesel as a fuel source is possible for the mine haul trucks. Currently, the mine fleet uses 100% diesel.
  - Utilize new, commercially available technologies/autonomous haulage to increase operational effectiveness and reduce costs (Year 3+).
  - Optimize remote assisted drilling to its full potential. While the base case includes remote assisted drilling, additional benefits can be achieved via labour mine productivity improvements.
  - Investigate the availability of high-quality second-hand mining equipment.
- CAPEX related potential opportunities:
  - Pursue possible improvements in the TMF deposition plan and dam raise schedule, and the elimination of certain temporary structures in the initial construction in order to improve initial CAPEX.
  - Complete the ramp-up to commercial production one month earlier than planned.
  - Consider the possibility of equipment leasing to reduce upfront capital while protecting overall project economics.
  - Actively pursue government financial assistance for certain existing infrastructure relocations.

26. RECOMMENDATIONS

26.1 Hardrock Project Recommendations

After the completion of the 2016 Hardrock Project Feasibility Study, Greenstone Gold Mines ("GGM") successfully completed work plans in 2017 and 2018 to further de-risk the project. Federal approval of the Environmental Assessment was received in December 2018 and Provincial approval was received in March 2019. Permitting work has advanced, and after thorough consultation with the affected Indigenous communities and agencies, all pre-construction permit applications have been submitted. All permits required for construction were submitted. The forecast approval timelines support the planned construction schedule. GGM has now signed agreements with the local Indigenous communities, and implementation of these agreements is underway. The Independent Tailings Review Board ("ITRB") was implemented in 2017 to provide advice and guidance during the Tailings Management Facility ("TMF") detailed engineering phase and extensive TMF geotechnical programs have been completed. Conceptual construction execution plans were developed for the TMF, Goldfield Creek and Highway 11 and the construction schedule for these facilities has been de-risked. The 2018 and 2019 drilling programs were successful and provided the basis for improved resource and reserve estimates.

The scope of the Technical Report Feasibility Study Update included updating the resource and reserve models, revising the mine plan, advancing detailed engineering in higher risk areas and completing firm price bid processes for all major process and power plant equipment, effluent and sewage treatment plants and the mine mobile equipment fleet. This work formed the basis of the capital cost, operating cost and project economic update.

The Technical Report Feasibility Study Update has reconfirmed the technical feasibility of the Project and significantly improved the economic results.

The list of preliminary recommendations that follows was prepared by GGM and G Mining Services Inc. ("GMS") and reflects recommendations for subsequent phases of work, including completion of the detailed engineering phase, the construction phase and the operations phase. The cost of addressing each of these recommendations are generally within the scope of Project CAPEX, sustaining capital, closure costs and OPEX outlined in this Report.

26.1.1 Exploration and Geology

- Revalidate collar coordinates from the 2018 RC drilling campaign at Hardrock before the commencement of mining activities.

- Incorporate underground drill holes at Hardrock (not included in Mineral Resource) and adjust mineralization interpretation to produce a more locally accurate block model for internal purposes.
- Update the level of definition of the PAG (potentially acid generating) and non-PAG model.
- Undertake metallurgical test-work for the Kailey Deposit to confirm metallurgical recoveries assumed in the MRE.
- Retake core duplicates of existing Metalore-era drill core at Brookbank to confirm historical results where QA/QC protocols were lacking. Compile and digitise all QA/QC data for the Ontex-era drilling pre-2009 (present in drill logs and assay certificates).
- Resample drill core at Key Lake to increase the overall sample coverage and to overcome the effects of undersampling in the past.
- Undertake further resource definition drilling at Kailey, targeting the No.9 Zone near surface to convert existing Inferred to Indicated category, and to discover new ounces in the existing pit shell.
- Undertake a scoping study at Brookbank to understand if it could be a potential source of higher-grade plant feed for the overall Hardrock operation.

Table 26.1: Costs Associated with Exploration & Geology-Related Recommendations

Activity	Cost (CAD)
Metallurgical Test Work at Kailey (drilling, sampling, metallurgy)	300,000
Resampling of Drill Core at Brookbank and Key Lake	50,000
Expansion Drilling at Kailey	400,000
Scoping Study at Brookbank	150,000

26.1.2 Detailed Engineering Phase

- Review specific sections of waste rock storage designs C and D based on the latest geotechnical stability analysis produced by Wood in September 2019.
- Review specific sections of the overburden storage design based on the latest geotechnical stability analysis produced by Wood in August 2019.
- Conduct additional pit slope geotechnical work such as detailed review of variation in structural fabric orientation to identify possible localized sub-domains with stronger controls on achievable bench face angles; and conduct sensitivity analyses on slope saturation and lower effective shear strength. Additional laboratory testing such as triaxial testing and intact shear strength of foliation is recommended.

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