EQUINOX GOLD CORP.

Technical Report on the Greenstone Gold Mine
Geraldton, Ontario

QUALIFIED PERSONS:
Alexandre Dorval, P.Eng.
Réjean Sirois, P.Eng.
Kenneth Arthur Bocking, P.Eng.
Michelle Fraser, P.Geo.
Nicolas Vanier-Larrivée, P.Eng.
Pierre Roy, P.Eng.
Carl Michaud, P.Eng.
Darrol van Deventer, P.Eng.

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FORWARD-LOOKING INFORMATION

This Technical Report contains certain forward-looking information and forward-looking statements within the meaning of applicable securities legislation and may include future-oriented financial information (collectively, “Forward-looking Information”). Forward-looking Information in this Technical Report includes, but is not limited to, statements regarding: plans and expectations for the Greenstone Gold Mine, including estimated mine life, mining costs and production rates; estimates of Mineral Resources and the conversation of Mineral Resources to Mineral Reserves; projected metallurgical recoveries; and anticipated environmental liabilities. Forward-looking Information can be identified by the use of words such as “will,” “expect,” “achieve,” “strategy,” “increase,” “plan,” “potential,” “intend,” “anticipate,” “expect,” “estimate,” “target,” “objective” and similar expressions and phrases or statements that certain actions, events or results “may,” “could,” or “should” occur, or the negative connotation of such terms. The material factors or assumptions regarding Forward-looking Information contained in this Technical Report are discussed in this report, where applicable.

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Cautionary Note to U.S. Readers Concerning Estimates of Mineral Reserves and Mineral Resources

Disclosure regarding Equinox Gold's mineral properties, including with respect to Mineral Reserve and Mineral Resource estimates included in this Technical Report, was prepared in accordance with National Instrument 43-101—Standards of Disclosure for Mineral Projects (NI 43-101). NI 43-101 is a rule developed by the Canadian Securities Administrators that establishes standards for all public disclosure an issuer makes of scientific and technical information concerning mineral projects. NI 43-101 differs significantly from the disclosure requirements of the Securities and Exchange Commission (the “SEC”) generally applicable to U.S. companies. Accordingly, information contained in this Technical Report is not comparable to similar information made public by U.S. companies reporting pursuant to SEC disclosure requirements.

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Abbreviations, Acronyms, and Units of Measure

Abbreviations and Acronyms

AAS .... Atomic Absorption Spectrometry
ABA .... Acid-Base Accounting
AECO .... Alberta Energy Company
AERT .... Aboriginal Environmental Review Team
AFN .... Aroland First Nation
Ag .... Silver
AISC .... All-In Sustaining Cost
APV .... Aquatic Protection Value
ARD .... Acid Rock Drainage
As .... Arsenic
Au .... Gold
AZA .... Animbiigoo Zaagi'igan Anishinaabek First Nation

BGB	Beardmore-Geraldton Greenstone Belt
BIF	Banded-iron Formation
BNA	Bongwi Nevaashi Anishinaabek
BWI	Ball Mill Work Index
BZA	Biinjitiwaabik Zaaging Anishinaabek
C	Carbon
Ca	Calcium
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 Definitions Standards	*CIM Definition Standards for Mineral Resources and Mineral Reserves* dated May 10, 2014
CIM	Canadian Institute of Mining, Metallurgy & Petroleum
CIP	Carbon-in-Pulp
CN	Cyanide
COG	Cut-Off Grade
COPC	Constituent of Potential Concern
CQA	Construction Quality Assurance
CRM	Certified Reference Material
CSD	Critical Solid Density
Cu	Copper
CV	Coefficient of Variation
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
EIS/EA	Environmental Impact Statement and Environmental Assessment
EM	Electromagnetic
EMMPs	Environmental Management and Monitoring Plans
EMP	Environmental Management Plans
EOR	Engineer of Record
ETP	Effluent Treatment Plant

FA	Fire Assay
Fe	Iron
FEL	Front-End-Wheel Loaders
G&A	General & Administrative
GFN	Ginoogaming First Nation
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.
GPS	Global Positioning System
GRG	Gravity Recoverable Gold
HCl	Hydrochloric Acid Solution
HDPE	High-Density Polyethylene
HG	High Grade
HONI	Hydro One Networks Inc.
HPC	Hazard Potential Classification
HPGR	High-Pressure Grinding Rolls
HSE	Health, Safety, and Environmental
HVAC	Heating, Ventilation, and Air Conditioning
IAAC	Impact Assessment Agency of Canada
ICMI	International Cyanide Management Institute
ICPAES	Inductively Coupled Plasma Atomic Emission Spectroscopy
ICPMS	Inductively Coupled Plasma Mass Spectroscopy
ID^3	Inverse Distance Cube
IDF	Inflow Design Flood
IEC	International Electrotechnical Commission
IESO	Independent Electricity System Operator
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
KPIs	Key Performance Indicators
LAA	Local Assessment Areas
LEL	Lowest Effect Level
LG	Low Grade
LIMS	Low Intensity Magnet Separation
LLFN	Long Lake #58 First Nation
LNG	Liquid Natural Gas
LOM	Life-of-Mine

LTRAs	Long Term Relationship Agreements
MARC	Maintenance and Repair Contract
MCC	Motor Control Centers
MECP	Ministry of the Environment, Conservation and Parks
MG	Medium Grade
MHT	MacLeod High Tailings
Mine	Greenstone Mine
MINES	Ministry of Mines
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
MOECC	Ministry of the Environment and Climate Change
MOWL	Maximum Operating Water Level
MRE	Mineral Resource Estimate
MTO	Ministry of Transportation
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
O.Reg.	Ontario Regulation
OG	Original
OK	Ordinary Kriging Methodology
OPEX	Operating Expenditures
OPP	Ontario Provincial Police
OPP	Ontario Provincial Police
PAG	Potentially Acid Generating
Pb	Lead
PDA	Project Development Area
PEA	Preliminary Economic Assessment
PLC	Programmable Logic Controller
PMF	Probable Maximum Flood
POE	Power Over Ethernet
POX	Pressure Oxidation
Premier	Premier Gold Mines Limited
PV	Present Value

PWQO	Provincial Water Quality Objective
Q2	Second Quarter
Q3	Third Quarter
Q4	Fourth Quarter
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
ROY	Rest of Year
RPA	Roscoe Postle Associates Inc.
RQD	Rock Quality Designation
RSMIN	Red Sky Métis Independent Nation
RWI	Rod Mill Work Index
S	Sulphur
SAR	Species at Risk
SCC	Standard Council of Canada
SD	Standard Deviation
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
TBTE	TBT Engineering Limited
TCPL	TransCanada PipeLines Limited
TK	Traditional Knowledge
TLRU	Traditional Land and Resource Use
TMF	Tailings Management Facility
ToR	Terms of Reference
TRLU	Traditional Land and Resource Use
TS	Transmission Station
U CoG	Underground Cut-Off Grade
VC	Valued Components
VFD	Variable Frequency Drive
VLF-EM	Very Low Frequency-Electromagnetic
VSA	Vacuum Swing Adsorption
WHIMS	Wet High Intensity Magnetic Separation
WRSA	Waste Rock Storage Area (or Waste Rock Dump)
WSP	WSP Canada Inc.

XRF .... X-Ray Fluorescence

Units of Measure

% w/w .... percent weight by weight
°C .... degree Celsius
°F .... degree Fahrenheit
° .... degree
Ø .... diameter
µm .... micron (10^{-6} metre)
A .... Ampere
ft .... foot or feet
g .... gram
g/L .... grams per litre
g/t Au .... grams of gold per tonne
g/t .... grams per tonne
h .... hour
h/a .... hours per annum
h/d .... hours per day
ha .... hectares
hp .... horsepower
Hz .... hertz
in .... inch (imperial unit)
k .... thousand (kilo)
kg .... kilograms
kg/t .... kilograms per tonne
km .... kilometre
km/h .... kilometre per hour
koz .... thousands of troy ounces
kPa .... kilopascal
kt .... kilotonne, or thousands of tonnes
kV .... kilovolt
kW .... kilowatt
kWh .... kilowatt hour
kWh/t .... kilowatt hour per tonne
L .... litre
m .... metre
M .... million
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

masl	metres above sea level
mg	milligram
mg/L	milligram per litre
min	minute
mL	millilitre
mm	millimetre
Moz	million troy ounces
MPa	megapascal
Mt	million tonnes
MVA	Megavolt-ampere
MW	Megawatt
N	Newton
oz	Troy ounce (31.10348 grams)
P_{80}	80% of material passing grain size or sieve size material
ppb	parts per billion
ppm	parts per million
psi	Pounds per square inch
rpm	revolutions per minute
t	tonne (1,000 kg) (metric ton)
t/a	tonnes per annum
t/m^3	tonnes per cubic metre
V	volt

1 SUMMARY

Equinox Gold Corp. (Equinox Gold) has prepared this report titled *Technical Report on the Greenstone Gold Mine, Geraldton, Ontario* (the "Technical Report"), dated October 1, 2024, with an effective date of June 30, 2024.

1.1 Introduction

The Greenstone Gold Mine ("Mine" or "Greenstone Mine"), formerly known as the Hardrock Project, is currently undergoing commissioning and ramp-up to full production. Construction of the Mine began in the fourth quarter (Q4) of 2021, first ore was introduced into the grinding circuit in April 2024 and first gold was poured in May 2024. The Mine is operated by Greenstone Gold Mines (GGM). Equinox Gold also acquired the remaining 40% interest to consolidate 100% ownership of GGM and the Mine in May 2024.

The scope of this Technical Report includes updates on the geology and Mineral Resources of the Greenstone Mine and satellite deposits of Brookbank, Kailey, and Key Lake, as well as updates on the Mineral Reserves. The mining, infrastructure, and processing sections of this Technical Report refer only to the Mine. This Technical Report supersedes the previous Technical Report titled *NI 43-101 Technical Report, Hardrock Project, Ontario, Canada* prepared for Premier Gold Mines Limited (Premier) with an effective date of December 16, 2020, and a published date of January 26, 2021.

Various engineering consulting firms have contributed to this Technical Report and project update as follows:

- G Mining Services Inc. (GMS)—overall Technical 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 estimate (MRE); Mineral Reserves (pertaining to the Greenstone Mine only); mining methods; operating costs pertaining to mining; review of capital costs.
- Stantec Consulting Limited (Stantec)—climate and physiology and environmental; permitting, and closure unless otherwise noted; social aspects.
- Soutex Inc. (Soutex)—metallurgical testing; recovery methods; mineral processing operating cost; process plant and supporting infrastructure.
- WSP Global Inc. (WSP)—tailings management facility (TMF); Goldfield Creek diversion and geotechnical engineering for the open pit and waste rock storage areas; TMF closure plan; permitting of TMF-related facilities.

Unless otherwise stated, all the information and data contained in the Technical Report or used in its preparation have been provided by GGM, and all currencies are expressed in US dollars ($).

1.2 Property Description and Land Tenure

The Mine is approximately 275 km northeast of Thunder Bay, Ontario. The Mine includes three blocks of claims known as Hardrock, Brookbank, and Viper, which are spread over a distance of more than 100 km and are 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 Foxear targets.

The Mine consists of a contiguous block of patented claims, mining leases, licences of occupation, and cell claims covering 39,072.1 ha, of which 15,862.7 ha relates to Mine claims. All claims, leases, and licences of occupation are beneficially held by GGM and are subject to terms under several agreements.

1.3 Mineral Resource Estimates

1.3.1 Greenstone Mine

Since the previous Mineral Resource was released in 2021, substantial drilling has been conducted and was successful in de-risking the current Mineral Resource estimate (MRE) for the early years of production. Reverse circulation grade control drilling (RCGC) on a 20 m (X) by 10 m (Y) spacing was undertaken in 2018, 2019, 2021, 2022, targeting the first three benches of production, and partially tested an additional four benches in certain areas. In 2022, 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 2024 MRE is constrained by a pit optimization that extends deeper and incorporates more resources compared to the 2019 MRE.
- The reduction of internal dilution within the 22 principal domains has resulted in a 24% increase in average gold grade of assays within these domains and a higher overall gold grade in the Mineral Resource.
- Grade capping was revisited in 2024 (due to the refined wireframes) and new capping thresholds were chosen. They are generally in line with the capping chosen in 2019.
- RCGC drilling and validation diamond drilling conducted in 2018, 2019, and 2021 confirmed grade continuity and generally intersected similar to higher grades than expected in the 2019 block model.

The MRE was prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum *CIM Definition Standards for Mineral Resources & Reserves* (adopted May 19, 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 Definition Standards for Mineral Resources & Reserves*. GMS is not aware of any environmental, permitting, legal, title-related, taxation, socio-political, marketing, or other relevant issue that could materially affect the MRE.

The in-pit Mineral Resources of the Hardrock deposit are constrained within the design pit using a cut-off grade of 0.30 g/t Au. In addition to in-pit Mineral Resources, an underground MRE was estimated adjacent to and below the open pit using a 2.0 g/t Au cut-off grade. The open pit and underground Mineral Resources (exclusive of Mineral Reserves) are summarized in Table 1-1.

The MRE 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 3-D using Leapfrog GEO software based on a litho-structural model and the drill-hole database. The drill-hole database used in the estimate contained 462,540 sampled intervals from 738,232 m of diamond drilling in 1,846 holes, and 27,389 assay results from 30,183 m of RCGC drilling in 549 holes. Channel samples and blasthole 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 3-D block modelling and 3-pass Inverse Distance Cube (ID^3) interpolation with high-grade restraining.

Mineral Resources were classified as Measured in areas within 15 m of the RCGC drilling, and as 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 Mineral Resource estimated in Pass 1 to 3 in the external grade shell domain (500, 501, and 506) were downgraded to Inferred category.

Table 1-1: Mineral Resource Estimate (Exclusive of Mineral Reserves) for the Greenstone Mine

Category	In-Pit >0.3 g/t Au			Underground >2.0 g/t Au		
	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)
Measured	-	-	-	-	-	-
Indicated	19,008	1.21	738	10,959	4.20	1,480
M+I	19,008	1.21	738	10,959	4.20	1,480
Inferred	6,892	1.49	331	19,479	3.88	2,432

Notes:

- The Independent and Qualified Person for the MRE, as defined by NI 43-101, is Réjean Sirois, B.Sc., P.Eng., of GMS., and the Effective date of the estimate is June 30, 2024.
- These Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability.
- Mineral Resources are presented **exclusive** of Mineral Reserves.
- In-pit results are presented undiluted within a merged surface of the pit optimization shell at $1,700/oz Au and the 2024 pit design.
- In-pit Mineral Resources are stated at a cut-off grade of 0.30 g/t Au.
- Underground Mineral Resources are presented undiluted and are defined as blocks below and adjacent to the 2024 pit optimization at a cut-off grade of 2.00 g/t Au.
- Any discrepancies in the totals are due to rounding effects.
- 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 reference mining cost: $1.97/t, Incremental bench cost ($/10 m bench): $0.03, Milling cost: $6.98/t, Royalty: 3.0%, general and administration (G&A): $3.31/t, Sustaining capital: $0.92/t, Gold price: $1,700/oz, Milling recovery: 91.1% and Exchange rate 1.28 CAD/USD.

1.3.2 Other Deposits

The previous MREs for the regional Brookbank, Kailey, and Key Lake deposits were prepared by Mr. James Purchase, P.Geo., who was Director of Geology and Resources at GMS at the time, using Leapfrog EDGE (Version 5.1). The MRE is based on a drilling database for the three deposits provided to GMS on June 26, 2020. In 2024, the new QP validated the work done by Mr. Purchase and agreed with the modelling, chosen parameters, and estimation processes that subsequently led to the MRE stated in the previous Technical Report.

All Mineral Resources are effective as of June 30, 2024. There are no Mineral Reserves currently estimated for these regional deposits. Refer to Table 1-2.

Table 1-2: Summary of Brookbank, Key Lake, and Kailey Mineral Resources

Deposit	Mining Method	Category	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)
Brookbank	Open Pit	Indicated	1,147	2.24	83
		Inferred	45	2.07	3
	Underground	Indicated	2,281	7.06	517
		Inferred	706	3.38	77
Key Lake	Open Pit	Indicated	3,761	1.16	141
		Inferred	1,839	1.39	82
Kailey	Open Pit	Indicated	11,276	0.96	348
		Inferred	4,858	0.87	136

Notes:

- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- No Mineral Reserves are quoted for Brookbank, Kailey or Key Lake.
- The independent and qualified person for the 2024 Brookbank, Kailey and Key Lake MRE's is Mr. Rejean Sirois, B.Sc., P.Eng., Senior Technical Advisor, Geology & Resource of G Mining Services Inc., and the Effective date of the estimate is June 30, 2024.
- Open-pit Mineral Resources are constrained within a pit shell using a gold price of $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 $13.77/t is assumed for Brookbank, $1.31/t for Kailey and $3.47/t for Key Lake.
- Open Pit Mineral Resources are reported at a cut-off grade of 0.60 g/t Au for Brookbank, and 0.40 g/t Au for Kailey and Key Lake. Underground Mineral Resources are reported at a cut-off grade of 2.4 g/t Au for Brookbank.
- 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.
- 2019 CIM definitions were followed for Mineral Resource Estimates.

1.4 Mineral Reserves

Mineral Reserves for the Greenstone Mine's open pit is shown in Table 1-3.

Table 1-3: Greenstone Mine Open Pit Mineral Reserve Estimate

Category	Diluted Ore Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)
Proven	6,817	1.16	255
Probable	137,846	1.23	5,445
Total P&P	**144,662**	**1.23**	**5,700**

Notes:

- CIM definitions were followed for Mineral Reserves.
- Effective date of the estimate is June 30, 2024.
- Mineral Reserves are estimated at a cut-off grade of 0.30 g/t Au.
- Mineral Reserves are estimated using a long-term gold price of $1,550/oz and an exchange rate of 1.28 CAD/USD.
- A minimum mining width of 15 m was used.
- Bulk density of ore is variable but averages 2.78 t/m^3.
- The average life-of-mine (LOM) strip ratio is 5.5:1.
- Dilution factor is 17.2%.
- Numbers may not add due to rounding.

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 into Probable Mineral Reserves and Measured Mineral Resources into Proven Mineral Reserves. 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 factor and a 1.2% ore loss factor.

Golder (now WSP) carried out a feasibility-level pit-slope design study and the conclusions of this study were used as inputs to the pit optimization and design process.

1.5 Mining

Mining is being carried out using conventional open pit techniques with 10 m benches. An Owner-mined operation is in place, 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 is performed by blasthole drill rigs with both rotary and down-the-hole (DTH) drilling capability. Blastholes are loaded with bulk emulsion. The majority of the loading in the pit is carried out by two 29 m^3 hydraulic face shovels, one 15 m^3 hydraulic excavator, and one 30 m^3 front-end wheel loader. Haulage is performed with a combination of 224-tonne (Caterpillar 793-08) and 216-tonne (Caterpillar 793F) mine haul trucks. The presence of historical underground stopes was considered when designing the pit, mainly for the voids in the F Zone. Most of the other underground openings are backfilled with sand fill or rock fill.

Mining of the main pit will occur in five main phases. Waste rock will be disposed of in four waste dumps with three located around the pit and one further to the south. The open pit generates 788.6 Mt of overburden and waste rock (inclusive of historical tailings and underground backfill) over the LOM for an average LOM strip ratio of 5.5:1.

The LOM plan provides 15 years of mine production (from the third quarter [Q3] of 2024 to second quarter [Q2] of 2039) as shown in Figure 1-1. Annual mine material movement peaks at 72 Mt in 2025 and is maintained for 10 years until 2034. Material movement gradually declines from 2035 until the end of the mine life in 2039. The maximum processing plant production targets 27,000 t/d (9.86 Mt/a), which is achieved in 2025 and is sustained until 2038.

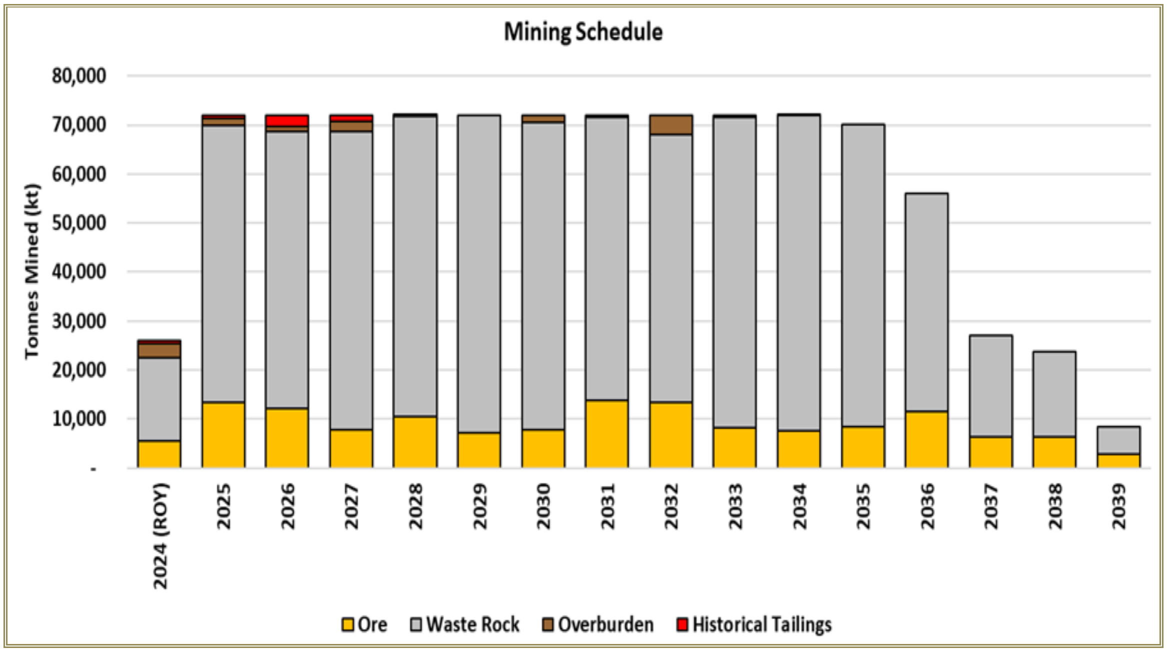


Figure 1-1: LOM Annual Tonnes Mined

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 best practices.

Prior to the start of the 2021 feasibility study, and 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.

During the preliminary economic assessment (PEA) and feasibility study, 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 was 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 grinding and ball milling, to reduce throughput risk and increase energy efficiency.

In the detailed engineering phase, additional leach testwork 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 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 389 koz for the first five years of production (commencing January 2025 to December 2029), with an average head grade of 1.36 g/t Au and an average metallurgical recovery of 90.8%. LOM production (commencing January 2025 to April 2039) averages 332 koz with an average head grade of 1.21 g/t Au and an average metallurgical recovery of 90.8%.

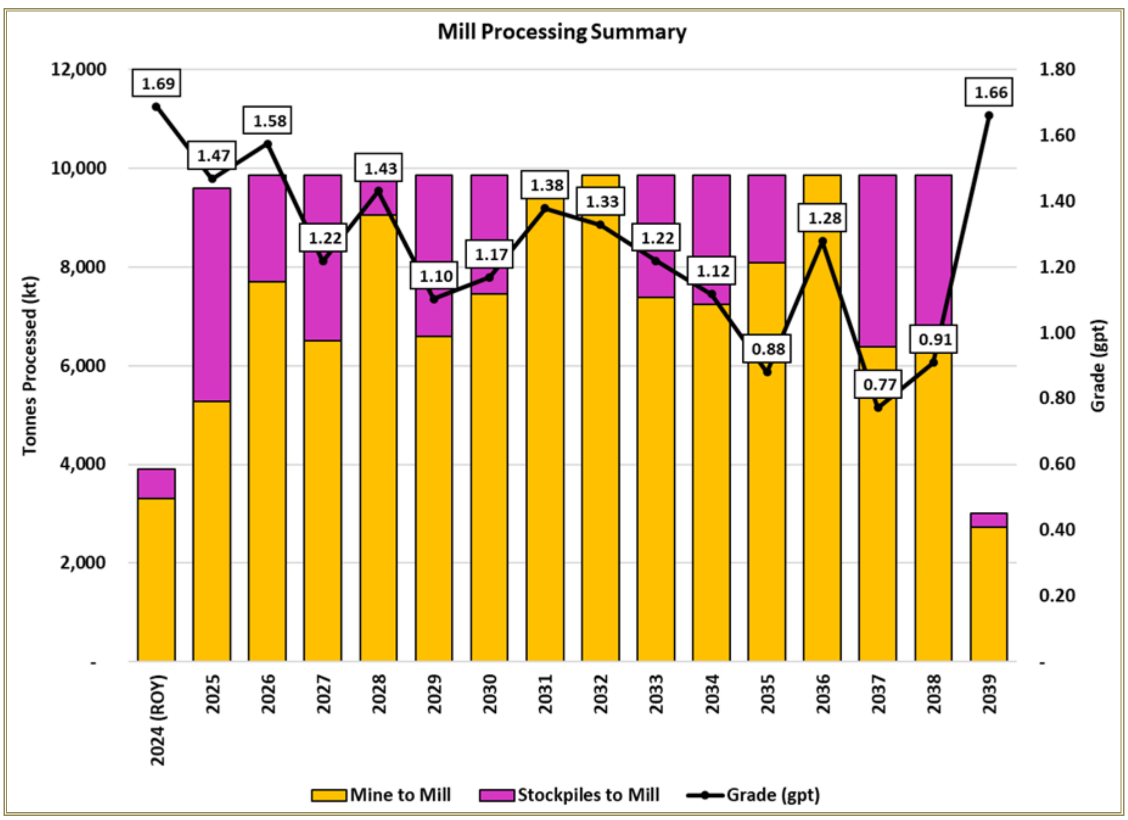


Figure 1-2: LOM Annual Ore Tonnes Processed and Average Gold Grades

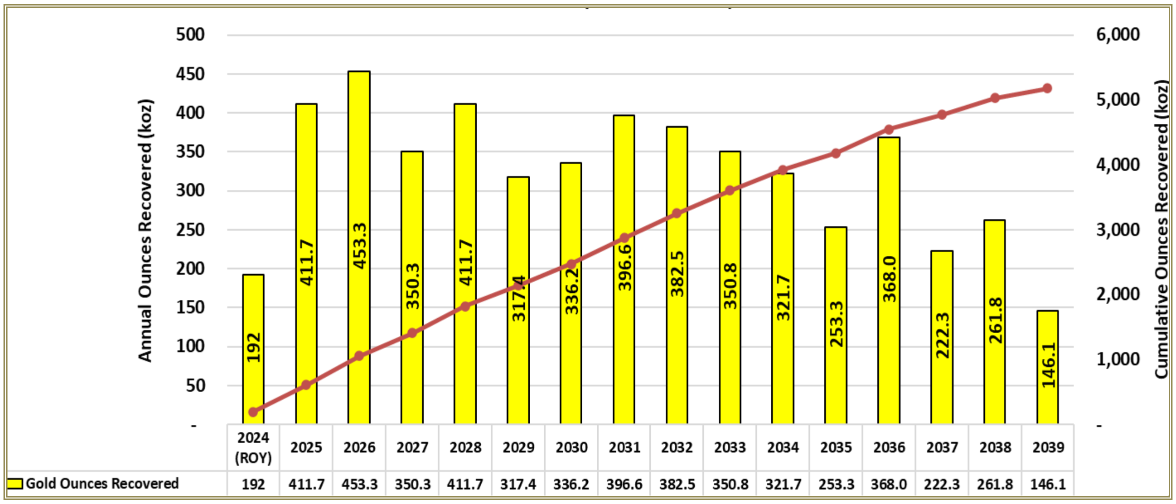


Figure 1-3: Annual LOM Gold Production

1.7 Mine Infrastructure and Services

The Mine is within a district that is host to numerous mines and processing facilities and has access to good transportation and regional mining-related infrastructure. The Mine is near the Trans-Canada Highway 11, TransCanada PipeLines Limited Canadian Mainline (TCPL Mainline) natural gas pipeline, a Hydro One electrical substation, and the town of Geraldton hosts a municipal airport, which has a 1,500 m runway capable of accommodating small charter aircraft. Geraldton has its own potable water treatment system and water distribution network.

The general infrastructure to support mining and processing activities includes:

- 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.

A length of Trans-Canada Highway 11, a Hydro One 115 kV station, and a Ministry of Transportation Ontario (MTO) patrol station were relocated to allow development of the Mine. Existing infrastructure within the footprint of the property limits that will need to be relocated in the future includes:

- Ontario Provincial Police (OPP) station
- Historical MacLeod and Hardrock tailings (portions covering the open pit mine).

Portions of a golf course and the MacLeod–Cockshutt (MacLeod–Mosher) mine headframe were purchased from the municipality. Private properties in the MacLeod townsite and Hardrock townsite (65 in total) and a gas station were also purchased.

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

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 be relocated in Years 6 to 9 of operations. Relocation of the historical MacLeod tailings commenced in January 2024 and is ongoing, with all tailings transported and deposited into the TMF.

1.8 Water Management

Two types of effluents will be generated during Mine activities: mine effluent and sanitary effluent. The water quality standards applicable to mine effluent are defined in the applicable Environmental Compliance Approvals (ECA) and Federal Metal and Diamond Mining Effluent Regulations (MDMER) Effluent Criteria. The ECAs identify discharge locations and 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 MDMER criteria at the end of the pipe and the Provincial Water Quality Objectives (PWQO) (MOE, 1994) for 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, with excess water sent to the effluent water treatment plant for treatment prior to discharge to the Southwest Arm of Kenogamisis Lake. A seepage collection system was installed to manage seepage from the historical Macleod tailings. Surface water runoff from the exterior of the TMF dams and any seepage through the dams or foundations is collected in a series of ponds and pumped back into the TMF reservoir for reuse in processing.

1.9 Tailings Management Facility

The TMF is a series of constructed dams with a final maximum height of 35 m and crest length of approximately 7,400 m. The TMF is currently designed to receive approximately 145 million tonnes (Mt) of mill tailings at an average dry density of 1.34 t/m^3. A cyanide destruction system is used to process all tailings water before it is sent to the TMF. An allowance has been made within the TMF to store the historical tailings and contaminated soils being relocated from the open pit area.

The TMF dams are and will continue to be constructed primarily using waste rock from mining operations. The dams will be constructed in stages and in the downstream direction. Construction of the TMF starter dams was completed in 2023. The first (Stage 1) dam raise will be completed in 2024 to a crest elevation of 344 m, and the planned ultimate crest elevation will be 365 m.

Tailings geochemistry indicates that less than 10% of the ore is considered potentially acid generating (PAG). This amount will be reduced through oxidization during ore processing, thereby reducing the overall acid rock drainage (ARD) potential for the tailings.

Tailings are deposited in the TMF from the dam crests as a conventional slurry to produce a wide exposed beach. This beach will displace the tailings pond away from the dams towards natural ground along the western edge of the facility to enhance long-term dam stability. A barge-mounted pump system, located near the north side of the TMF, reclaims water from the TMF pond and pumps it back to the processing plant.

Closure of the TMF involves lowering of the spillway and vegetating the exposed tailings beaches. Runoff from the pond, when deemed suitable for discharge to the environment, will be directed through the spillway.

1.10 Environmental Studies

Environmental baseline studies were conducted for the Mine between 2013 and 2021 and were used to identify environmental constraints during the development of layouts and designs for the Mine. This environmental baseline was the basis for determining incremental changes and predicting environmental effects associated with the Mine.

A final environmental impact statement/environmental assessment (EIS/EA) was completed and approved by provincial and federal regulatory agencies. Project interactions were analyzed for 13 valued components (VC) to determine potential environmental effects associated with the Mine for construction, operation, and closure phases. In addition to the VCs, the effects assessment also considered effects of the environment on the Mine, accidents and malfunction scenarios, and cumulative effects. Environmental management and monitoring plans (EMMP) were developed and implemented and include measures related to both compliance and EIS/EA monitoring for all phases of the Mine.

GGM submitted a Closure Plan and Financial Assurance to the Ministry of Mines, it received approval on March 30, 2021. Since approval of the initial Closure Plan, GGM has filed two amendments, one in December 2023 and another in 2024 to account for detailed design, and to address mitigation measures to address the erosion of the Goldfield Creek diversion channel. At the end of mining operations, the main features requiring closure will include the 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 Mine can proceed to final close out status.

The results of the final EIS/EA, including implementing the identified mitigation measures, supports the conclusion that the Mine will not cause significant adverse environmental effects. Since completing the final EIS/EA, GGM has completed slight modifications of Mine components, which form the basis for the final mine plan used for this Technical Report. Active consultation with stakeholders (community members, agencies, and interested parties) and Indigenous communities has been undertaken throughout Mine planning and will continue as the Mine progresses through permitting and detailed engineering.

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

1.11 Operations Organization

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 715 employees (reached in Year 4).

1.12 Capital Cost Estimate

The non-sustaining capital cost is shown in Table 1-4, and is estimated to be $318.4 million for the LOM operating period.

Table 1-4: Non-Sustaining Capital Cost LOM Summary

Capital Cost—Non-Sustaining	Total Cost ($M)
Building and Infrastructure	3.5
Machinery and Equipment	5.8
Non-Sustaining—Leased Equipment	232.2
Project Carryover	51.0
Capitalized Development	25.9
Total Cost	**318.4**

Major items included in the non-sustaining capital include the relocation cost of the Ontario Provincial Police station, the payment for the off-site laboratory purchase (located in Geraldton), the purchase of the seventh gas-powered generator, all the lease payments for the mining fleet, the cost of the MacLeod Township demolition, and the rehabilitation work for the Goldfield Creek diversion.

The sustaining capital cost is shown in Table 1-4, and is estimated to be $608.8 million for the LOM operating period.

Table 1-5: Sustaining Capital Cost LOM Summary

Capital Cost—Sustaining	Total Cost ($M)
Buildings, Infrastructure and Hardware/Software	47.6
Machinery and Equipment	45.1
Major Capital Repairs	313.0
Tailings Management Facility Expansions	138.2
Mining Fleet Equipment Purchase	64.9
Total Cost	**608.8**

Major items included in the sustaining capital include major capital repairs for the mining fleet, TMF expansions, new mining fleet equipment purchases, a new camp accommodation area, and strategic spares for the processing plant.

In addition, a portion of the major waste-stripping costs for the open pit is capitalized. If the waste stripping volume in any quarterly period is greater than the waste-stripping level of the overall average LOM stripping ratio, then it is considered as capitalized stripping. The total LOM capital stripping is $338 million.

1.13 Operating Cost Estimate

Operating costs are summarized in Table 1-6. The operating costs include mining, processing, and G&A. The average operating cost is $845/oz Au or $30.22/t milled over the LOM operating period.

Table 1-6: Operating Costs Summary

Category	Total Costs ($M)	Unit Cost ($/t milled)	Cost per oz ($/oz)
Mining	2,512	17.36	485
Processing	1,076	7.44	208
G&A	784	5.42	152
Total Operating Costs	**4,372**	**30.22**	**845**

Note: G&A = general and administrative.

The average mining cost during operations is estimated at $2.70/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.

1.14 Interpretation and Conclusions

1.14.1 Geology and Mineral Resources

- GMS carried out extensive work, including site visits, in validating the data used to generate the 2024 MRE (detailed in Section 12) and finds it suitable to support the MRE.
- In 2024, the QP (Mr. Réjean Sirois) validated the work done by Mr. Purchase and agreed with the modelling, chosen parameters, and estimation processes that subsequently led to the current MRE (detailed in Section 14).
- Mineral Resources were classified using logic consistent with the CIM definitions referred to in NI 43-101.
- The QP considers the 2024 MRE fit for use for mine planning purposes.

1.14.2 Mining and Mineral Reserves

- Modifying factors were applied to convert Mineral Resources to Mineral Reserves, including cut-off grades, mining dilution, and mining recovery factors. Only Measured and Indicated Mineral Resources are used to state Mineral Reserves.
- The estimated Mineral Reserves used a gold price of $1,550/oz, and an effective date of June 30, 2024.
- At a cut-off grade of 0.30 g/t Au, the Proven Mineral Reserves total 6.8 Mt at an average grade of 1.16 g/t Au, for 255 koz of contained gold. The Probable Mineral Reserves total 137.8 Mt at an average grade of 1.23 g/t Au, for 5,445 koz of contained gold. The total Proven and Probable Mineral Reserves is 144.7 Mt at an average grade of 1.23 g/t Au, for 5,700 koz of contained gold.
- The mining activities will occur over a period of 15 years (from start of commercial production to the end of in-pit mining) and excluding the pre-production period.

- The open pit will generate 788.6 Mt of overburden and waste rock (inclusive of historical tailings and underground backfill), for a strip ratio of 5.5:1.

1.14.3 Metallurgical Testing and Mineral Processing

- The process design criteria have been established based on test results, Owner and Vendor recommendations, and industry practices.
- Processing options were selected based on the results of this testwork and are well-known technologies currently used in the mining industry.
- The gold recovery process consists of a crushing circuit; an HPGR/ball mill grinding circuit; pre-leach thickening and cyanide leaching; a 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 90.8 and is based upon metallurgical testing completed comprising 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 on the block attributes and the target grind size.

1.14.4 Infrastructure

- Existing infrastructure within the footprint of the property limits was relocated or purchased and dismantled, except for the OPP station which is yet to be relocated. The most significant relocation was that of the TransCanada Highway 11 which was achieved in August 2023. All private properties within the Mine footprint have been purchased.
- A 65 MW natural gas-fired power plant with a designed capacity of 46.5 MW was constructed and commissioned.
- As with the other main infrastructure, the administration building, truck shop, reagent storage, explosives plant, and tailings management facility have been sized to support the mine and process operation.
- Goldfield Creek, which traversed the TMF footprint, was permanently diverted northeast to Kenogamisis Lake in November 2022. The Goldfield Creek diversion channel design meets the fish offset guidelines. The GFC diversion dike required for the diversion was designed and constructed 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 rockfill dams and the water pond is pushed to the west to abut on natural ground.
- Surface water runoff from TMF dams and any seepage through the dams or foundation are collected in ponds and pumped back to the TMF reservoir.

1.14.5 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 Mine will not cause significant adverse environmental effects, including effects from accidents and malfunctions, effects of the environment on the Mine, and cumulative effects.

- EMMP were developed and implemented and include measures related to both compliance and EIS/EA monitoring for all phases of the Mine. The collective monitoring activities associated with the Mine will also be used to inform adaptive management for the Mine, as required. The management and monitoring requirements have been incorporated into Mine plans and budgets.
- Permits required for construction and early operation of the Mine have been obtained and conditions of the permits have been implemented, as appropriate.
- Mitigation measures for the Goldfield Creek diversion channel have been implemented to de-risk potential additional erosion of the channel as redesign and rehabilitation of the Goldfield Creek diversion channel continues.
- Active consultation with stakeholders (community members, agencies, and interested parties) and Indigenous communities has been undertaken throughout Mine planning and construction and will continue as the Mine progresses.
- GGM has established LTRAs with five local Indigenous communities. The agreements establish increased clarity regarding GGM's ability to develop the Mine, and the Indigenous communities' opportunities to benefit from future mining opportunities in the region, including the potential to extend the life of the Project.

1.14.6 Capital and Operating Costs

- The non-sustaining capital cost is estimated to be $318.4 million for the LOM operating period.
- The sustaining capital cost is estimated to be $608.8 million for the LOM operating period.
- The average operating cost is $845/oz Au or $30.22/t milled over the LOM operating period. The operating costs include mining, processing, and G&A.
- The average mining cost during operations is estimated at $2.70/t mined including re-handling costs.

1.15 Risks and Opportunities

1.15.1 Risks

The following is a discussion of the key risks for the Mine with summaries of the related controls and risk mitigation strategies.

Gold Production

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. During operations, ongoing optimization of the metallurgical performance will be carried out via leach testwork, and throughput vs. grind-size trade-offs will be evaluated on a regular basis in conjunction with anticipated gains from the HPGR circuit due to microcracking.

Permitting

With ongoing constraints in the public sector, GGM is monitoring the risk of agencies not meeting a reasonable timeframe for any ongoing or future permitting approvals. To facilitate the approval timeframes, consultation with Indigenous communities and agencies is undertaken on key permit applications prior to submission.

The permit that governs the annual mined quantities is in the process of being increased from 70 Mt/a to 72 Mt/a.

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 fieldwork.

A detailed Tailings Facility Construction Management Plan, including a QA/QC program, has been implemented for construction for current and future expansions of the TMF. A tailings deposition plan and a dam-raising schedule have been developed to ensure capacity for the mill tailings during operations. An Operations Management and Surveillance (OMS) Manual following the guidelines of the Mining Association of Canada has been put in place for the TMF.

An Independent Tailings Review Board (ITRB) was established to provide oversight during the lifecycle of the TMF and is an ongoing process. 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 design of the TMF.

Pit Wall Stability

A comprehensive pit-slope management program is put in place by the Mine's geotechnical engineering department, to manage risks attributable to potential movement of the exposed rock faces. Rock mass failure is considered a low risk 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.

Stability of Historical Tailings

Attention to mining practices when mining proximal to the historical tailings have been implemented, especially focusing on controlling vibrations attributable to blasting activities. Emphasis is placed on minimizing exposure of excavated tailings slopes. Rockfill will continue to be installed on the slopes following the advancement of the excavation.

Water Management

The Mine is bordered on three sides by lakes and is cross-cut by small streams. There are several risks associated with the use, treatment, and discharge of water during operations and closure. These risks and associated treatment plans are as follows:

- The risk of unacceptable contaminants such as arsenic seeping from the TMF, historical tailings, and waste rock storage areas have been mitigated by seepage collection ditches and collection ponds that allow for water to be collected and recycled to the process plant during operations to ensure the required water quality objectives are met.

- The risk of water ingress into the open pit are manageable based on historically low dewatering rates from the low permeability of the host rock. Pit dewatering will be maintained a minimum 20 m below the lowest mining bench elevation.
- The process plant relies on water collected from the historical underground workings, the open pit, and 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 of a water deficit.
- The treated water from the effluent treatment plant is required to meet certain water-discharge criteria established for the Mine, which includes ammonia. An ammonia treatment option has been designed and permitted for the water treatment plant, and can be implemented if ammonia concentrations within the pit (from explosives use) reach the given trigger threshold as defined in the permit.

1.15.2 Opportunities

Several potential opportunities exist to further improve the overall economics and sustainability of the Mine.

Revenue-Related Potential Opportunities

- Use of the Mine's process plant and TMF for ore processing from other GGM properties including the Greenstone underground resource and the regional exploration projects.
- The Mine is permitted for 30,000 t/d, providing an opportunity to increase throughput.
- Connecting the natural gas power plant to the provincial electrical grid to either sell spare energy during shutdowns when excess generating capacity is available or to provide electrical stability to the grid.
- Study the potential to economically remove magnetite from the tailings and produce a concentrate product for sale.
- Study the potential to economically reprocess historical tailings from the Mine's property and other nearby GGM properties.
- From the Mine, open-pit expansion to the west and incorporating the underground deposit to the regional scale, there is significant potential for resource growth and discovery.

OPEX-Related Potential Opportunities

- The potential to blend liquid natural gas and diesel as a fuel source for the mine haul trucks. Currently, the mine fleet only uses diesel.
- The use of new, commercially available technologies (i.e., autonomous haulage) to increase operational effectiveness and reduce costs.
- Optimizing the existing remote-assisted drilling to achieve additional labour productivity improvements.

1.16 Recommendations

1.16.1 Mine Geology Recommendations

- Current block models should be updated using the RCGC drilling information performed since March 23, 2022, the cut-off date for the database used for the September 2022 MREs.
- A detailed study should be undertaken to fully understand the impact of including blasthole sampling assays on the "ore control" polygons. Blasthole sampling assays do not have the same quality as DDH

and RCGC, and their usefulness for grade control within the main wireframes should be fully evaluated. Following this, the Mine's technical services team may consider using DDH and RCGC assay results for the design of the ore-control polygons or only using blasthole assays to find additional mineralization outside the main wireframes within the external grade shells.

- Detailed lithology and structural mapping of the pit walls should be completed regularly, and this information used to update and further refine the litho-structural model.
- A desktop study for the potential underground extension below the 2024 pit design should be conducted and should include sensitivity analysis to cut-off grades and mining methods.
- The current (and more conservative) Mineral Resource classification criteria should be reassessed with knowledge gained from further drilling and reconciliation. A 15,000-m diamond drilling program using oriented core is proposed for an all-inclusive cost of $3.0 million to reduce risk associated with the mineralization contained within the external grade-shells.
- Additional drilling should be carried out to convert Inferred Mineral Resources (exclusive of Mineral Reserves) to a higher confidence category. A program of 8,000 m of diamond drilling is proposed for an all-inclusive cost of $1.8 million.
- Continue with establishing and refining reconciliation practices and procedures to evaluate the MRE and operational effectiveness on monthly, quarterly and annual bases.

1.16.2 Exploration Recommendations

- Undertake further Mineral Resource definition drilling at Kailey, targeting the No. 9 Zone near the surface to convert existing Inferred to Indicated Mineral Resources and to discover new gold-bearing zones in the existing pit shell.
- Undertake metallurgical testwork 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 digitize all QA/QC data from the Ontex-era drilling pre-2009 (present in drill logs and assay certificates).
- Selective sampling of gold-bearing zones was completed on much of the historical Brookbank drill core. Cut and sample wider, continuous intervals consistently along strike to confirm areas of barren rock adjacent to the main ore zones and perhaps identify new mineralized zones.
- Resample the drill core at Key Lake to increase the overall sample coverage and overcome the effects of previous under-sampling.

1.16.3 Engineering Recommendations

- 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. Conduct additional laboratory testing (i.e. triaxial testing) and intact shear strength of foliation.

2 INTRODUCTION

The Greenstone Gold Mine (Greenstone Mine or Mine), formerly known as the Hardrock Project, is currently undergoing commissioning and ramp-up to full production. Construction of the Mine began in the fourth quarter (Q4) of 2021, first ore was introduced into the grinding circuit in April 2024 and first gold was poured in May 2024. Equinox Gold also acquired the remaining 40% interest to consolidate 100% ownership of the Mine in May 2024.

The scope of this report, titled *Technical Report on the Greenstone Gold Mine, Geraldton, Ontario* (this "Technical Report") with an effective date of June 30, 2024, includes updates in the geology and Mineral Resources of the Greenstone Mine and the Brookbank, Kailey, and Key Lake satellite deposits. Conversely, updates on the Mineral Reserves, mining, infrastructure, and processing sections of this Technical Report refer only to the Greenstone Mine. This Technical Report supersedes the previous Technical Report titled *NI 43-101 Technical Report, Hardrock Project, Ontario, Canada* with an effective date of December 16, 2020, and a published date of January 26, 2021.

Various engineering consulting firms have contributed to this Technical Report and project update; their responsibilities are as follows:

- G Mining Services Inc. (GMS)—overall Technical 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 estimate (MRE); Mineral Reserve estimates (pertaining to the Greenstone Mine only); mining methods; economic analysis; operating costs pertaining to mining; review of capital costs.
- Stantec Consulting Limited (Stantec)—climate and physiology; environmental; permitting, and closure unless otherwise noted; social aspects.
- Soutex Inc. (Soutex)—metallurgical testing; recovery methods; mineral processing operating cost; process plant and supporting infrastructure.
- WSP Global Inc. (WSP)—tailings management facility (TMF); Goldfield Creek diversion and geotechnical engineering for the open pit and waste rock storage areas; TMF Closure Plan; permitting of TMF-related facilities.

Table 2-1 gives a detailed summary of the qualified persons (QP) responsible for each section of this Technical Report.

Table 2-1: Summary of Qualified Persons

QP	Company	Sections
Alexandre Dorval, P.Eng.	GMS	1.4, 1.5, 1.14.2, 1.16, 15, 16, 25.1.2
Réjean Sirois, P.Eng.	GMS	1.3, 1.14.1, 1.16, 4–12, 14, 23, 25.1.1, 26.1, 26.2
Kenneth Arthur Bocking, P.Eng.	WSP	1.9, 18.1, 18.2, 18.3, 26.3, parts of 1.14.4 and 25.1.4
Michelle Fraser, P.Geo.	Stantec	1.8, 1.10, 1.14.5, 5, 20, 25.1.5
Nicolas Vanier-Larrivée, P.Eng.	GMS	1.7, 1, 18.4,18.5, 18.6, parts of 1.14.4 and 25.1.4
Pierre Roy, P.Eng.	Soutex	1.6, 1.14.3, 13, 17, 25.1.3
Carl Michaud, P.Eng.	GMS	1.12, 1.13, 1.14.6, 21, 25.1.6
Darrol van Deventer, P.Eng.	Equinox Gold	1.1, 1.2, 1.12, 1.13, 1.14.6, 2, 3, 19, 22, 24, 25.2, 26.3

2.1 Site Visits

The QPs visited the Mine site on the following dates:

- Alexandre Dorval, P.Eng., GMS, July 24, 2024
- Réjean Sirois, P.Eng., GMS, visited the site on numerous occasions since August 2016. The last visit was between July 22 to 25, 2024
- Kenneth Arthur Bocking, P.Eng., WSP, September 3 to 5, 2024
- Michelle Fraser, P.Geo., Stantec, July 21 and 22, 2024
- Nicolas Vanier-Larrivée, P.Eng., GMS, July 24, 2024
- Pierre Roy, P.Eng., Soutex, May 31 to June 7, 2024
- Darrol van Deventer, P.Eng., Equinox, July 22 to 25, 2024.

2.2 Sources of Information and Data

Unless otherwise stated, all the information and data contained in the Technical Report or used in its preparation have been provided by GGM, and all currencies are expressed in US dollars ($).

2.3 Abbreviations, Acronyms, and Units of Measure

Unless otherwise noted, the Technical Report uses the International System of Units (metric system). A list of the main symbols, units of measure, abbreviations, acronyms, and initialisms used in this Technical Report are presented below the table of contents.

3 RELIANCE ON OTHER EXPERTS

Unless otherwise stated, all the information and data contained in this Technical Report or used in its preparation have been provided by Greenstone Gold Mines GP Inc. (GGM).

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

- SGS Minerals Services (SGS), ThyssenKrupp, and SimSAGe provided metallurgical reporting and studies as referenced in used in Section 13, managed principally by GGM.
- Golder Associates Ltd. (Golder), now part of WSP Global Inc. (WSP), provided rock mechanics and open pit geotechnical studies used in Section 16.
- SGS provided laboratory geochemical and mineralogical testing, managed principally by GGM.
- Golder relied on the oriented core data MD Engineering (MDE) collected for evaluating the open pit geotechnical parameters and pit-slope studies. Golder also validated the MDE methodology and ~5% of the total oriented core. Golder believes that the remainder of the core data were also collected in a professional manner.

4 PROPERTY DESCRIPTION AND LOCATION

This section is an update from the 2021 Report, issued January 26, 2021. Some comments have been added to cover available information through June 2024.

The Greenstone Gold Property includes three blocks of claims known as the Hardrock, Brookbank, and Viper areas. The Greenstone Mine is in the southeast portion of the Hardrock claims block.

4.1 Location and Access

The Mine is approximately 275 km northeast of Thunder Bay, Ontario, and approximately 4 km south of the Town of Geraldton, Ontario (Figure 4-1), in Ontario's Thunder Bay Mining Division; locations of all claims fall within National Topographic System (NTS) Sheets 42 E/10 and 42 E/11. Thunder Bay has a population of around 110,000, and provides support services, equipment, and skilled labour for mineral exploration and the mining industry. Rail, national highway, port, and international airport services are also available in Thunder Bay. Geraldton has a population of approximately 1,900 and provides support services such as food and lodging.

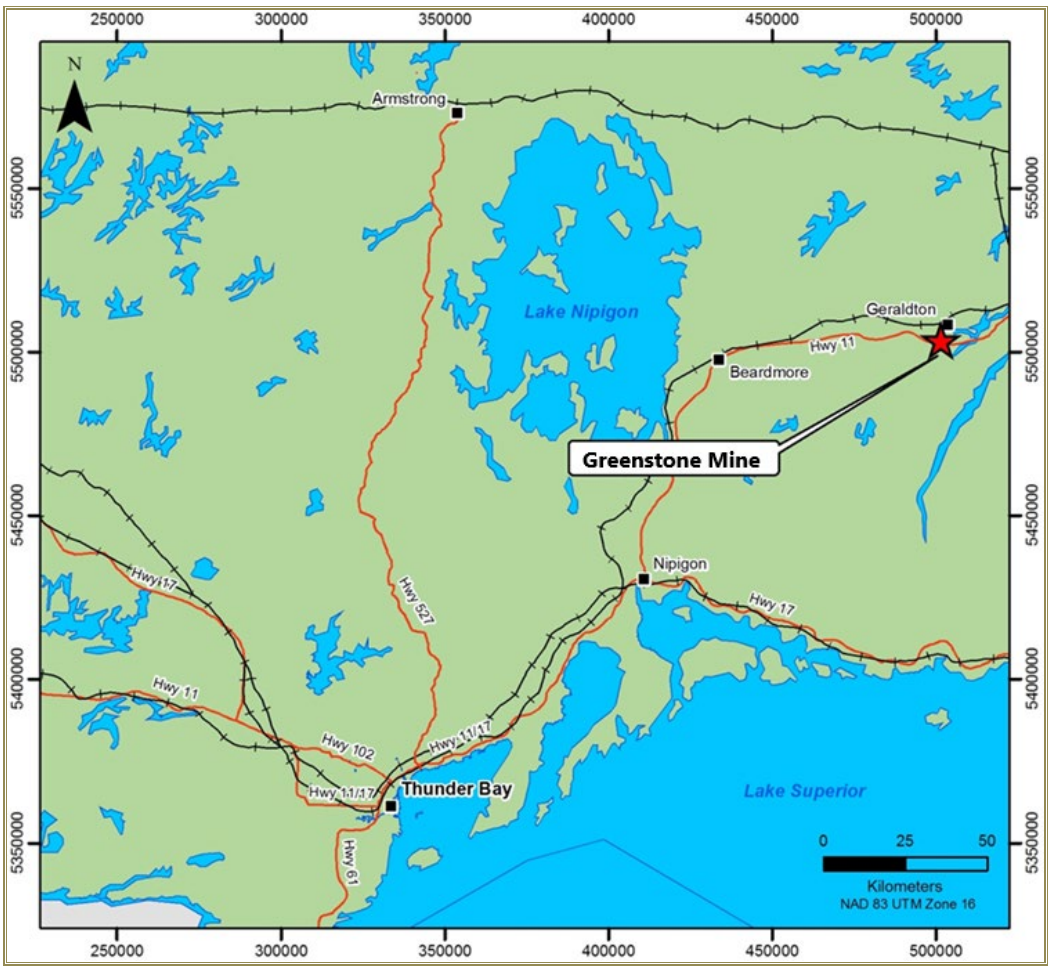


Figure 4-1: Location of the Greenstone Mine

The Mine area covered by the MRE in this Technical Report is in the townships of Errington and Ashmore on NTS Sheet 42E/10, approximately 4 km south of Geraldton. The approximate coordinates of the Mine's resource areas geographic centre are 49°40'47"N and 86°56'32"W (UTM Zone 16N coordinates: 504175.9E and 5503024N; NAD 83).

4.2 Property Description

As of June 2024, GGM's property holdings consisted of three blocks of contiguous mining claims known as the Hardrock, Brookbank, and Viper areas (Figure 4-2). The Mine 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 39,072.1 ha, summarized in Table 4-1. The properties are in the townships of Lindsley, Errington, Ashmore, Parent, Salsberg, and McKelvie in the Thunder Bay Mining Division. A leasehold patent of mining rights, surface rights, or 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. A detailed listing of all the claims pertaining to GGM was received in July 2024, and the current Mineral Resource QP testified that the claims are all in good standing with rent paid up to June 30, 2024.

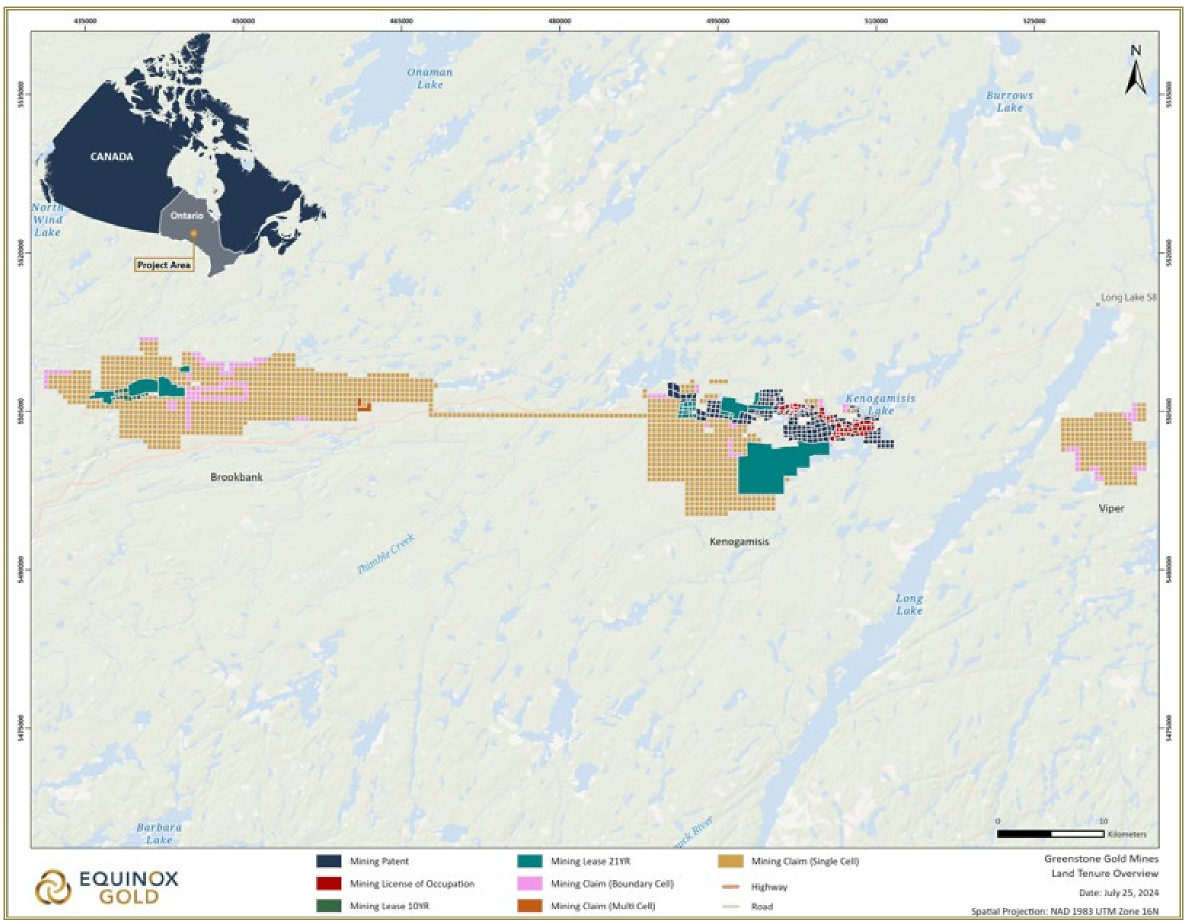


Figure 4-2: Overview of GGM's Land Tenure

Table 4-1: Summary of Types of Land Tenure in GGM Land Package—as of June 30, 2024

Property	No. of Cell Claims	No. of Patents	No. of Leases	No. of MLOs	Area (ha)
Hardrock (or Kenogamisis)	482	191	24	78	16,203
Brookbank	938	0	19	0	18,481
Brookbank-Kenogamisis Bridge	48	0	0	0	1,003
Viper	216	0	0	0	4,246
Total	**1,684**	**191**	**43**	**78**	**39,933**

Several past-producing underground gold mines are on the property, including Hard Rock, MacLeod–Cockshutt, and Mosher (all later combined as the Consolidated Mosher); Little Long Lac; Bankfield; Jellicoe; and Magnet. There are also some less-significant historical occurrences of gold mineralization within the property boundary. The mineralized zone that hosts the most recently delineated Mineral Resources is within or adjacent to the former Hardrock and MacLeod–Cockshutt mines (Figure 4-3).

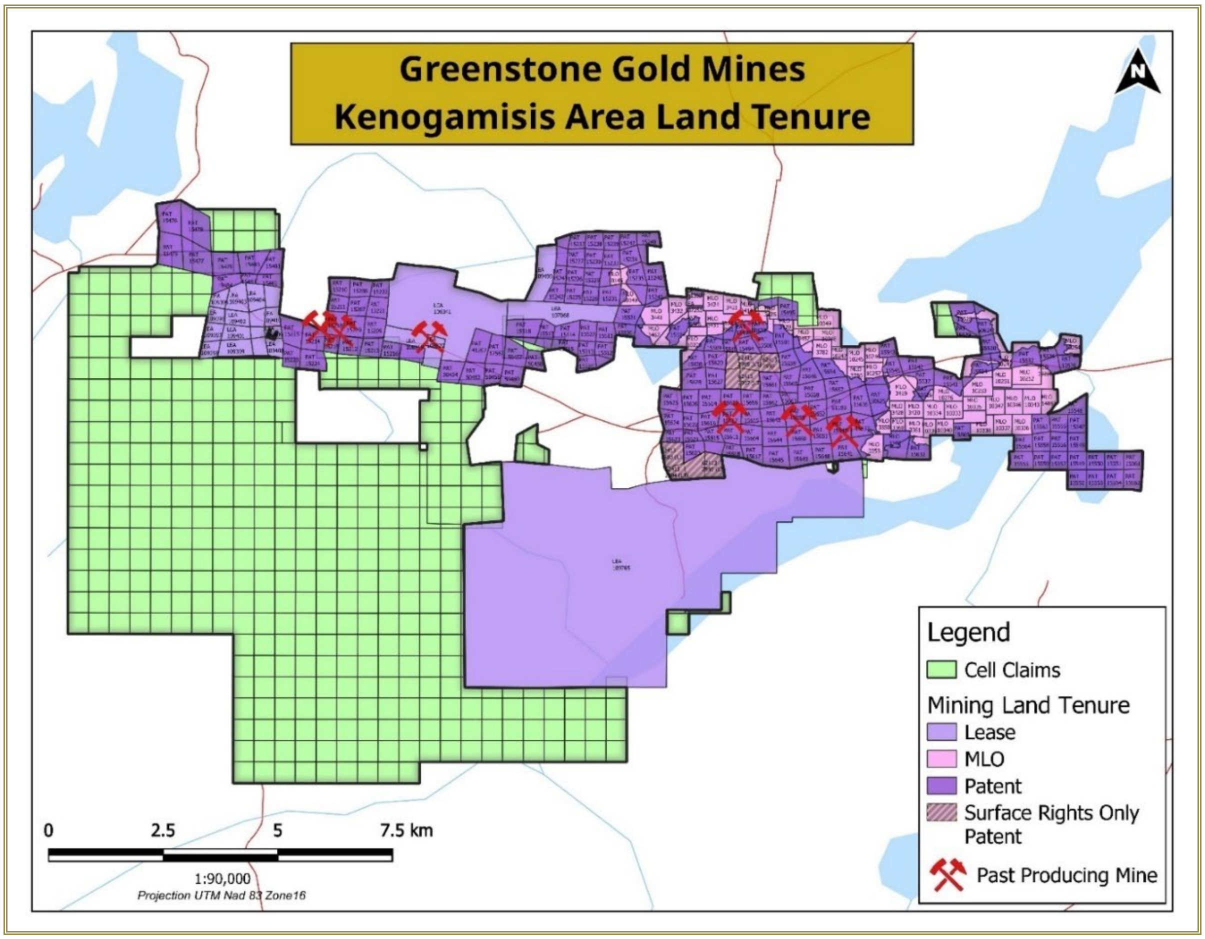


Figure 4-3: Greenstone Mine Properties—Past Producing Mines

4.3 Greenstone Mine 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 totalling 15,862.7 ha (Figure 4-4). 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 Greenstone Mine area. The lease, LEA-109765, is subject to renewal in 2039.

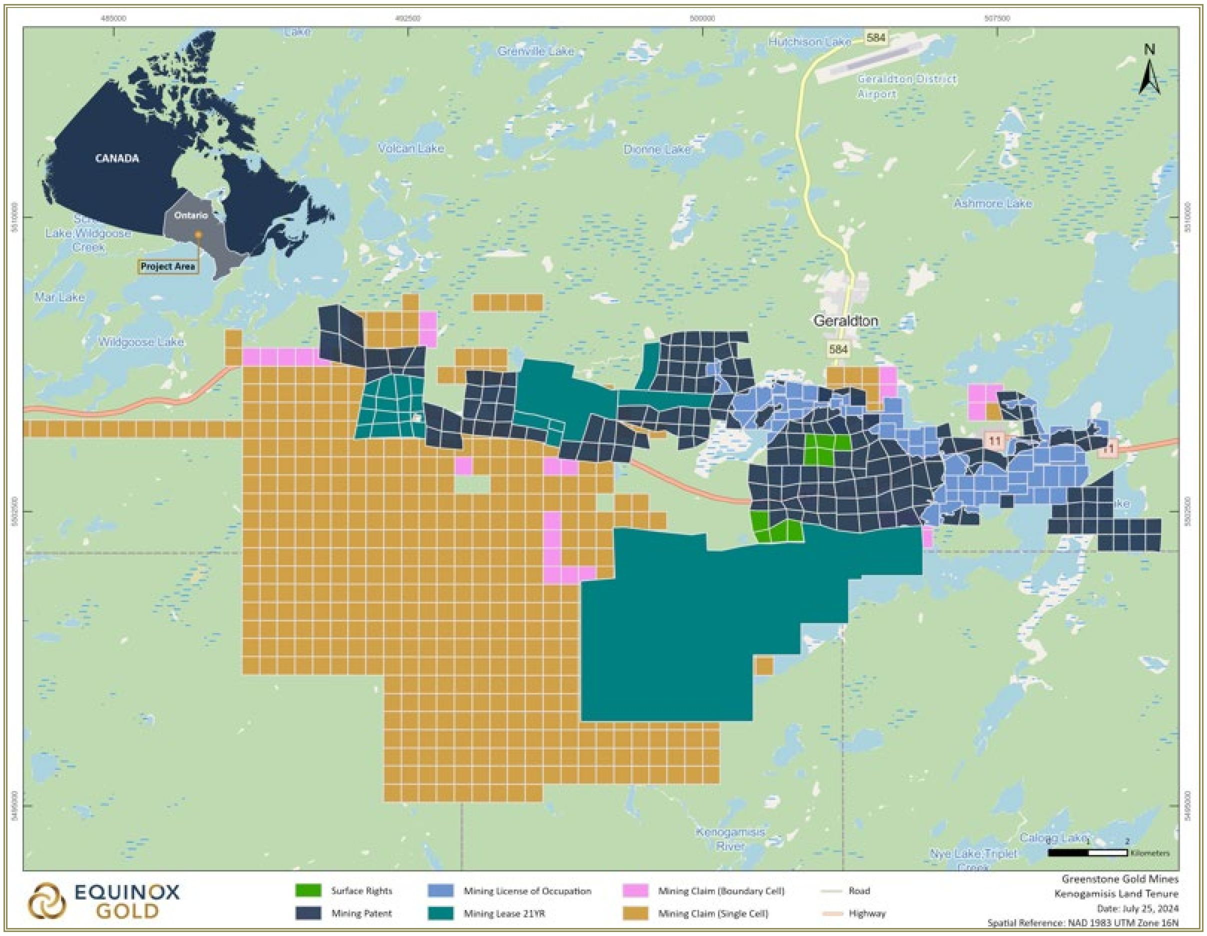


Figure 4-4: Greenstone Mine Properties

4.3.2 Hardrock Agreement Overview

The Hardrock land package is an amalgamation of multiple historical mining properties with several underlying agreements and royalties. Gignac et al. (2016) provide a detailed history of the various agreements and acquisitions. A summary of royalties currently in effect is listed below and shown in Figure 4-5:

- Essar Steel Algoma Inc. (2% net smelter return [NSR])

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

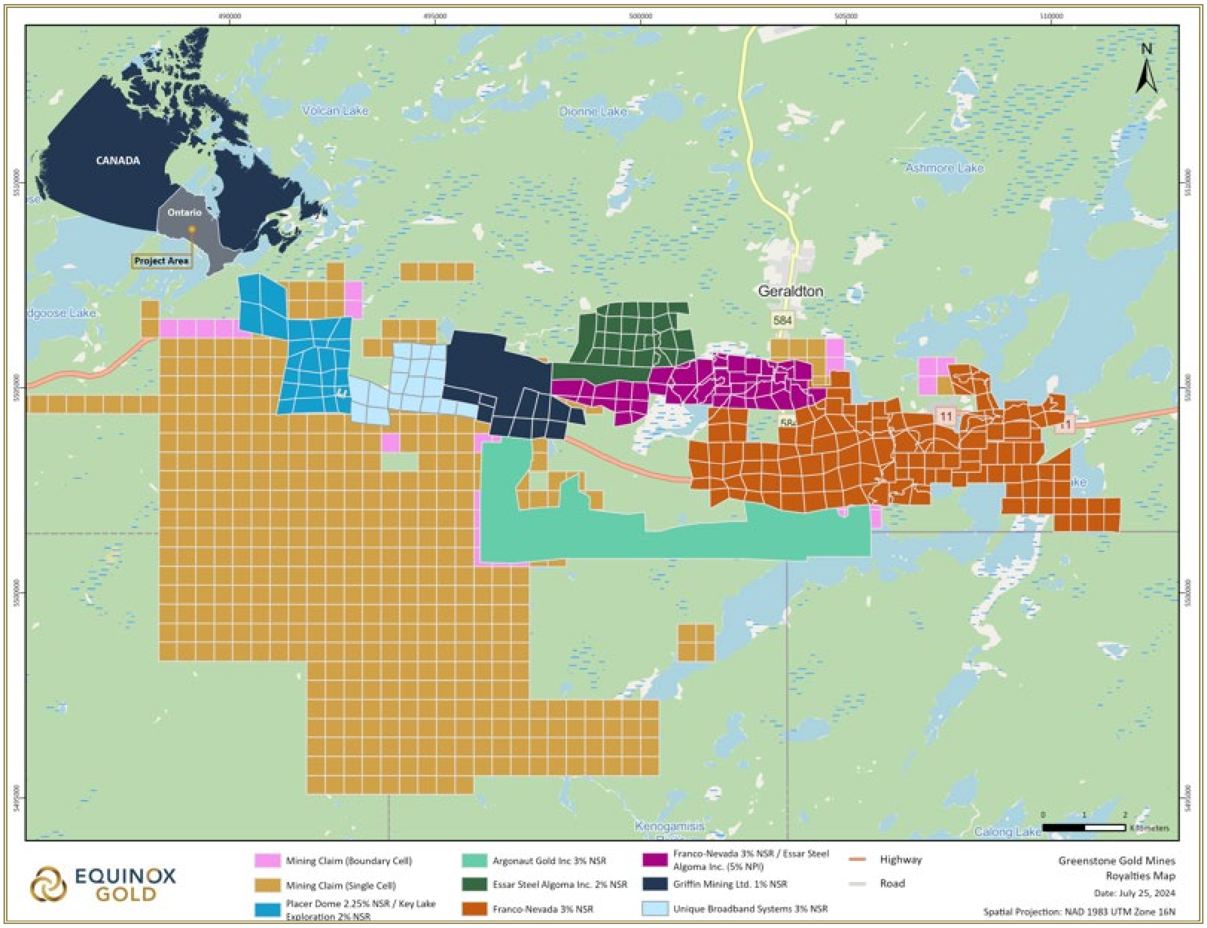


Figure 4-5: Greenstone Mine Royalties

In October 2018, a mining lease was granted over CLM 535, which covers the southern part of the Greenstone Mine area. The lease, LEA-109765, is subject to renewal in 2039. In December 2016, GGM acquired the surface rights for the patented claims in Errington and Ashmore townships—TB 10604 to TB 10608, TB 11879, TB 11885, TB 11886, and TB 11888.

On May 13, 2024, Equinox Gold announced that the Company had completed its acquisition of the remaining 40% of GGM from certain funds managed by Orion Mine Finance LP (the "Greenstone Acquistion"), giving Equinox Gold 100% ownership of GGM and the Greenstone Mine.

As part of the Greenstone Acquistion, the Company assumed obligations under a stream agreement with Nomad Royalty Company Ltd, dated October 28, 2021, as amended (the "Stream Agreement"). Under the Stream Agreement, the Company is required to deliver an amount of refined gold equal to 2.375% of the gold produced from Greenstone, until the Company has delivered a cumulative total of 120,333 ounces, and 1.583% of the gold production from Greenstone thereafter. In exchange for the gold deliveries, the Company will receive consideration equal to 20% of the spot gold price at the time of delivery.

4.3.3 Greenstone Gold Property Partnerships

On March 9, 2015, Centerra Gold Inc. (Centerra) and Premier Gold Mines Limited (Premier) formed a 50-50 partnership for exploring, developing, and operating the GGM properties. GGM was formed to hold and manage the partnership assets. Centerra made an initial cash contribution to the partnership in the amount of $85 million for its 50% limited partner interest. In accordance with the Partnership Agreement, Centerra committed to solely fund up to $185 million in capital to develop the Hardrock Project, following which all funding for the Partnership would be made pro-rata.

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

On May 13, 2024, Equinox Gold announced that the Company had completed its acquisition of the remaining 40% of Greenstone Gold Mine GP Inc. from certain funds managed by Orion Mine Finance Management LP (Orion), giving Equinox Gold 100% ownership of GGM and the Greenstone Mine.

4.3.4 Agreement with Tombill Mines

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

4.4 Brookbank Property Area

The Brookbank property area is within 1:50,000 scale NTS Sheet 42E/12 and lies 10 km northeast of Beardmore (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 northeast of the Thunder Bay airport on the Trans-Canada Highway. The project area hosts the Brookbank deposit, and the Cherbourg and Foxear targets.

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

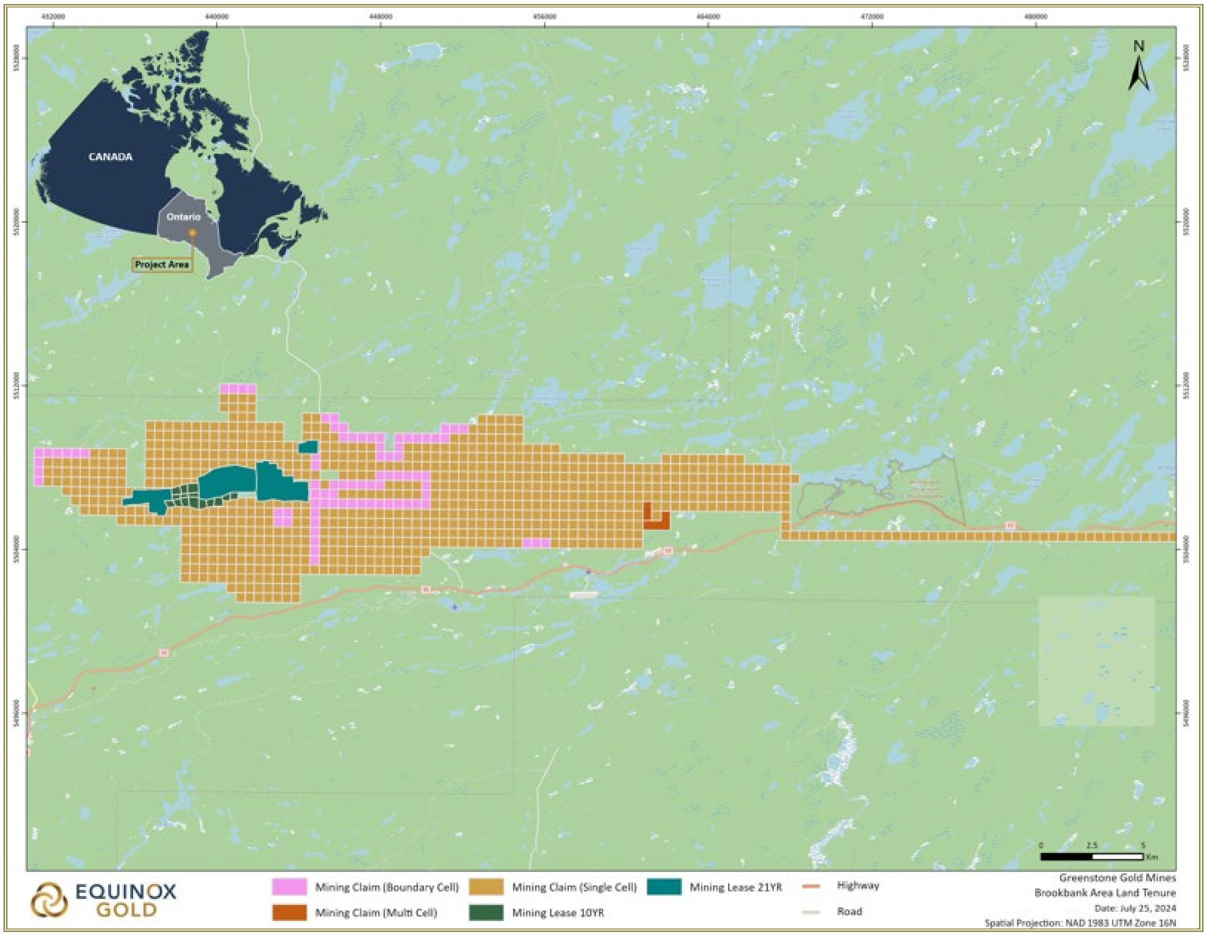


Figure 4-6: Brookbank Property

4.4.1 Brookbank Agreements

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

4.5 Viper Property

Premier staked the Viper claims between 2013 and 2015 (Figure 4-7). GGM owns 100% of the Viper claim group. The Viper property consists of 216 contiguous cell claims totalling 4,250.9 ha.

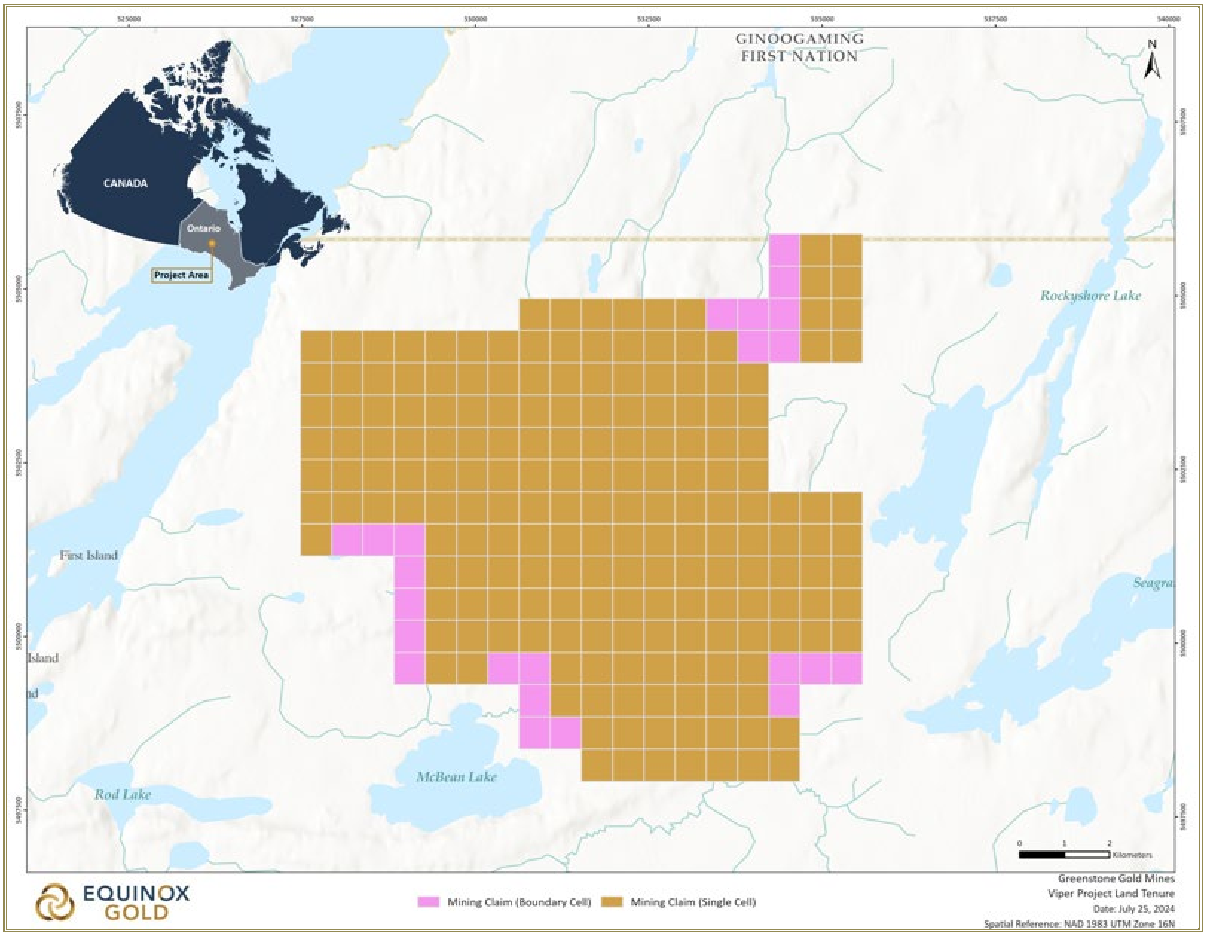


Figure 4-7: Viper Property

4.6 Permits

Permits are required to undertake drilling, surface stripping, and trenching. Table 4-2 lists all the permits in place for the GGM properties as of June 30, 2024.

Table 4-2: Permits on GGM Properties

Permit	Permit No.	Issued by	Effective Date	Expiry Date
Closure Plan Amendment	HP-MG003-EV-130-0019_0	Ministry of Mines	12-29-23	n/a
ECA (air/noise)	6588-CLLFPD	Ministry of the Environment, Conservation and Parks	07-27-23	n/a
ECA (Northside and Full Scale ETP)	0735-C9PMD6	Ministry of the Environment, Conservation and Parks	04-25-22	n/a
ECA (On-site Landfill)	5892-BZWFUP	Ministry of the Environment, Conservation and Parks	05-14-21	n/a
ECA (Southside TMF and GFCD)	6354-CF8HG7	Ministry of the Environment, Conservation and Parks	08-31-22	n/a
ECA (Temporary ETP and MHT Seepage Collection)	1846-CGYLYN	Ministry of the Environment, Conservation and Parks	09-02-22	n/a
ECA (Mill and TMF Commissioning)	7554-D4EPL5	Ministry of the Environment, Conservation and Parks	04-23-24	10-31-24
Federal Decision Statement	n/a	Impact Assessment Agency of Canada	12-10-18	n/a
Provincial Notice of Approval	Order in Council 404/2019	Ministry of the Environment, Conservation and Parks	04-04-19	n/a
Fisheries Act Authorization	14-HCAA-00498	Fisheries and Oceans Canada	11-21-19	12-31-24
Amendment to Schedule 2 of Metal and Diamond Mining Effluent Regulations under the *Fisheries Act*	n/a	Environment and Climate Change Canada	09-10-20	n/a
Release of Tree Reservation for Patented Land	n/a	Ministry of Natural Resources	2019 and 2021	n/a
Permit to Remove Trees on Crown Land	P10033	Ministry of Natural Resources	04-01-21	03-31-24
Aggregate Permit for S4 Pit	626461	Ministry of Natural Resources	03-06-20	NA
Aggregate Permit for T2 Pit	626462	Ministry of Natural Resources	03-16-20	NA
Aggregate Permit for S1 Pit	626463	Ministry of Natural Resources	03-06-20	NA
Aggregate Permit for TMF Quarry	626528	Ministry of Natural Resources	12-15-21	NA
PTTW (Northside and Full Scale ETP)	P-300-3202796853	Ministry of the Environment, Conservation and Parks	04-06-23	11-22-26
PTTW (Temporary ETP and MHT Seepage Collection)	2855-BMBLGL	Ministry of the Environment, Conservation and Parks	06-18-20	05-31-30
PTTW (Southside 1 [TMF, Aggregate Pits, GFCD])	1532-CGGG2Q	Ministry of the Environment, Conservation and Parks	09-08-22	06-30-26
PTTW (Southside 2 [Culvert Crossings])	6064-C44KSW	Ministry of the Environment, Conservation and Parks	06-22-21	06-30-26
PTTW (Freshwater Intake)	PTTW #P-300-5187643949	Ministry of the Environment, Conservation and Parks	01-23-24	12-01-28
PTTW (Temporary Bypass Channel Construction Dewatering)	PTTW #P-300-1225855748	Ministry of the Environment, Conservation and Parks	01-23-24	02-01-25
PTTW (Temporary Bypass Channel Service-Life Operations)	PTTW #5745-D3CKES	Ministry of the Environment, Conservation and Parks	03-21-24	03-31-29
Travel Permit—Kenogamisis Dam	n/a	Ministry of Natural Resources	03-27-23	12-31-24
Land Use Permit (SW1 Water Intake Line on Kenogamisis Lake)	NP2019-0448-LUP001	Ministry of Natural Resources	08-01-20	07-31-30
Land Use Permit (Temporary ETP Discharge Line)	NP2020-0459-LUP001	Ministry of Natural Resources	08-01-20	07-31-30

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

This section is an update from the 2021 Report, issued January 26, 2021. Some comments have been added to cover available information through June 2024.

5.1 Accessibility

5.1.1 Greenstone

The Greenstone Mine is 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, 275 km southwest on Ontario King's Highway 11 (Trans-Canada Highway). Public roads are maintained by various levels of government. Geraldton also hosts a municipal airport equipped to accommodate small aircraft.

Since 2021, a network of well established road access covering the main infrastructure of the Mine is now built. The south portion of the Mine is accessed via Highway 11 and the remainder can be easily accessed by four-wheel-drive vehicles via numerous logging and bush roads that branch off the paved highways. Drill roads provide excellent access to the areas GGM is exploring. Those areas of the Mine not serviced by roads can be accessed by all-terrain vehicle (ATV), on foot, or by boat during the summer, and snowmobile in the winter.

5.1.2 Brookbank, Key Lake, Kailey, and Viper Properties

The Brookbank, Key Lake, Kailey, and Viper properties are also within the Municipality of Greenstone. The main part of the Brookbank property can be accessed via the Wendigokan Road, an all-weather gravel road that leads from to Highway 11. The eastern part of the Brookbank property is accessible by Highway 801, a paved secondary road that also leads from Highway 11.

Figure 5-1: Greenstone Mine Main Access Routes

5.2 Climate

The Mine area has a continental climate typical of temperate regions in the mid-latitudes influenced by both polar and tropical air masses. In this climate, seasonal temperature variations are represented by short, warm summers and long, cold winters.

The nearest permanent weather monitoring station is approximately 14 km north of the Mine at the Greenstone Regional Airport, which services Geraldton and the surrounding area. Weather statistics for the period between 1991 and 2020 record a mean daily temperature of 0.8°C, with a maximum and minimum daily temperature of 27.9°C and −39.3°C, respectively. The mean annual precipitation comprises 546.4 mm of rainfall and snowfall of 249.0 cm. On average, precipitation was recorded on 167 days a year. The annual average relative humidity in the morning is about 84.0%. The annual average wind speed for the area is about 11.4 km/h, and the prevailing winds are westerly. In summer, winds blow most frequently from the west and south, while in the fall and winter, the most frequent winds are westerly.

Weather conditions do not seriously hinder exploration and mining activities on the property, but adjustments to the type of work performed are subject to seasonal variation—for example, geological mapping in the summer and drilling on frozen lakes in the winter.

5.3 Local Resources

The Mine benefits from local human resources and services available in Geraldton and surrounding areas. Geraldton has a population of approximately 1,800 people and is part of the Municipality of Greenstone, which also includes Longlac, Nakina, Beardmore, and an extensive area of unincorporated land. The Municipality's population is 4,300. Throughout all phases of the development of the Mine, GGM has undertaken extensive consultation with local Indigenous communities and Geraldton.

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

5.4 Infrastructure

GGM has established an exploration office in Geraldton, near the Mine, for desktop work, core logging, sample preparation, and storage. This is a large, converted warehouse that consists of several offices, a boardroom, and a significant amount of space and industrial shelving for storage. Currently the space is predominantly used for processing and shipping of RC drilling samples. Core-logging benches and core racks are installed to support core-drilling programs. Space is allocated for a core-cutting room and installation of diamond saws, previously housed in a core-cutting shed at the former exploration site. There is off-site storage at the Magnet mine site for core, RC rep samples, and assay pulps and rejects. A house in the MacLeod townsite contains all historical maps and sections from the mines within the tenement area, as well as records for exploration projects.

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

GGM now owns and operates a laboratory with sample preparation and analytical capabilities in Geraldton, having acquired the facility from Actlabs. In times of heavy sample flow, this facility may cooperate with the Actlabs facility in Thunder Bay to ensure timely turnaround.

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

The Mine is within a mining-friendly district with active mines and milling facilities at Hemlo, Thunder Bay, Kapuskasing, and Timmins. The district also has good transportation and regional mining-related infrastructure.

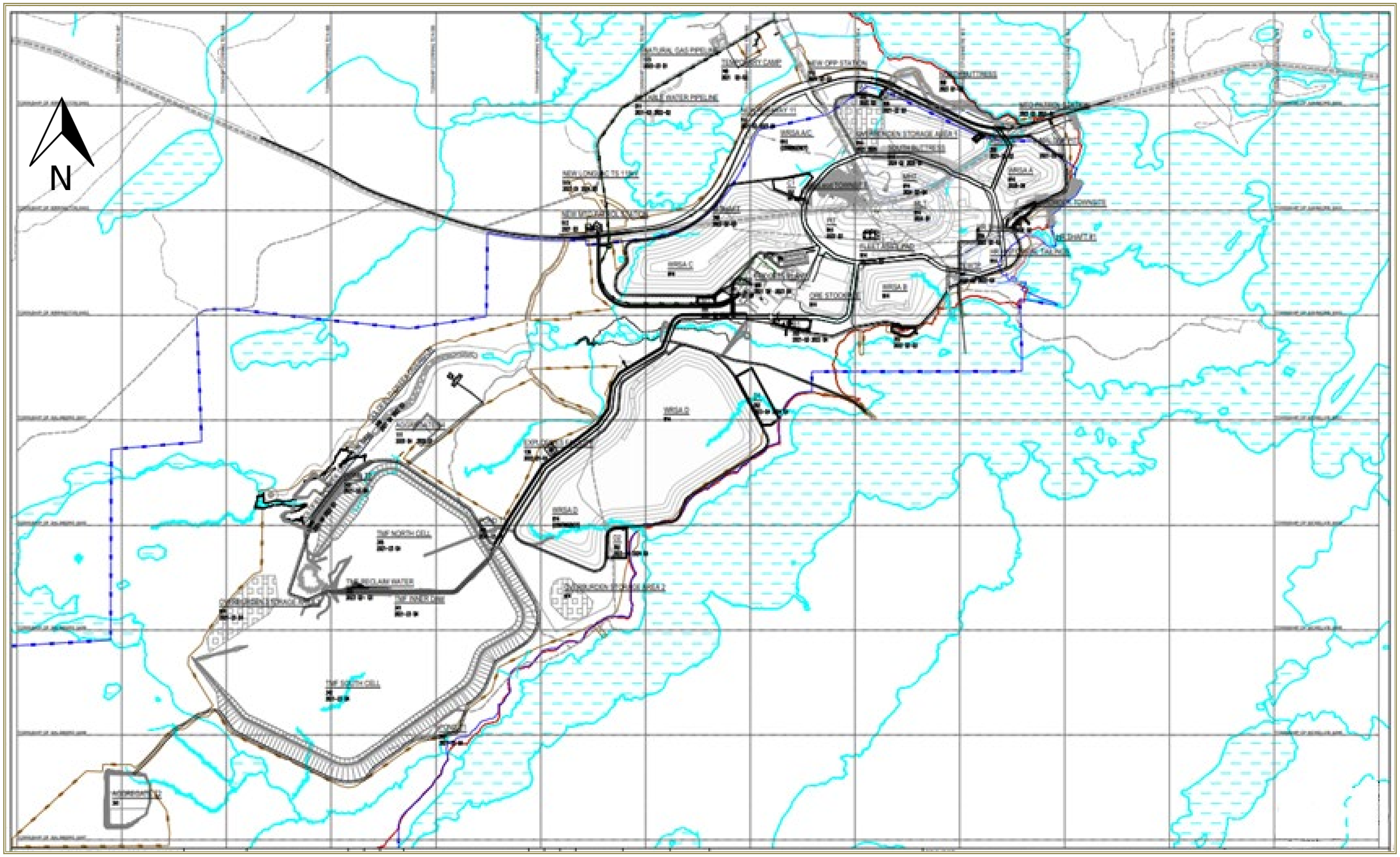
There are adequate surface rights for the planned mining-related infrastructure, including waste rock storage areas, tailings management facility, and processing and administration facilities (Figure 5-2). The arrangement of mining-related infrastructure is constrained by the surrounding lakes and watercourses.

5.4.1 Water

Geraldton has its own potable water treatment system and water distribution network. The GGM field office and houses within the Mine's subdivision are serviced by this system. The plan is to connect to the Municipality's potable water in the future.

5.4.2 Sewage

Geraldton has its own sewage treatment facility. However, the sewage-collecting network does not extend south of Kenogamisis Lake. Consequently, houses in the MacLeod and Hardrock townsites have their own septic beds.



Note: The grid is 1 by 1 km

Figure 5-2: General Site Layout

5.5 Physiography

The Mine lies within the Canadian Boreal Shield 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 abound in this landscape, and drainage patterns are typically dendritic, with sporadic angular drainage influenced by bedrock outcrops.

The Mine area topography 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. Lower-lying areas are characterized by swamps and ponds, with overall very-poor drainage throughout the area. The surrounding land is at about 335 masl. The largest lake adjacent to the Mine is Kenogamisis Lake, which bounds the Mine to the south, east, and north. This lake's elevation is about 330 masl.

Vegetation in the area is dominated by coniferous trees; the most common species are 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

This section summarizes the historical work done on the Hardrock, MacLeod–Cockshutt, and Mosher mines, derived from the 2021 Report on the MRE for the Hardrock, Brookbank, and Key Lake Projects.

Table 6-1 presents the statistics on gold production, diamond drilling, and underground development for all three historical mines. A detailed chronological summary of the historical work carried out on these mines since 1980 is provided in Table 6-8. Figure 6-1 shows the area of the current MRE and historical work.

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. "Hardrock" Smith discovered gold-bearing quartz stringers near the Hardrock Number 1 shaft, and Tom Johnson and Robert Wells discovered gold on Magnet Lake, which later hosted the Bankfield Gold mine. T. A. Johnson and T. Oklend soon follow with the discovery of gold 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—the first successfully producing mine. To the west of the 1931 Hardrock 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 ore body in greywacke, was identified on both the MacLeod–Cockshutt and Hardrock properties.

Production on the Mosher Long Lac mine began in 1962 (west of, and immediately down-plunge of the same mineralized zones exploited in the MacLeod–Cockshutt mine); then, in 1967, the MacLeod–Cockshutt, Mosher, and Hardrock mines amalgamated and remained in production until 1970. The consolidated Hardrock, 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 at 4.9 g/t Au) in the period from 1934 to 1970.

In the 1980s, Lac Minerals Ltd. (Lac Minerals) 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., Hardrock 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 mainly focused on the near-surface portion of the F Zone and targets along the plunging nose of the albite porphyry.

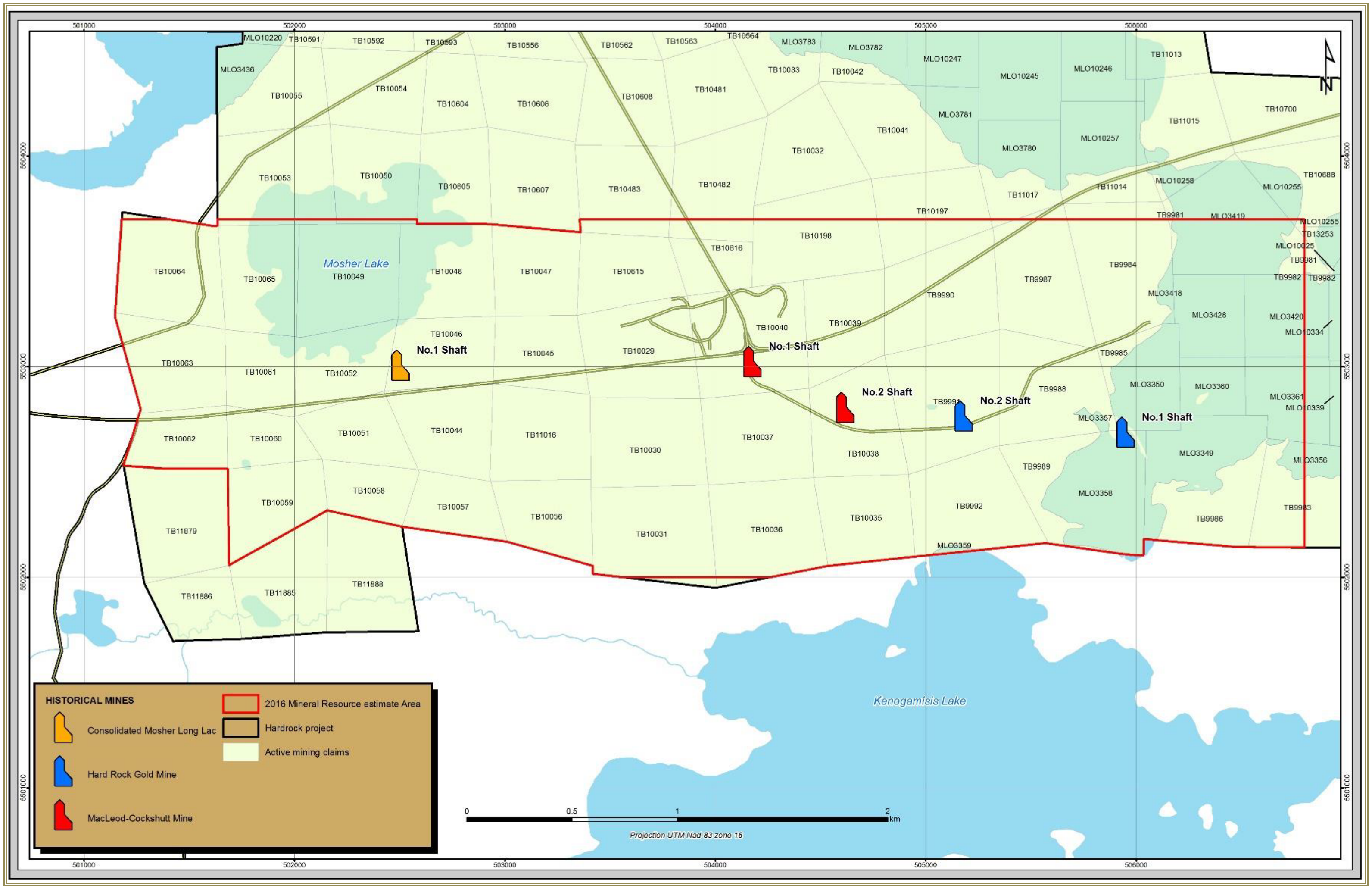


Figure 6-1: Greenstone Mine 2024 Resource Estimate Area (red outline) Representing Limits of Historical Work

Table 6-1: Gold Production, Diamond Drilling, and Underground Development Statistics—Little Long Lac, Hardrock, MacLeod–Cockshutt, Mosher Long Lac, and MacLeod Mines

Description	Past-Producing Mines					Total
	Little Long Lac	Hard Rock	MacLeod–Cockshutt	Mosher Long Lac	MacLeod–Mosher	
Years of Production	1934–1953	1938–1951	1938–1967	1962–1966	1967–1970	n/a
Ore Milled (ton)	1,780,516	1,458,375	9,403,145	2,710,657	1,656,413	**15,228,590**
Ore Milled (ton)	1,615,713	1,323,038	8,530,533	2,459,108	1,502,698	**13,815,377**
Au Grade (oz/ton)	0.34	0.19	0.15	0.12	0.11	**0.14**
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**	**16,933**	**1,083**	**0**	**32,037**
Total Length of Underground DDH (m)	**23,353**	**67,423**	**224,168**	**59,591**	**1,043**	**352,226**
Total Length of Drifting (m)	**Unknown**	**10,572**	**32,698**	**7,292**	**7,259**	**57,822**
Total Length of Crosscutting (m)	**Unknown**	**3,608**	**8,976**	**3,267**	**3,369**	**19,221**
Total Length of Raising (m)	**Unknown**	**1,878**	**10,589**	**2,467**	**4,300**	**19,235**

Note: DDH = diamond drill holes.

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

(NOTE: The "Pit Resource" and "Ramp Resource" are historical in nature and should not be relied upon. They are unlikely to conform to current NI 43-101 criteria or *CIM Definition Standards for Mineral Resources and Reserves* (CIM Definition Standards, 2014), 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, which changed its name to Barrick Gold Corporation in 1995. The former Lac Minerals 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 RC holes in overburden, sonic holes in historical tailings, and an additional 40,000 feet of diamond drilling, mainly on the 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 & White, 1997). The agreement ended in 1997. Barrick, through Lac Properties Inc., began a rehabilitation program, which continued until 2001. This saw construction of the current visitor's centre, re-contouring and seeding of the historical MacLeod tailings near Highway 11, and capping of old mine shafts.

In 2000, Lac Properties Inc. (Lac Properties) retained Golder to conduct a stability assessment of the F Zone crown pillar of the MacLeod–Cockshutt mine (Telesnicki & Steed, 2007). During their investigation, Golder drilled a borehole (369.5 m) to determine whether caving had occurred above the stopes. Their study also included a literature review of the properties of the mined material at the Hardrock mine, rock mass classification of the rock core from the investigation borehole and a correlation of numerical modelling results with the field investigation and conclusions. The drilling 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. The classification of the rock mass overlying the workings indicated that the quality was "good" to "very good." Due to the depth of the mine workings and the 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, Lac Properties retained Golder to conduct a stability assessment of the crown pillar of the Hardrock mine (Soni & 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 Hardrock mine, rock mass classification of the rock core from the investigation boreholes, and a correlation of numerical modelling results with the field investigation and conclusions. The drilling indicated that the crown pillar overlying the workings was intact at the time of the study. Golder observed no unravelling or caving of the crown pillar above the working, and no unexpected geometries were encountered. The 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 safety factor.

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

In 2007, Premier began signing various agreements to gain an interest in the property and began exploration drilling on the property (see Section 4).

Following Premier's acquisition of Lac Properties' claims in late 2008, 91,802 m in 346 holes were drilled, with work focused on the North Iron Formation area, the Hardrock-Porphyry Hill area and the Hardrock-East Pit area.

In March 2010, Reddick et al. (2010) published a new MRE for the Hardrock deposit and a supporting NI 43-101 technical report. The 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 were applied to constrain the resources, assuming a resource with bulk mineable characteristics. Contained metal and MRE are summarized in Table 6-2.

Table 6-2: Mineral Resources—Hardrock Area (Reddick et al., 2010)

Mineral Resources Class	Tonnage (Mt)	Gold Grade (g/t)	Tonnage (Mton)	Grade (oz/ton)	Contained Gold (koz)
Indicated	11.6	1.82	12.7	0.053	675
Inferred	7.3	1.81	8.1	0.053	425

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

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

A regional prospecting program was completed during summer 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 from trace amounts to 3 g/t Au.

Diamond drilling continued in 2010 on and around the old Hardrock, 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. Later, regional exploration became a more important focus, with the exploration of magnetic targets and other targets surrounding historical mine sites on the property. The main zones 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 drifted on the 13th level. No follow-up occurred below that level.

In 2011, Premier drilled 204 diamond drill holes (DDH) 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. These and other zones mentioned above are described in detail in Section 8.5.3.

Murahwi et al. (2011) prepared an updated MRE for the Hardrock deposit and a supporting NI 43-101 technical report. Contained MRE 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/t)	Estimated Gold Grade (g/t)	Tonnes (kt)	Contained Gold (oz)
Open Pit	Measured	0.83	2.446	6,865	540
Open Pit	Indicated	0.83	2.280	5,833	428
Open Pit	Measured + Indicated	0.83	2.370	12,698	968
Open Pit	Inferred	0.83	2.483	615	49
Underground	Measured	2.80	5.993	2,312	446
Underground	Indicated	2.80	5.827	5,757	1,079
Underground	Measured + Indicated	2.80	5.875	8,069	1,524
Underground	Inferred	2.80	5.397	6,187	1,074
OP + UG	Measured	-	3.340	9,177	986
OP + UG	Indicated	-	4.042	11,590	1,506
OP + UG	Measured + Indicated	-	3.732	20,767	2,492
OP + UG	Inferred	-	5.133	6,802	1,123

Premier drilled 125 DDH totalling 68,549 m between January and October 2012. 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 close to the historical Hardrock 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.

Murahwi et al.'s (2013) NI 43-101 technical report presented an updated MRE for the Hardrock deposit. Those contained-metal and MRE are summarized in Table 6-4.

Table 6-4: Mineral Resources—Hardrock Deposit (Murahwi et al., 2013)

Cut-Off Category	Mineral Resource Category	Tonnes (kt)	Gold Grade (g/t)	Gold Ounces (koz)
Open Pit	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	Measured (M)	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

Note: Numbers may not add up due to rounding.

Between October 31, 2012, and August 9, 2013, 153 DDHs (72,776.4 m) were drilled on the Hardrock deposit. These holes were included in an updated MRE InnovExplo Inc. (InnovExplo) prepared in 2013, which was presented in Brousseau et al.'s (2013) NI 43-101 technical report. Premier released the updated MRE on October 29, 2013. Contained metal and MRE are summarized in Table 6-5.

Table 6-5: Mineral Resources—Hardrock Deposit (Brousseau et al., 2013)

Resource Type	Parameters Cut-Off (g/t Au)	Area		Total
		In-Pit >0.50 g/t Au	Underground >3.00 g/t Au	
Indicated	Tonnes (kt)	50,228	5,522	55,750
	Grade (g/t Au)	1.46	5.01	1.81
	Au (koz)	2,352	889	3,241
Inferred	Tonnes (kt)	17,793	16,919	34,711
	Grade (g/t Au)	1.50	5.38	3.39
	Au (koz)	859	2,925	3,784

Note: numbers may not add up due to rounding.

Between August 10, 2013, and December 31, 2013, Premier added 144 DDHs on the Hardrock deposit, totalling 66,606.7 m. None of these holes were included in Brousseau et al.'s (2013) MRE.

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

Between January 1, 2014, and May 26, 2014, Premier added 38 DDHs on the Hardrock deposit, 12,653.6 m in all (Brousseau et al., 2014). Thirteen DDHs from 2013 were also deepened in 2014, adding 2,867.3 m. Seven historical DDHs were resampled to add new assay results in the 2014 MRE. These holes were not previously sampled and had therefore been rejected from the 2013 database (Brousseau et al., 2013). These holes represented 5,709 new metres in the 2014 database. InnovExplo included the new data in its updated MRE presented in Brousseau et al.'s (2014) NI 43-101 technical report. Premier released the updated Mineral Resource on August 25, 2014.

Contained metal and MRE are summarized in Table 6-6.

Table 6-6: Mineral Resources—Hardrock Deposit (Brousseau et al., 2014)

Resource Type	Parameters Cut-Off (g/t Au)	Area		Total
		In-Pit >0.50 g/t Au	Underground >3.00 g/t Au	
Indicated	Tonnes (kt)	83,868	5,169	89,037
	Grade (g/t Au)	1.47	5.40	1.70
	Au (koz)	3,973	898	4,870
Inferred	Tonnes (kt)	10,225	12,922	23,147
	Grade (g/t Au)	1.53	5.40	3.69
	Au (koz)	501	2,242	2,744

Premier carried out two small drilling programs in the past-producing Bankfield mine (Brousseau et al., 2014). The Bankfield mine is 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 about 10 km west-southwest of Geraldton. Between December 15, 2013, and January 24, 2014, two DDHs were drilled for 1,043 m. Six DDHs were added in this area in 2014, totalling 2,513 m. None of these holes were included in Brosseau et al.'s (2014) MRE.

From 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 comprised three stripped areas with detailed geological mapping and channel sampling. The channels were five metres apart in the east–west direction and sampled to the extent of the outcrop every one metre. This work aimed to verify and establish structural elements and grade continuity at surface. In addition, 128 mechanical test pits were completed on the

Hardrock Project to evaluate the overburden thickness. The results of these test pits were not used in the 2014 MRE update (Brousseau et al., 2015).

In February 2015, Premier and Centerra Gold Inc. formed a definitive 50/50 partnership to develop the Hardrock Property. In July 2015, the joint partnership was named Greenstone Gold Mines GP Inc.

Through 2015, work continued to support a feasibility study to mine the Hardrock Property. GMS of Montreal, Québec, authored an NI 43-101-compliant report in December 2016. The feasibility study envisaged an open pit mining and milling operation processing 27,000 t/d of ore, 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 pushback on the north wall of the pit would mine out a portion of the historical MacLeod tailings and would also require relocating Highway 11, the MTO garage, and the OPP station. The pit and waste rock piles would effectively occupy a large portion of the historical MacLeod and Hardrock townsites. To this end, GGM embarked on a campaign to buy the surface rights from individual landowners in these areas. The 2015 feasibility study also included designs for waste rock piles, TMF, and related run-off collection ditches and water treatment. The TMF would be located southwest of the mine and mill facilities and would require relocation of Goldfield Creek. Total undiscounted cash flow, after taxes, was estimated at $1,636 million, or $709 million at a 5% discount rate, after total capital cost of $1,242 million and sustaining capital of $257 million, generating an after-tax IRR of 14.4%.

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

Definition drilling continued in 2019, with 76 RC holes totalling 5,946 m and 54 NQ-size DDHs for 12,108 m. The results of this drilling are also included in the current MRE.

Premier filed a NI 43-101 technical report that GMS prepared for the Hardrock Project, titled *NI 43-101 Technical Report Hardrock Project, Ontario, Canada*. This independent report, dated January 26, 2021 (effective date December 16, 2020), provides detail to the disclosure contained in the Company's news release issued on December 16, 2020. Contained-metal and MRE are summarized in Table 6-7.

Table 6-7: Summary of 2019 Mineral Resource (Inclusive of Open Pit Mineral Reserves) for the Hardrock Project (Sirois et al., 2021)

Category	In-Pit >0.3 g/t Au			Underground >2.0 g/t Au		
	Tonnage (Mt)	Gold Grade (g/t)	Gold Ounces (koz)	Tonnage (Mt)	Gold Grade (g/t)	Gold Ounces (koz)
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

Notes:

- The Independent and Qualified Person for the MRE, 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 $1,250 gold price and a revenue factor 0.78.
- Underground 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 Mineral 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/t Au; however, the official resource is at a cut-off grade of 0.30 g/t Au.
- Underground Mineral Resources were compiled at cut-off grades of 1.50, 2.00, 2.50, 3.00, 3.50, 4.00 and 4.50 g/t Au; however, the official resource is at a cut-off grade of 2.00 g/t Au.
- 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/t Au) was undertaken on raw assay data and established on a per zone basis, varying up to 140 g/t Au.
- Compositing was done on drill-hole intervals within the mineralized zones (2 m lengths).
- Mineral Resources were estimated using GEOVIA GEMS (Version 6.8.2 from drill-hole and surface-channel sampling, using a
- 3-pass inverse distance cube (ID^3) 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) = tonne x grade/31.10348. Calculations used metres, tonnes, and g/t.
- The number of 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%; general and administrative (G&A), $1.59/t; sustaining capital, $0.70/t; gold price, $1,625/oz; milling recovery, 91.1%.

A significant drilling campaign was conducted during the winter of 2021/2022 targeting inferred blocks at depth and east of the pit design outlined in the 2019 feasibility study document. Fifty-six DDHs totalling 15,421 m and 67 RC drill holes totalling 4,189 m were drilled.

GMS was retained to update the mineral resource block model for the Hardrock deposit, incorporating new drilling undertaken since the release of the previous block model. The scope of work for the 2022 block model update was:

- Incorporate new drilling undertaken in 2021 in the eastern portion of the deposit.
- Identify coherent zones of mineralization in the external grade shell domain to model manually to reduce the ratio of external grade shells to principal domains.
- Update gold-grade estimate using a similar approach to previous resource estimation.
- Update arsenic, sulphur, and total carbon model.
- Update void model using information obtained from the 2021 drilling.
- Update lithology model and use to recode bulk density.

The current 2024 MRE is based on those models validated and endorsed by the QP. Adjustments were made on the categorization of the MRE. Those adjustments will be explained in Section 14. Those block models were filtered by the current topography, pit designs, whittle shells, and latest optimization parameters up to June 30, 2024.

From September 2022 up to the end of June 2024, a total of 16,049 m of RCGC were drilled on the Hardrock deposit. These programs focused on areas that were under-drilled within the original planned starter pit, within the redesigned Phase 1B pit, and to fully define the North Zone voids before mining.

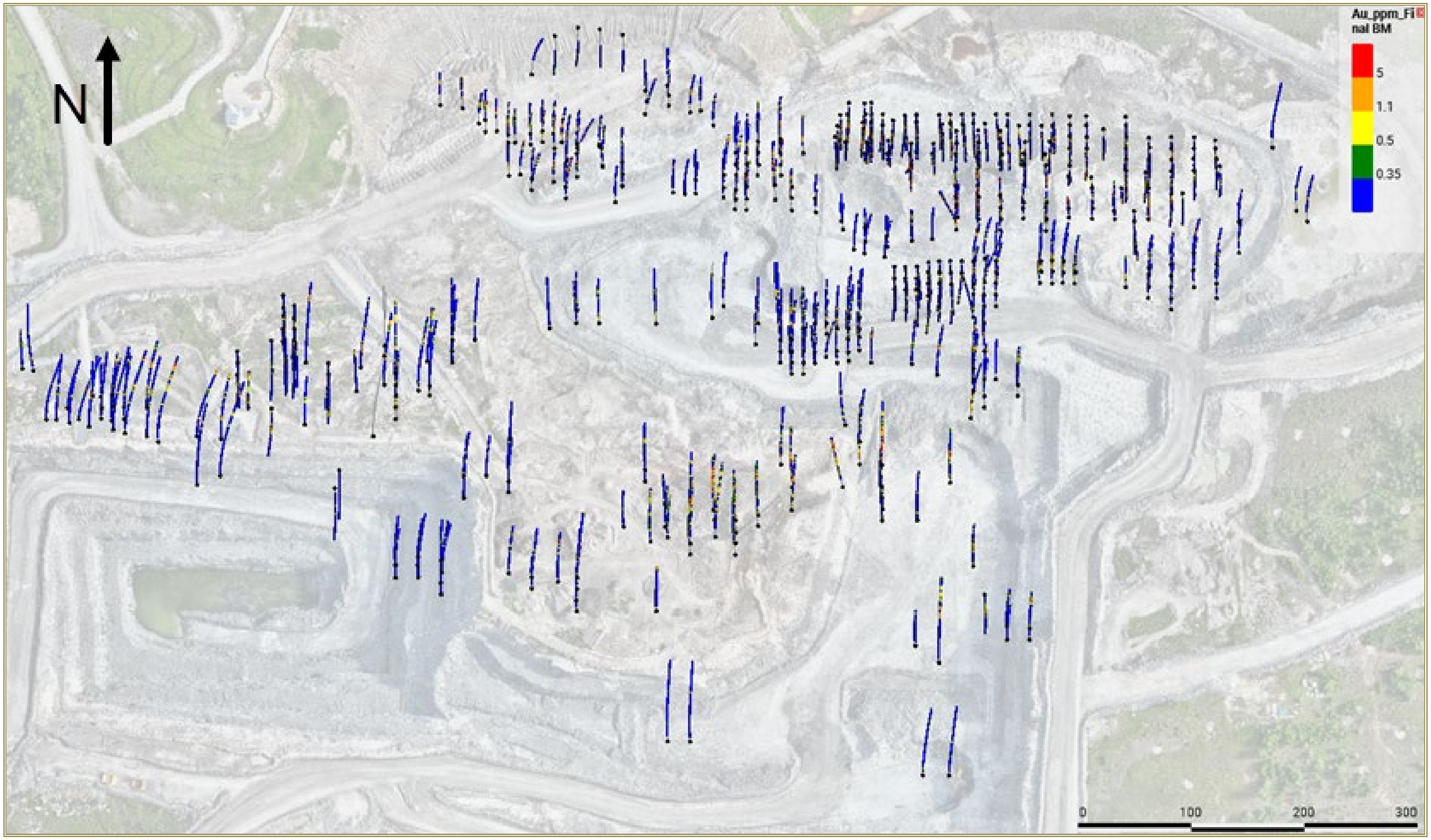


Figure 6-2: RCGC Programs Performed at the Greenstone Mine (since September 2022)

Table 6-8: Summary of Post-Production Exploration Activity

Year	Work Description	Activity	Comments [1]	Reference
1980	Studies of existing underground reserves; lithological reconnaissance		Studies of existing underground reserves; lithological reconnaissance	Gray (1994)
1982	"Ore reserves" and "ore potential" in the Hardrock and MacLeod/Mosher mines	Historical "reserves" of 1,300,000 tons at 0.140 oz/ton Au (Proven and Probable ore) [1] 80% of total ore below Level 13 of the 3.6 Mosher winze (No. 3 shaft) of the down-plunge of the F Zone and South Zone mineralization	"Ore reserves" and "ore potential" in the Hardrock and MacLeod/Mosher mines	Jarvi (1982)
1987	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 Hardrock D and F Zones, North and South porphyry, and Homestake–Porphyry Hill several IP anomalies were partially tested	Line cutting; ground magnetometer, VLF EM, and IP surveys; diamond drilling (37 DDH = 6,218.9 m	Gray (1994); 2012 Premier's Prospectus)
1988	Diamond drilling (40 DDH = 9,052.6 m)	DDH program targeted the open pit potential of the Hardrock D and F Zones, North and South Porphyry, and Homestake–Porphyry Hill	Diamond drilling (40 DDH = 9,052.6 m)	Gray (1994); 2012 Premier's Prospectus
1993	106 RC 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[1] Ramp resource: 1,600.000 tons at 0.127 oz Au/t[1]	106 RC RCO drill holes (1,483.2 m); diamond drilling (28 DDH = 5,125.2 m); geological resource estimate	Gray (1994); Mason and White (1993)
1994	17 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.	17 RCO drill holes (395.6 m); 21 sonic drill holes (304.8 m); diamond drilling (78 DDH = 11,961.9 m)	Gray (1994)
1995	Pre-feasibility study; MRE	Pit resource: 2,900,000 tons at 0.086 oz Au/ton[1] Underground resource: 1,400,000 tons at 0.131 oz Au/ton[1]	Reddick et al. (2010); Mason & White (1995)	Pre-feasibility study; MRE
1995	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	Diamond drilling (7 DDH = 1,024.4 m)	Murahwi et al. (2011, 2012)
1996	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	Diamond drilling (24 DDH = 1,024.4 m); metallurgical work on the previous sonic holes; samples from tailings; environmental assessment work	Reddick et al. (2010)

Year	Work Description	Activity	Comments [1]	Reference
1997	Diamond drilling (1 DDH = 185.0 m); geological resource estimate	Pit resource: 9,800,000 tons at 0.047 oz Au/ton[1] Tailings resource: 11,200,000 tons at 0.023 oz Au/t[1]	Diamond drilling (1 DDH = 185.0 m); geological resource estimate	Reddick et al. (2011)
2000	Diamond drilling (1 DDH = 369.5 m)	Diamond drilling program tested the F Zone crown pillars at the past producing MacLeod–Cockshutt mine	Diamond drilling (1 DDH = 369.5 m)	Telesnicki and Steed (2007)
2002	Diamond drilling (16 DDH = 2,116.8 m)	Diamond drilling program tested some crown pillars at the past-producing Hardrock mine	Diamond drilling (16 DDH = 2,116.8 m)	Soni and Steed (2002)
2008	Acquisition of the Lac Claims		Acquisition of the Lac Claims	
2009	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)	Diamond drilling (346 DDH = 91,802 m); Overburden stripping with power washing, mapping and sampling	
2010	Diamond drilling (279 DDH = 114,611 m); overburden stripping with power washing, mapping, and sampling; regional prospecting program	Three areas were stripped (East MacLeod, Headframe, and Portal Zones); diamond drilling focused on the same area as in 2009; main zones drilled were North, F, SP, NN, and K Discovery of the F2 and Z zones; new MRE and a supporting NI 43-101 technical report	Diamond drilling (279 DDH = 114,611 m); overburden stripping with power washing, mapping, and sampling; regional prospecting program	Premier Gold: Reddick et al. (2010)
2011	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 MRE and a supporting NI 43-101 technical report	Diamond drilling (204 DDH = 107,413 m)	Premier Gold: Murahwi et al. (2011)
2012	Diamond drilling (125 DDH = 68,549 m)	Diamond drilling program focused on the Fortune, HGN and P Zones; updated MRE and supporting NI 43-101 technical report	Diamond drilling (125 DDH = 68,549 m)	Premier Gold: Murahwi et al. (2013)
2012/13	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 MRE and supporting NI 43-101 technical report	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	Premier Gold: Brousseau et al. (2013)
2014	Preliminary economic assessment	Using the consistent gold price of $1,250/oz and an exchange rate of CAD/USD 1.00:0.95, the Hardrock Project generates an NPV of C$518.70 million (discounted at 5%) and an IRR of 23.02% before taxes; and C$358.97 million (discounted at 5%) and an IRR of 19.02% after taxes.	Preliminary economic assessment	Premier Gold: St-Laurent et al. (2014)
2014	38 DDH = 12,653,6 m) (from Jan. 01, 2014 to May 26, 2014)	Updated MRE and supporting NI 43-101 Technical Report	38 DDH = 12,653,6 m, from Jan. 01, 2014, to May 26, 2014	Brousseau et al. (2014)

Year	Work Description	Activity	Comments [1]	Reference
2015	Formation of a 50/50 Partnership	New NI 43-101 technical report	Formation of a 50/50 Partnership	Premier Gold: Brousseau et al. (2015)
2016	Feasibility Study	Updated MRE and supporting NI 43-101 technical report	Feasibility Study	Gignac et al. (2016)
2018	RC Drilling 405 holes = 19,995 m, blasthole drilling 62 holes = 535 m	Updated MRE (not published)	RC Drilling 405 holes = 19,995 m, blasthole drilling 62 holes = 535 m	Sirois (2018)
2019	Drilling 76 RC holes = 5,946 m, 54 DDH = 12,108 m	Resource update and project design work (this study)	Drilling 76 RC holes = 5,946 m, 54 DDH = 12,108 m	
2022	Greenstone Gold Mines GP Inc.	RCGC Drilling 67 holes = 4,189 m, 56 DDH = 15,421 m	Internal Resource update (not published)	Purchase (2022)
2023	Greenstone Gold Mines GP Inc.	As and S reviewed and creation of updated block models for these 2 attributes	Internal Block Models Updates (not published)	Beaulieu (2023)
Sep 2022 to Jun 2024	Greenstone Gold Mines GP Inc.	RCGC: drilling 496 holes = 28,002 m	Data not used for the 2024 Mineral Resource update.	GGM
2024	Greenstone Gold Mines GP Inc.	Update gold prices, optimisation parameters and resource shells	Updated MRE and NI 43-101 Technical Report (this study)	Sirois (2024)

Note: [1] 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 *Definition Standards on Mineral Resources and Reserves*, 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.
Very low frequency electromagnetic = VLF EM.

6.2 Brookbank

6.2.1 Exploration History

This section is an update from the 2021 Report, issued January 26, 2021. Some comments have been added to cover available information through June 2024.

The earliest known work on the Brookbank property is a program of surface trenching and limited diamond drilling that Connell Mining and Exploration Co. Ltd. (Connell Mining) carried out in 1934. 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 of 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 Brookbank–Sturgeon performed (if any).

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 DDHs totalling 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 DDHs totalling 3,567 m.

Between late 1982 and early 1983, Metalore drilled three widely spaced DDHs totalling 330 m to test the metavolcanic–metasedimentary contact on the Brookbank West property and one 453 m DDH on the Foxear property.

From September 1983 to March 1984, Metalore completed an additional 62 DDHs totalling 6,946 m, including four wedges. In July 1984, Metalore commissioned a combined helicopter-borne magnetometer, gamma-ray spectrometer, and VLF survey of its holdings in Sandra, Irwin, and Walters Townships, including the Brookbank property.

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

In 1986, Metalore concentrated on the Cherbourg Zone and completed 43 DDHs 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 DDHs totalling 11,203 m on the Brookbank Zone, and 10 DDHs totalling 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 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 x 15 m and took 215 channel samples totalling 244 linear metres. Detailed mapping was completed at a scale of one inch to ten feet. During 1989, drilling at the Brookbank Zone consisted of 18 DDHs totalling 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 DDHs drilled in 1989 and 15 of the pre-existing holes. Additional Placer Dome drilling at Cherbourg consisted of five DDHs totalling 1,437 m, with a further two DDHs totalling 984 m drilled at Foxear. Placer Drome dropped its option due to ongoing litigation between Ontex and Metalore.

From 1990 through to 1996, the Brookbank property 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).

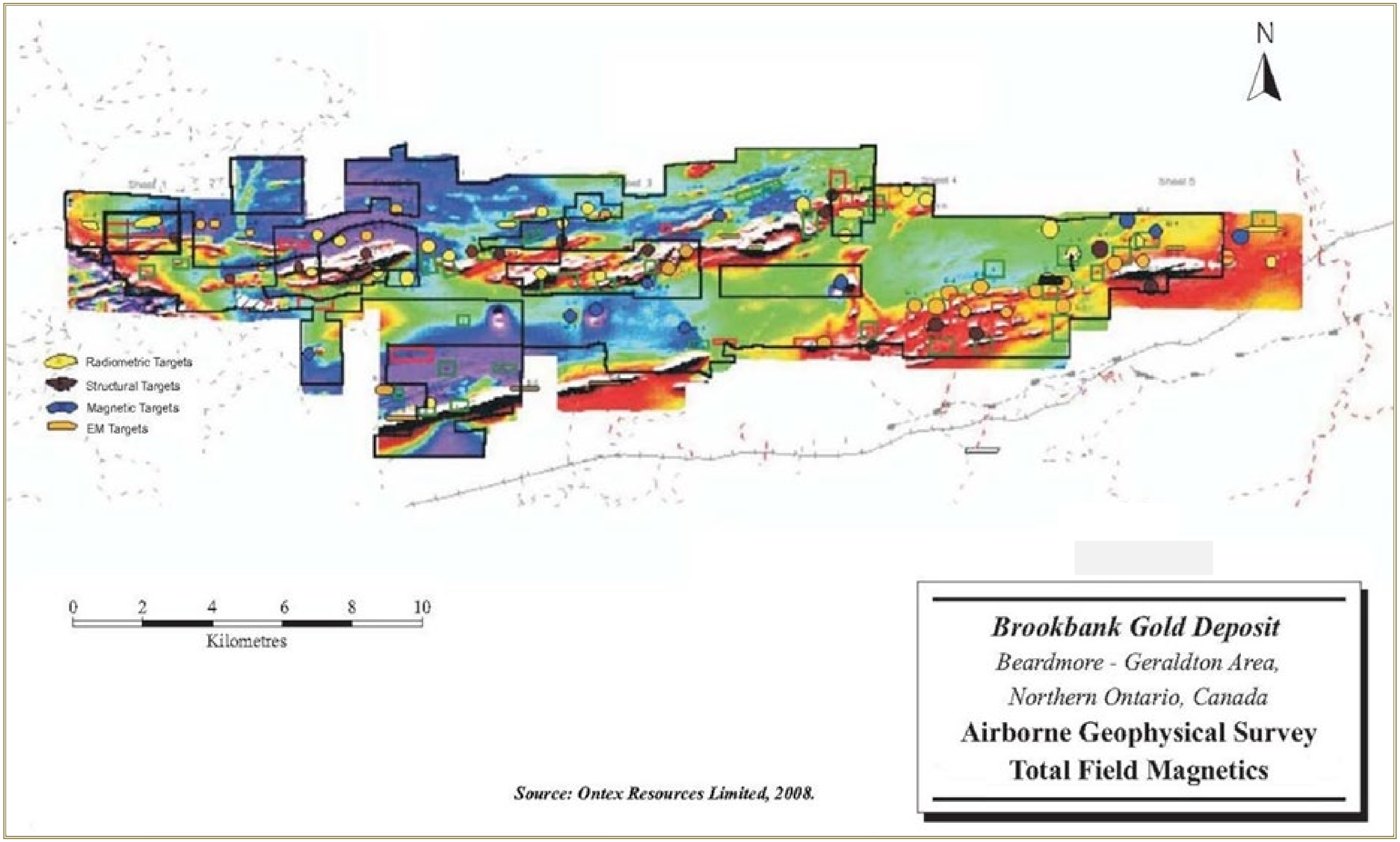
In 1993 and 1994, Metalore completed four DDHs totalling 533 m on the Brookbank Zone, fifteen DDHs totalling 2,107 m at Cherbourg, and seven DDHs (including one wedge) totalling 3,323 m at Foxear. Micon and J.R. Trussler & Associates both reviewed the data in 1994 on behalf of Metalore; the reviews were positive and both companies recommended additional work. However, the ongoing litigation between Ontex and Metalore precluded any work.

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, Ontex conducted all exploration on the property. 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 kilometres over the entire property in a north–south direction. The airborne program included collecting and delivering total field and calculated vertical gradient magnetics, VLF-EM, resistivity, and radiometrics K/Th/U ratio. The results are summarized in Figure 6-3.

The airborne survey results were reflective of geology and favourable 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 favourable structural break between known gold zones. However, the geophysical signature can be used to locate alteration on structural breaks that might contain mineralization.

The geophysical targets shown in Figure 6-3 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.



Source: Ontex Resources Ltd. (2008).

Figure 6-3: Major Helicopter Borne Geophysical Targets on the Brookbank Property

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 6-9 summarizes the historical drilling completed at and the Brookbank deposit prior to Premier's involvement.

Table 6-9: Ontex Historical Surface Diamond Drilling on the Brookbank Property (1999–2009 Drilling Campaigns)

Years	No. of Holes	Length (m)	Targeted Area
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 $0.0001 in cash for each Goldstone share held.

On August 16, 2011, Premier completed the previously announced acquisition of Goldstone for approximately $104 million. The Goldstone acquisition 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 adding the remaining portion of the Hardrock Project it did not previously 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.

6.2.2 Production History

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

6.2.3 Previous Resource Estimates

In 2009, Scott Wilson RPA completed a previous MRE on the Brookbank property for Ontex. This estimate is summarized in Table 6-10 and is contained in an NI 43-101 technical report dated May, 2009, titled *Technical Report on the Brookbank Gold Deposit, Beardmore-Geraldton Area, northern Ontario, Canada*. The Scott Wilson RPA Brookbank MRE is summarized in Table 6-10.

The 2009 MRE complies with the December 11, 2005, *CIM Definition Standards for Mineral Resources and Mineral Reserves* (CIM Definition Standards) as required by NI 43-101.

Mineral Resources—Hardrock Deposit

Table 6-10: Mineral Resource—Brookbank (Scott Wilson RPA 2009)

	Indicated Mineral Resources			Inferred Mineral Resources		
Zone	**Tonnes (kt)**	**Gold Grade (g/t)**	**Gold Ounces (koz)**	**Tonnes (kt)**	**Gold Grade (g/t Au)**	**Gold Ounces (koz)**
Brookbank	1,217	8.8	346	813	7.4	193
Cherbourg	80	10.1	26	141	8.1	37
Foxear	35	4.3	5	54	3.7	7
Total	**1,332**	**8.8**	**376**	**1,009**	**7.3**	**236**

Notes:

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

In 2012 Micon completed an NI 43-101-compliant MRE for GGM on the Brookbank deposit. This is summarized in Table 6-11.

Table 6-11: Mineral Resource—Brookbank (Micon 2012)

Area	Category	Tonnes (kt)	Gold Grade (g/t)	Gold Ounces (koz)
Open Pit	Measured	-	-	-
	Indicated	2,638	2.01	171
	M+I Pit	2,638	0.95	171
	Inferred	171	2.38	13
Underground	Measured	-	-	-
	Indicated	1,851	7.21	429
	M+I UG	1,851	7.21	429
	Inferred	403	4.07	53

Notes:

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

GMS completed an NI 43-101-compliant MRE on the Brookbank deposit for GGM in 2020. That estimate was diligently validated by the QP and was adopted for this Technical Report.

Table 6-12: Mineral Resource—Brookbank (GMS 2020)

Deposit	Cut-Off Category	Mineral Resource Category	Tonnes (kt)	Gold Grade (g/t)	Gold Ounces (koz)
Brookbank Property	Open Pit	Measured (M)	-	-	-
		Indicated (I)	1,147	2.24	83
		Subtotal M & I	**1,147**	**2.24**	**83**
		Inferred	45	2.07	3
	Underground	Measured (M)	-	-	-
		Indicated (I)	2,281	7.06	517
		Subtotal M & I	**2,281**	**7.06**	**517**
		Inferred	706	3.38	77

Notes:

- Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability.
- There are no Mineral Reserves at Brookbank.
- The independent and Qualified Person for the Brookbank MRE is Mr. James Purchase, P.Geo of GMS.
- The effective date of the Mineral Resource is September 3, 2020.
- Open pit mineral resources are constrained within an optimized pit shell using a gold price of $1,500, a CAD/USD exchange rate of 1.3 and a metallurgical recovery of 92%. An incremental ore haulage cost of $17.90/t milled is also assumed for Brookbank.
- Underground Mineral Resources 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/t.
- 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

In 2022, Greenstone Gold Mines requested BBA to perform a desktop-level study for the Brookbank deposit, based on ore being mined at Brookbank and transported by truck to the Greenstone processing plant at a rate of 800 tonnes per day (base case). The selected base case consisted of a combination of a small open pit mine and an underground mine. Base-case highlights are:

- Open Pit:
  - 520 kt at 1.90 g/t Au (31,800 oz)
  - Mine life of two years
  - Strip ratio = 3.2.
- Underground:
  - 2,880 kt at 4.90 g/t Au (454,000 oz)
  - Mine life of 11 years.
- Costs:
  - Pre-Production CAPEX = $35 million
  - Sustaining CAPEX = $214 million
  - AISC (open pit) = 1,091 $/oz
  - AISC (underground) = 1,167 $/oz.

- Economics:
  - NPV = $145 million (Based on a gold price of $1,550/oz)
  - IRR = 12%
  - Payback = 8.1 years.

6.3 Key Lake

The following Key Lake description is taken directly from the 2021 Report.

6.3.1 Exploration History

Placer Dome's drilling at Key Lake in the 1980s identified extensive zones of gold mineralization, but these were initially considered too low-grade to be economical (McCormack, 1984). Placer Dome conducted additional drilling in 1990 before abandoning the project. Subsequently, Cyprus confirmed two shallow mineralized shoots with average grades greater than 1 g/t Au (Gasparetto & Stevenson, 1996). Roxmark carried out some drilling in 2010 and 2011 and identified wide mineralized intervals, such as 1.6 g/t Au (0.047 oz Au/ton) over a drilled length of 30 m in KL-11-109 (including 11.9 g/t Au over 0.3 m). Higher grade intervals, such as 5.6 g/t Au (0.16 oz Au/ton) over 16.1 m in KL-11-112 (including 31.6 g/t Au 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 & White, 1986). The ore bodies comprised a series of veins, each with a maximum strike length of about 100 m and an 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

In 2012 Micon completed an NI 43-101-compliant MRE for GGM, summarized in Table 6-13 and described in more detail in Section 19.2.2.

Table 6-13: Mineral Resource—Key Lake (Micon, 2012)

Area	Indicated Mineral Resources			Inferred Mineral Resources		
	Tonnes (kt)	Gold Grade (g/t)	Gold Ounces (koz)	Tonnes (kt)	Gold Grade (g/t)	Gold Ounces (koz)
Open Pit	2,572	1.17	97	1,345	1.29	56
Underground	31	6.48	6	58	3.57	7
Total	**2,603**	**1.27**	**103**	**1,403**	**1.44**	**63**

Notes:

- CIM Definition Standards of November 27, 2010, were followed for Mineral Resource estimation.
- Totals may not add correctly due to rounding.
- Mineral Resources which are not Mineral Reserves have demonstrated economic viability.
- The Effective date of the estimate is December 31, 2012.
- Lower gold cut-off used for reporting open-pit Mineral Resources is 0.5 g/t Au, and 2.8 g/t for underground Mineral Resources

In 2012 Micon completed an NI 43-101-compliant MRE for GGM, summarized in Table 6-13 and described in more detail in Section 19.2.2.

Table 6-14: Mineral Resource—Key Lake (GMS 2020)

Deposit	Cut-Off Category	Mineral Resource Category	Tonnes (kt)	Gold Grade (g/t)	Gold Ounces (koz)
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:

- Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability.
- No mineral reserves are published at Key Lake.
- The independent and qualified person for the Key Lake MRE is Mr. James Purchase, P.Geo. of GMS.
- The effective date of the Mineral Resource is September 3, 2020.
- Open pit mineral resources are constrained within an optimized pit shell using a gold price of $1,500, a CAD/USD exchange rate of 1.3 and a metallurgical recovery of 90%. An incremental ore haulage cost of $4.51/t milled is also assumed for Key Lake.
- No underground Mineral Resources are quoted.
- Mineral Resources are quoted at an open-pit lower cut-off of 0.4 g/t.

6.4 Kailey (Little Long Lac)

The following Kailey description is taken directly from the 2021 Report.

6.4.1 Exploration History

Kailey is 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, various individuals staked claims. Sudbury Diamond Drilling Co. drilled the area of the gold discovery and outlined a commercial ore shoot. Little Longlac Gold Mines Ltd. was formed in 1933 to develop the mine. A three-compartment shaft was sunk to 137.16 m. An electric powerline reached the mine in 1934, 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, and so forth. 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 property area includes the past-producing Little Long Lac mine. From 1934 to 1956, the Little Long Lac mine produced 1,615,713 tonnes at a grade of 11.7 g/t Au, for a total of 605,499 oz of gold. 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 in the Geraldton area is presented in Table 6-15.

Table 6-15: Historical Mine Production in the Beardmore-Geraldton Area

Past-Producing Mines	Operating Period	Ore Milled Metric Measurements		Ore Milled Imperial Measurements		Gold
		Tonnes	(g/t Au)	Tons	(oz Au/t)	(oz)
Little Long Lac	1934–1954, 1956	1,615,713	11.66	1,780,516	0.34	605,449
Hardrock	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	1937–1942, 1944–1947	209,627	9.86	231,009	0.29	66,417
Magnet Consolidated	1938–1943, 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**

Source: Mason & McConnell (1983).

6.4.3 Previous Mineral Resource Estimates

Table 6-16 summarizes Micon's NI 43-101-compliant MRE in 2012.

Table 6-16: Mineral Resource—Kailey (Micon 2012)

Area	Category	Tonnes (kt)	Gold Grade (g/t)	Gold Ounces (koz)
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:

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

An NI 43-101 compliant MRE was completed by GMS for GGM in 2020. This is summarized in Table 6-17.

Table 6-17: Mineral Resource—Kailey (GMS 2020)

Deposit	Cut-Off Category	Mineral Resource Category	Tonnes (kt)	Gold Grade (g/t)	Gold Ounces (koz)
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:

- Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability.
- No mineral reserves are published at Kailey.
- The independent and qualified person for the Kailey MRE is Mr. James Purchase, P.Geo. of GMS.
- The effective date of the Mineral Resource is September 3, 2020.
- Open pit mineral resources are constrained within an optimized pit shell using a gold price of $1,500, a CAD/USD exchange rate of 1.3 and a metallurgical recovery of 90%. An incremental ore haulage cost of $1.70/t milled is also assumed for Kailey.
- No underground Mineral Resources are quoted.
- Mineral Resources are quoted at an open-pit lower cut-off of 0.4 g/t.

7 GEOLOGICAL SETTING AND MINERALIZATION

This section is an update from the 2021 Report, issued January 26, 2021. Some comments have been added to cover available information through June 2024.

7.1 Greenstone Mine Regional Geological Setting

The Greenstone Mine (formerly Hardrock Property or 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 & 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 2,711 to 2,685 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 2,769 Ma (Anglin et al., 1988).

Figure 7-2 shows the regional geology of the Beardmore–Geraldton area. The following has been drawn from the Hardrock Property 2010 NI 43-101 Report completed by Reddick Consulting Inc. (T. Armstrong, M. Srivastava, & 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 proximal 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 Bankfield–Tombill Deformation Zone is marked by the Bankfield–Tombill 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 banded 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.

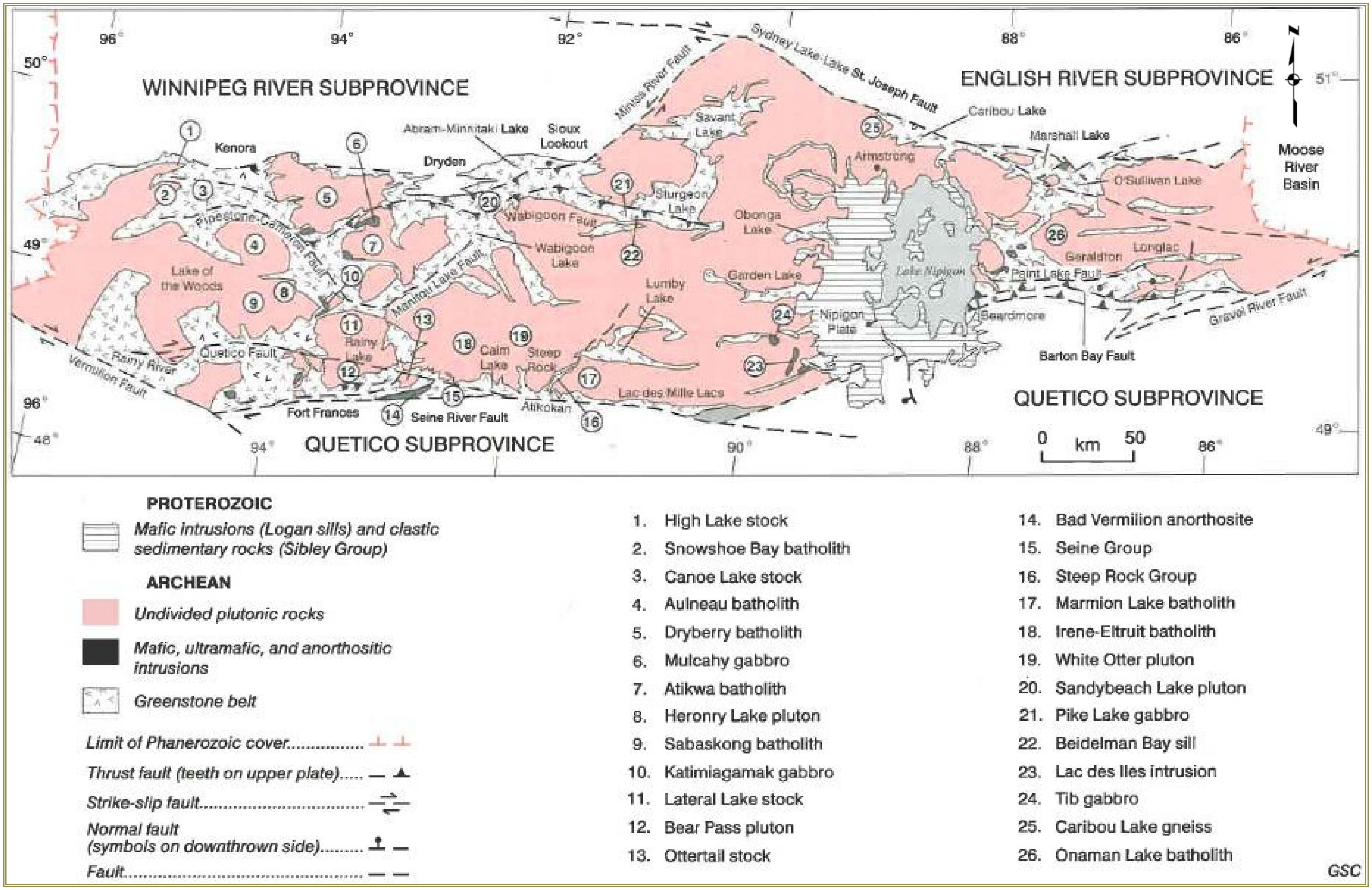


Figure 7-1: Plan of Major Geological Elements—Wabigoon Subprovince

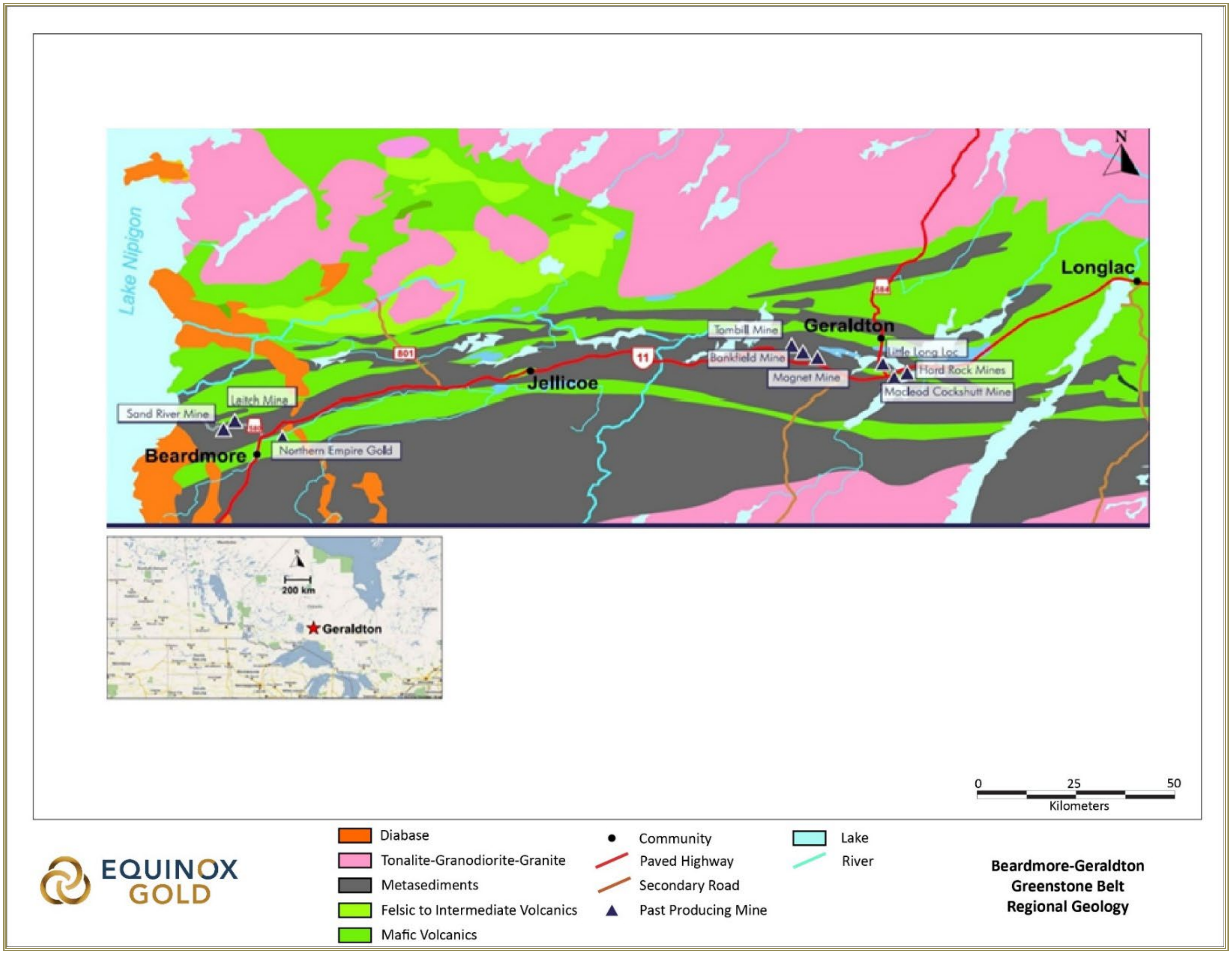


Figure 7-2: Regional Geology of the Geraldton–Beardmore Area

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 boudinage 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 postdates the tight folding of the primary lithological units.

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 orebodies appear to have been largely structurally controlled, the deformation of the orebodies has not been as intense as that of the host rocks (Table 7-1). Nevertheless, there are areas where local folding and boudinage 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

Regional Deformation Style	Description of Structures	
	Folding	**Foliation**
D1 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
D2 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 to 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}
D3 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 S_{3A}; Axial-planar to F_{3A}; Oriented ACW to bedding
	Dextral E-trending shear zones localized along S2 and lithological contracts	
	Z-shaped F_{3A} drag folds overprinting foliation in shear zones	Sinistral slip S_{3B} crenulation cleavage; Axial-planar to F_{3B}

Source: Lafrance et al. (2004); Tóth et al. (2013, 2014a, 2014b, 2015).

7.2 Property Geology

A map showing the Property geology can be seen in Figure 7-3. The following has been modified from the Hardrock Property 2010 NI 43-101 report completed by Reddick Consulting Inc. (T. Armstrong, M. Srivastava, & 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 & 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.”

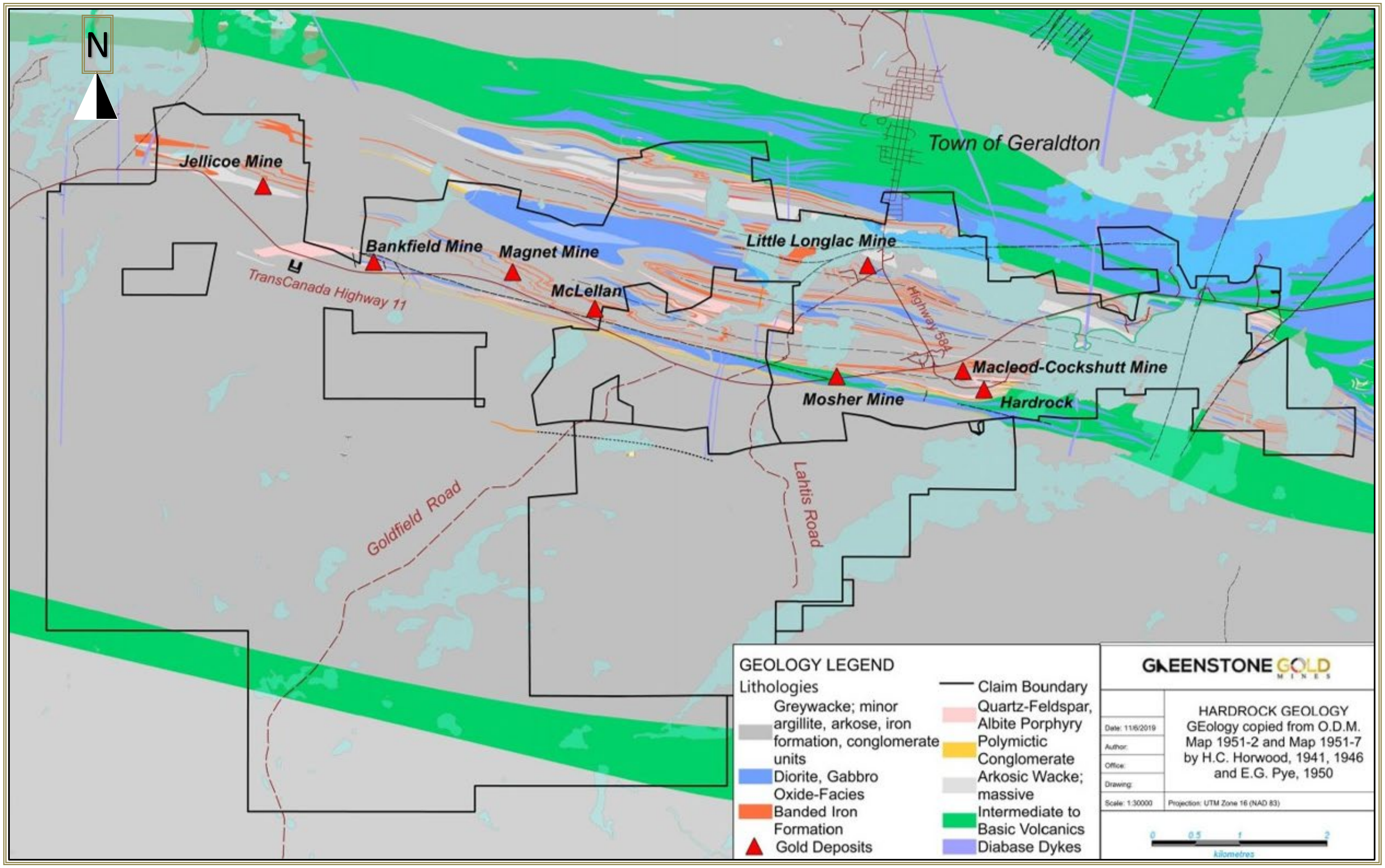


Figure 7-3: Geology of Historical Mines near Greenstone Mine

South of the Bankfield–Tombill Fault, the rocks 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% to 5% magnetite, making the distinction between argillite and lean iron formation difficult in places. Individual millimetre to centimetre-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 centimetres to decimetres thick, with millimetre-to-centimetre 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% millimetre to centimetre-scale magnetite beds and has been historically referred to as "Lean Iron Formation" in the mining terminology (Figure 7-4).

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 match the geometry of the conglomerate unit that occurs in the vicinity of the Hard Rock and MacLeod–Cockshutt Mines (Figure 7-5 and Figure 7-6).

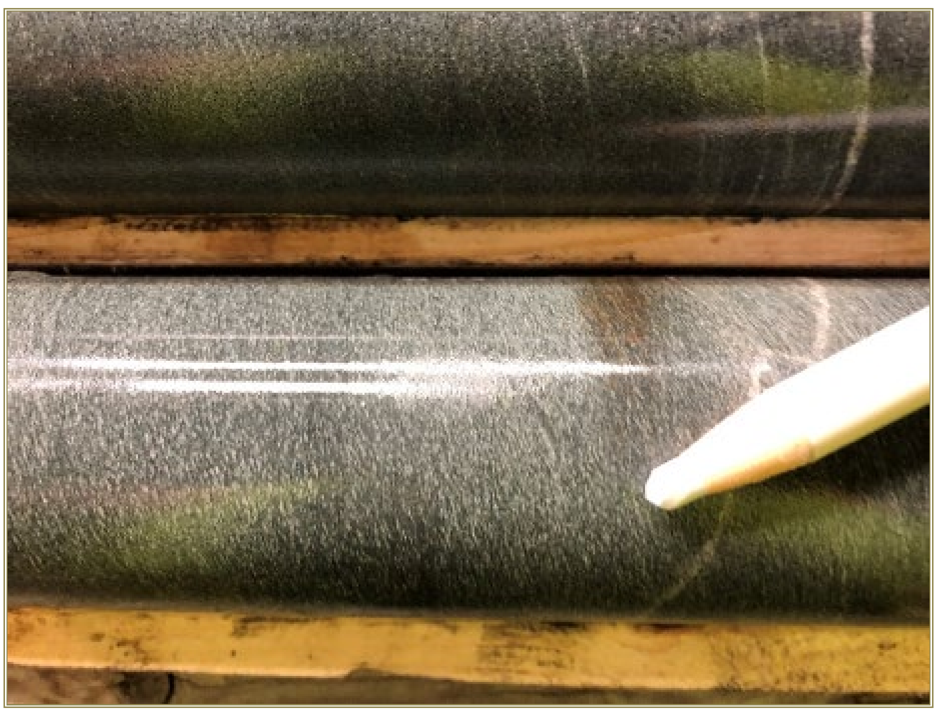


Figure 7-4: Arkosic Unit within Greywackes, Hardrock Deposit Area, DDH 19-21

Note: The folded contact and minor drag folds.

Figure 7-5: Contact of Hardrock Porphyry and Greywacke; Pen Points North

Figure 7-6: Contact of Sheared Gabbro (Right) with BIF (Left) DDH 19-52

7.3 Alteration

The Greenstone Mine 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 gold, silicon, potassium, barium, and carbon dioxide, and depletion in magnesium and calcium (Lavigne, 2009). This is manifested as silicification and quartz veining enveloped by sericite–carbonate alteration, accompanied by disseminated pyrite, arsenopyrite, and pyrrhotite (Figure 7-7).

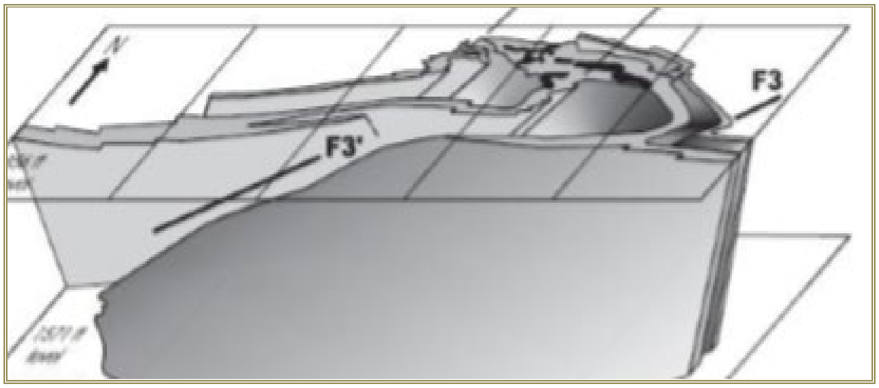
Figure 7-7: Quartz Carbonate Veins and Sericite Carbonate Alteration at the F Zone

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 on fluids but also on 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 & 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 m 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 subprovince scale, regional folds cut by D3 dextral shear zones are promising targets for discovering the next generation of large gold deposits."



Source: Lafrance et al. (2004).

Figure 7-8: Block Diagram of North Zone at the Historical MacLeod–Cockshutt and Hard Rock Mines (showing orebodies in black)

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 or folded, while the wall rock is foliated, silicified, and sericitized with disseminated pyrite, pyrrhotite, and arsenopyrite.

7.4.1 Identification of Gold Mineralization

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 reinterpreted, 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).

The North Domain consists 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 plunge to the west at 20° to 35°. 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.

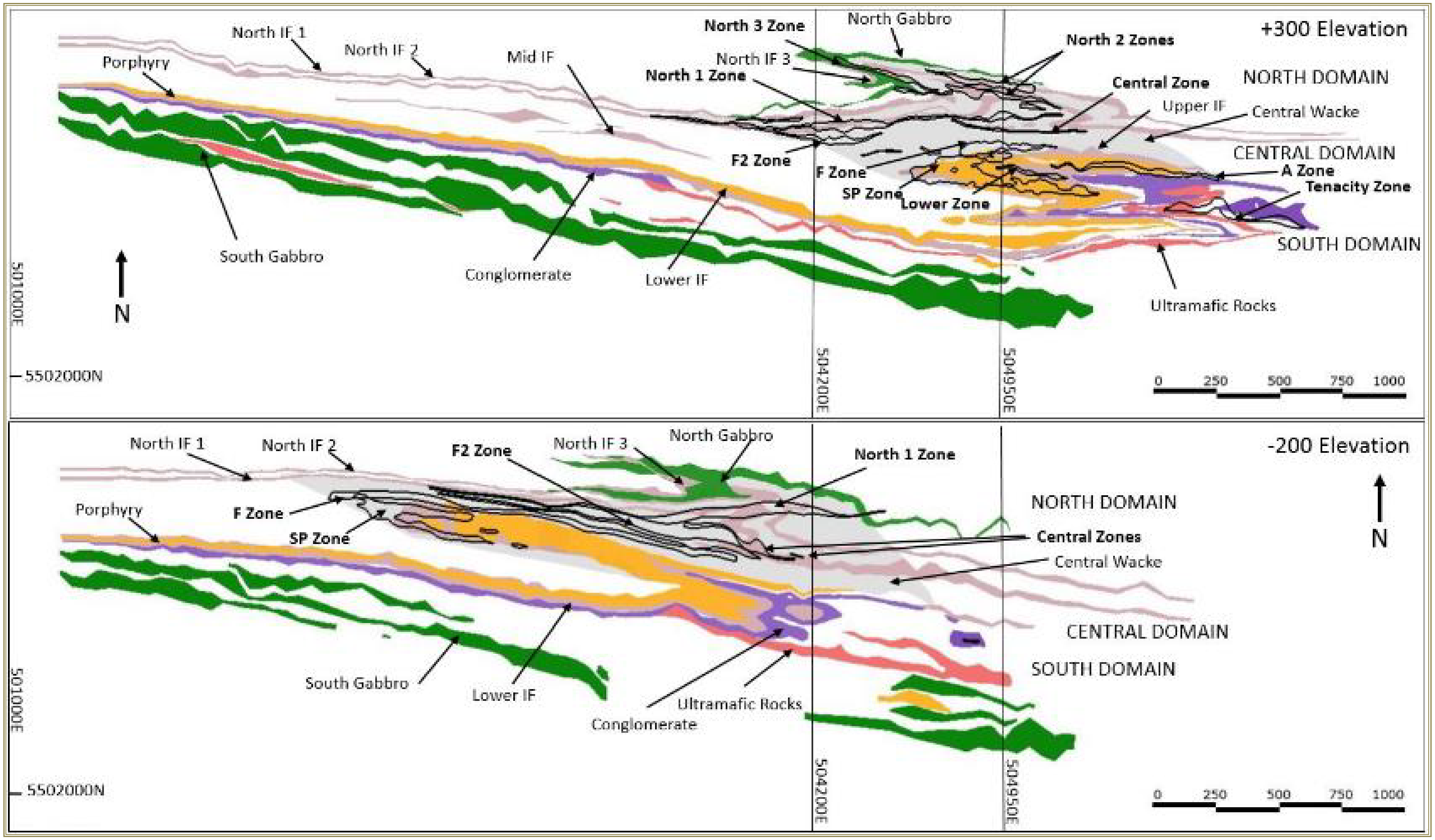
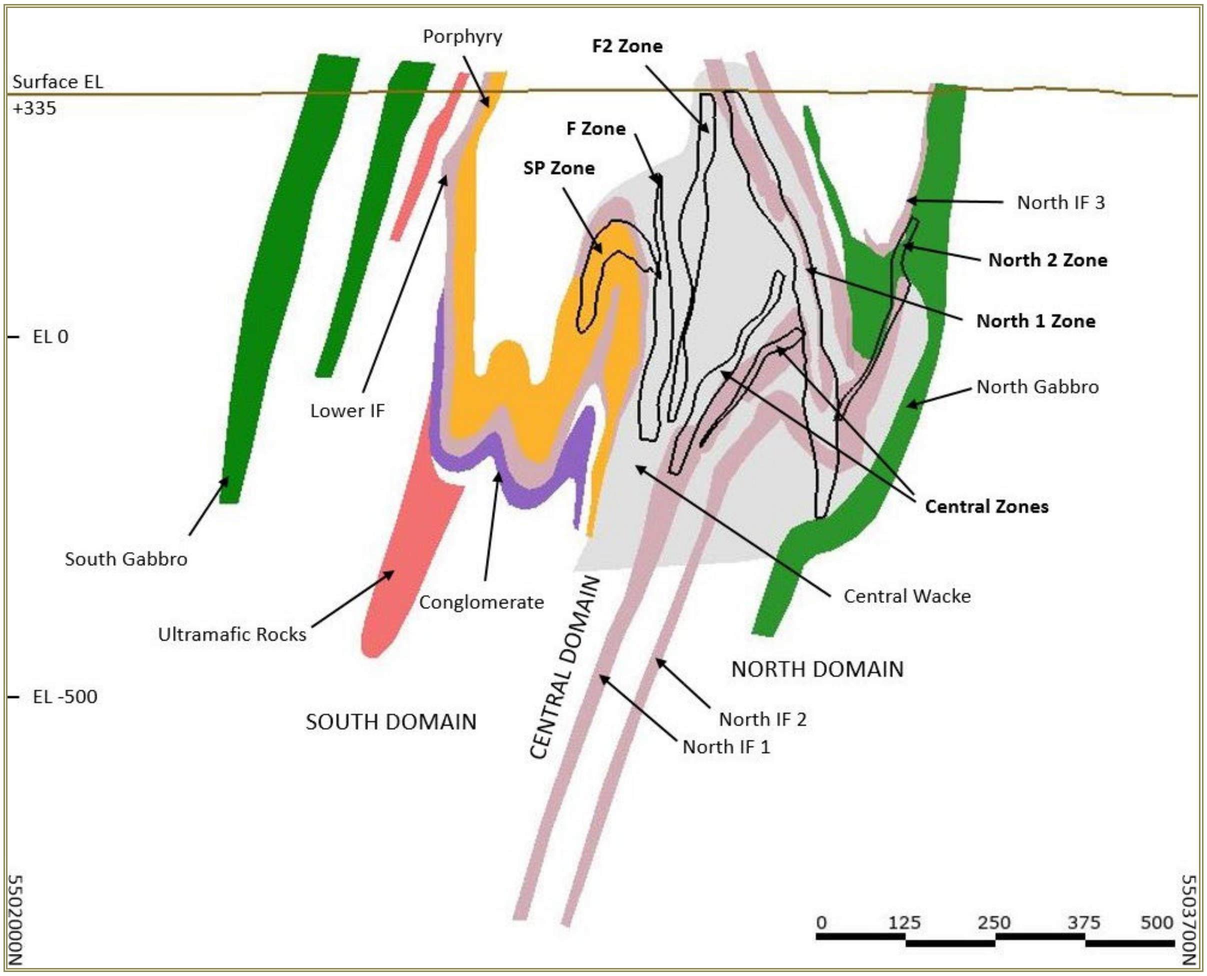
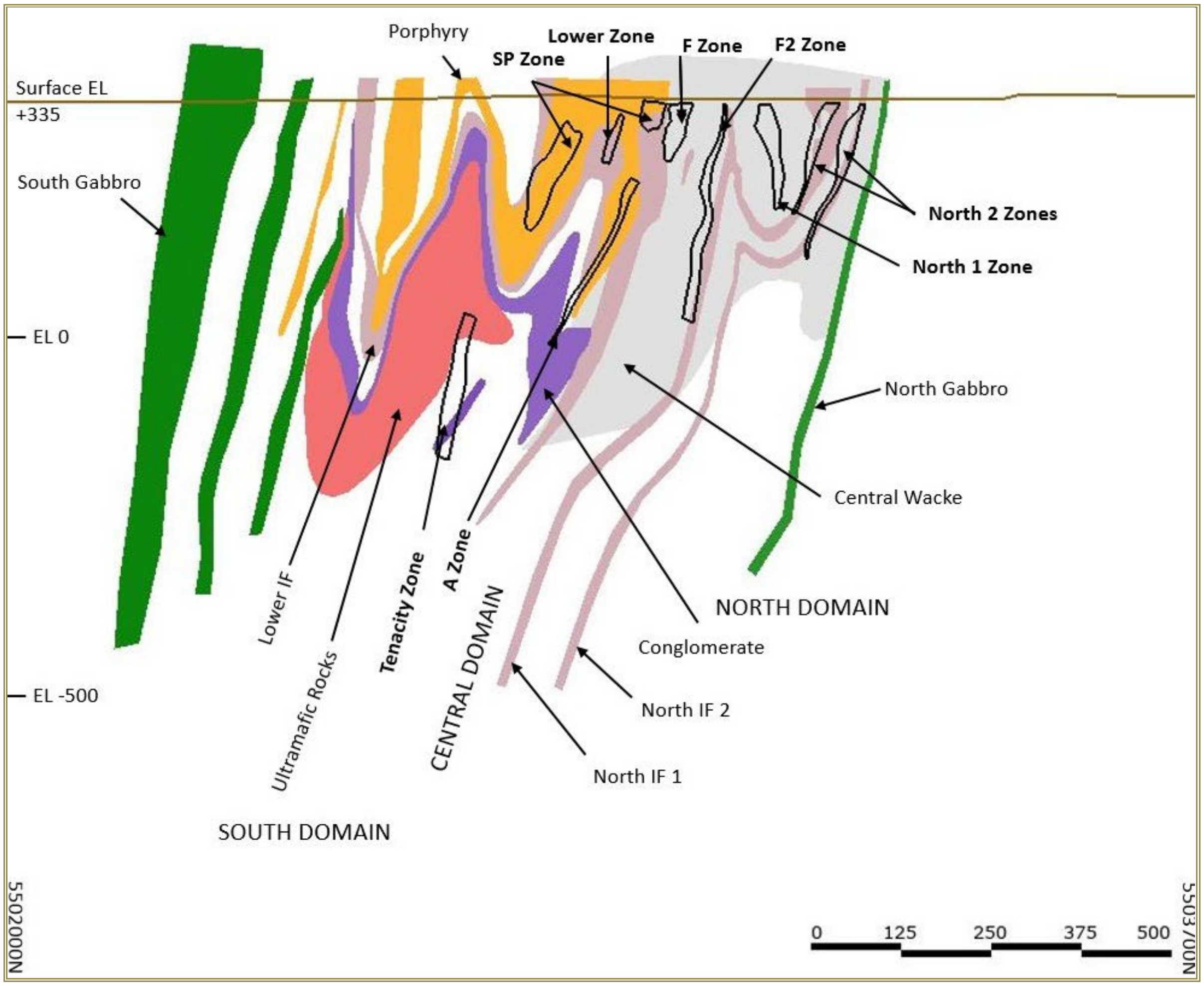


Figure 7-9: Plan View of Hardrock Litho-Structural Model showing Mineralized Zones at Elevations 300 m and −200 m (Projection: UTM Zone 16; NAD 83)



Note: Cross-section 4200, looking west.

Figure 7-10: Litho-Structural Model of Hardrock Showing Various Mineralized Zones



Note: Cross-section 4950, looking west.

Figure 7-11: Litho-Structural Model of Hardrock Showing Various Mineralized Zones

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 is 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. This scattered mineralization has not been defined for the most part.

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 it shows no evidence of mineralization.

Figure 7-12 and Figure 7-13 were taken from the pit walls in August 2024 and show the various relationships between rock types and the structural deformations.

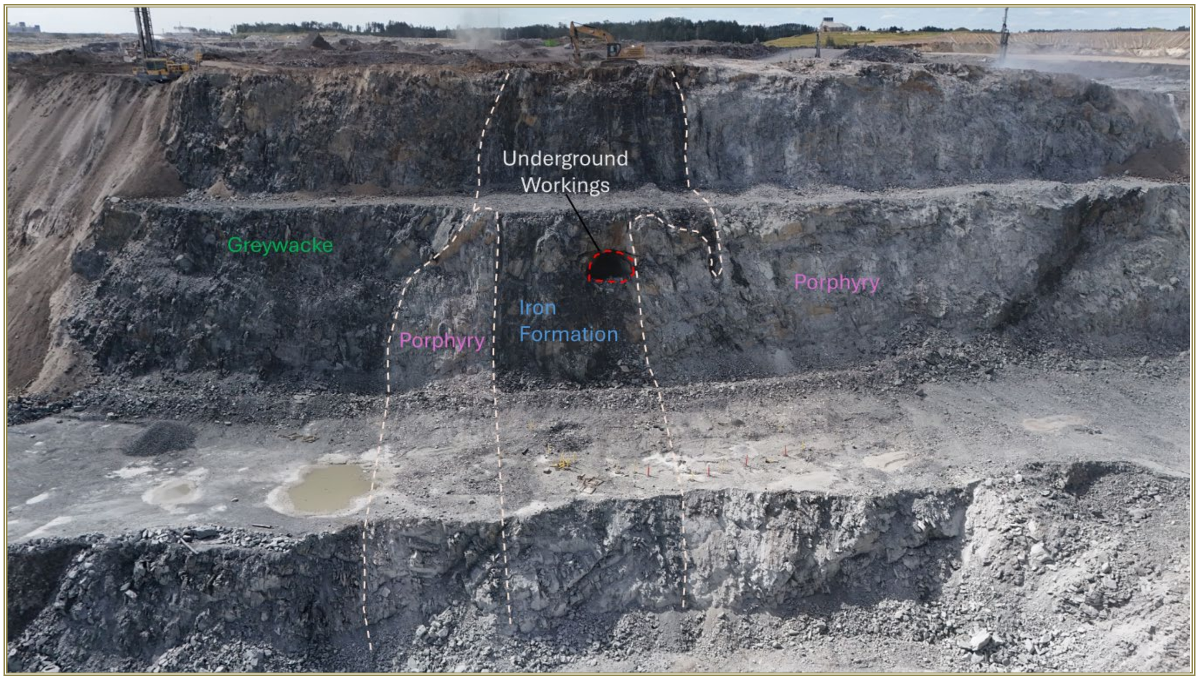


Figure 7-12: Greenstone Mine, East Pit at West Wall (Porphyry Hill)—Historical Underground Workings Visible Along Deformed Iron Formation-Porphyry Contact

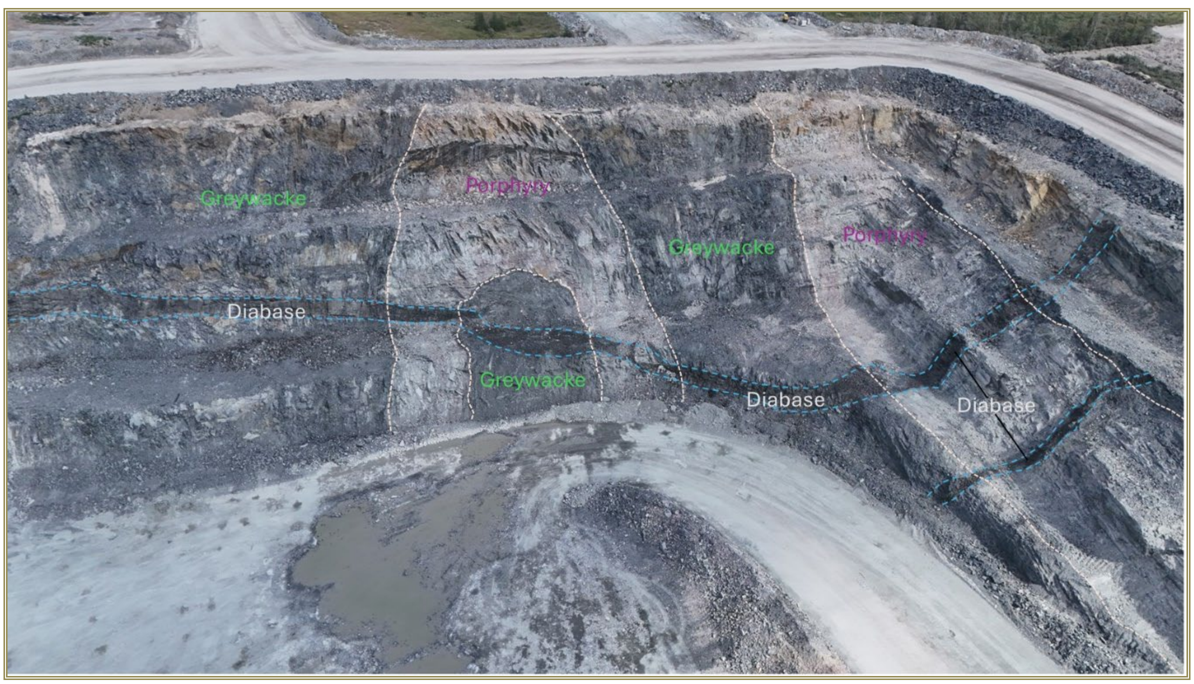


Figure 7-13: Greenstone Mine, East Pit at Southeast Corner—Folded Porphyry and Greywacke Units Crosscut by Younger Diabase Dikes

7.4.2 Style of Gold Mineralization

The following discussion on the style of gold mineralization was modified after 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 constitute 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/t Au are attainable from some stringers. Overall, zones having average grades of 4 g/t Au, 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 sulphide replacement-type mineralization, they appear to show an affinity towards rocks with higher iron content (Figure 7-14). 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 iron 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 sulphide having replaced the 0.5% to 1% disseminated hematite content noted in the less altered, hematite-stained porphyry.

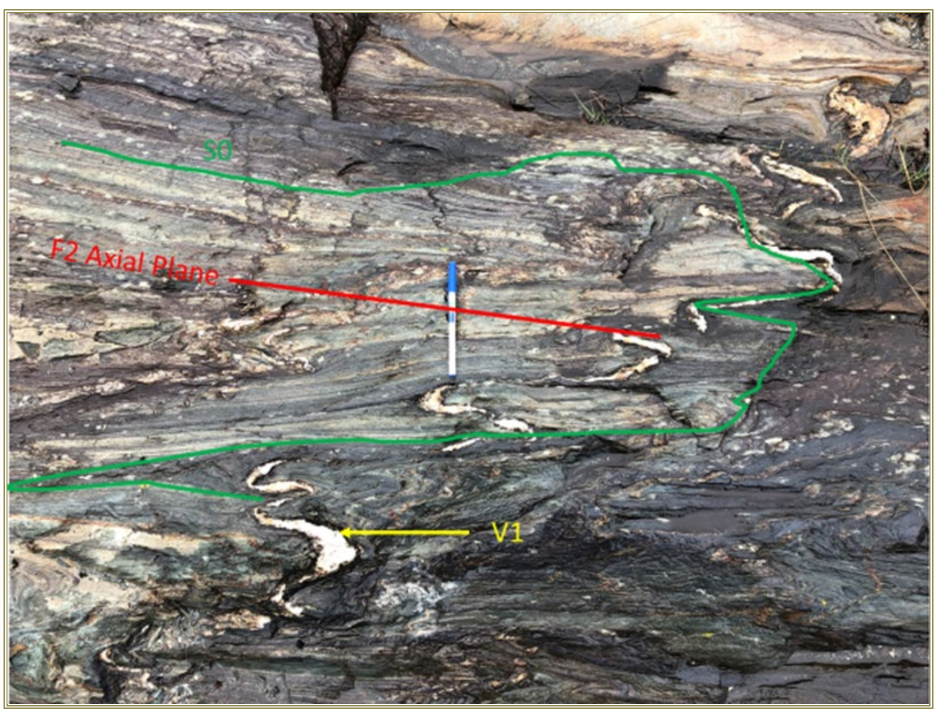


Figure 7-14: Deformed Quartz-Carbonate Stringers in BIF, Headframe Outcrop

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 (Figure 7-15. An understanding of this deformation is critical in determining which drill-hole extrapolations have the best probability of intersecting mineralization.

Figure 7-15: Folded Quartz-Carbonate Veins within Altered Quartz-Porphyry, Porphyry Hill

Sulphide Replacement Mineralization

Zones that are categorized as sulphide 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 iron oxide at the margins of quartz veins, within the hinge zones of folded BIFs. The auriferous sulphide 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/t Au or greater.

7.4.3 Mineralization by Zone

The following descriptions of mineralization modified 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 for Hardrock Mine

	2024 Name	Former Names	Historical Name	Description
North Domain	North 1 Zone	High-Grade North Zone North Zone	North Zone	Iron formation sulphide replacement
	North 2 Zone		n/a	Iron formation sulphide replacement
	North 3 Zone	North Wall Zone	n/a	Iron formation sulphide replacement
Central Domain	F Zone	F Zone	F Zone	Quartz-carbonate stringers in greywacke
	F2 Zone	Fortune (F2) Zone	n/a	Quartz-carbonate stringers in greywacke
	Central Zone	n/a	n/a	Quartz-carbonate stringers in greywacke
South Domain	Tenacity Zone	Tenacity Zone	B Zone	Quartz-carbonate stringers in greywacke and conglomerate
	SP2 Zone	SP Zone	n/a	Quartz-carbonate stringers in greywacke and minor Iron formation sulphide replacement
	SP Zone		South Zone/Trench Zone	Quartz-carbonate stringers in porphyry and greywacke and minor Iron formation sulphide replacement
	Lower Zone	P Zone	P Zone	Quartz-carbonate stringers
	A Zone	A Zone	A Zone	Quartz-carbonate stringers in greywacke and lesser porphyry

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 sulphides, 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 is 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 resources.

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 sulphides 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 sulphide 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. Sulphide 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 Mine Property Deposits (Brookbank, Key Lake, and Kailey)

The following modified from the Technical Report prepared by GMS on January 26, 2021.

7.5.1 Regional Geological Setting

The regional geological setting described in Section 7.1 for the Greenstone Mine, and summarized in Table 7-1, is applicable to the Greenstone Gold Property as a whole (formerly the Trans-Canada Property), which includes the Brookbank, Key Lake, and Kailey properties.

7.5.2 Brookbank Property Local Geology

The Brookbank property is underlain by metavolcanic and metasedimentary rocks that trend predominantly east–west and dip steeply to vertically to the south. Metavolcanic 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 metavolcanic rocks are locally intruded by quartz-feldspar porphyritic dykes. See Figure 7-16 for a geological map of the prospect.

Mafic metavolcanic rocks are fault-bounded against domains of metasedimentary rocks. The northern domain consists of a polymictic conglomerate with pebble- to boulder-sized, rounded to subrounded 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 an affinity with Timiskaming Formation conglomerates in the Timmins (Porcupine) Mining District.

Metasedimentary domains south of Windigokan Lake also contain polymictic conglomerate, feldspathic and quartzose sandstone, 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 metavolcanic 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 metavolcanic 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 metavolcanic and metasedimentary rocks along the western boundary of the property in Sandra Township and along the western boundary of Irwin Township.

The Brookbank property 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 7-16). The 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.

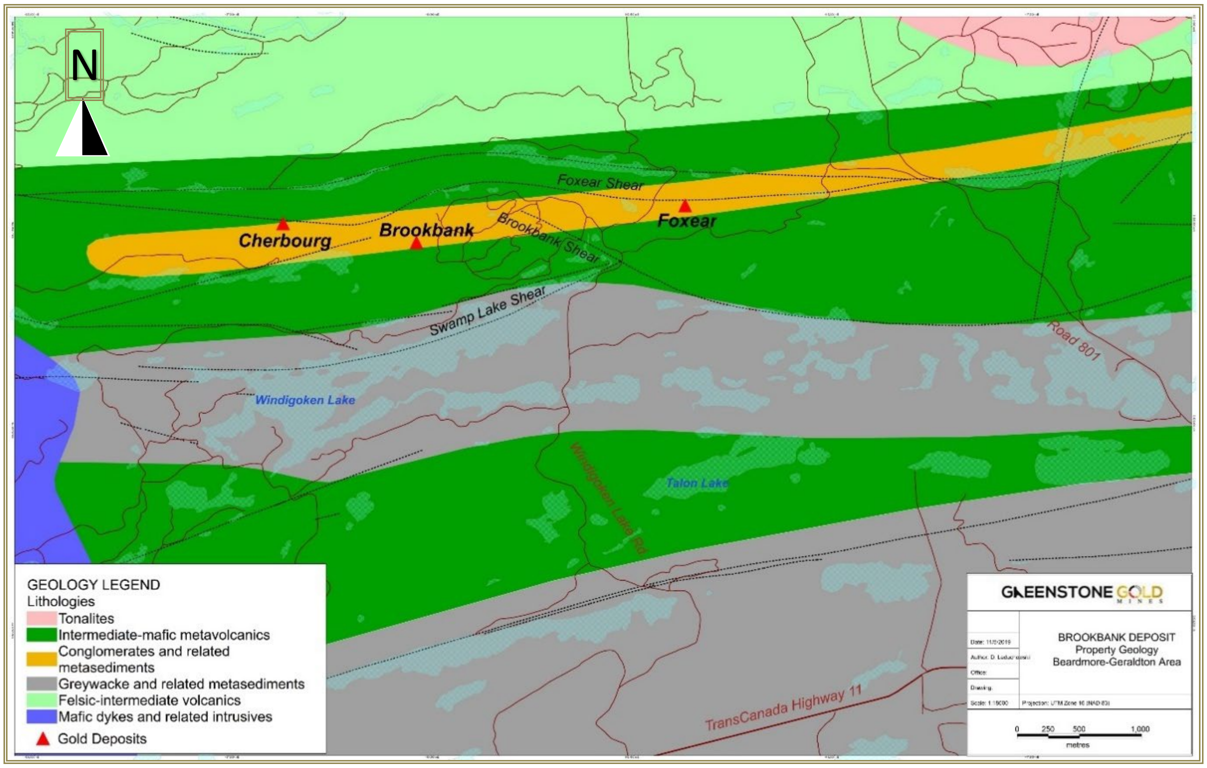


Figure 7-16: Brookbank Property Geology

7.5.3 Brookbank Property Mineralization

The 6.5 km-long Brookbank shear zone hosts the Brookbank deposit and numerous other targets including Cherbourg and Foxear (Figure 7-16). 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.

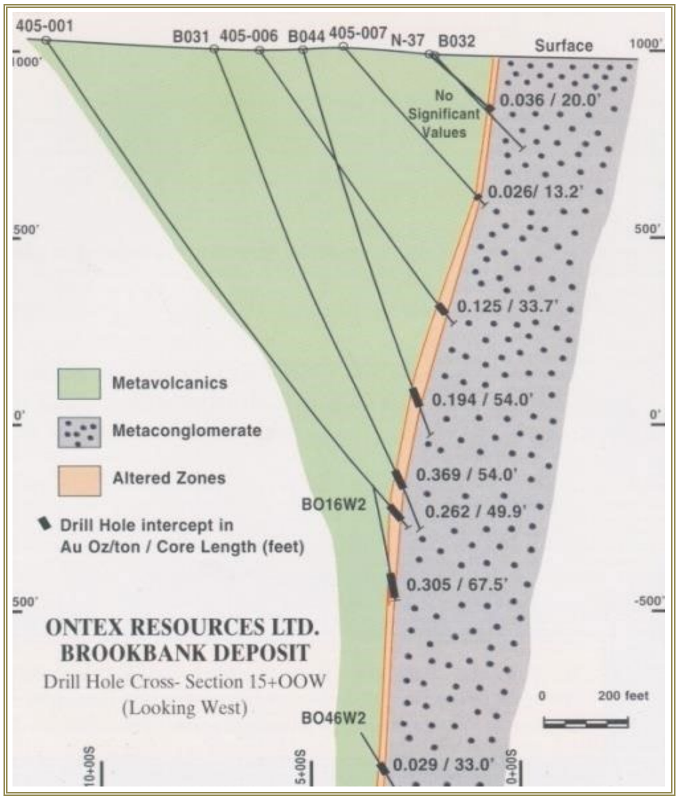
The zones of mineralization at Brookbank, Cherbourg, and Foxear occur within one of several bands of intense deformation and hydrothermal alteration at or near the contact between domains of mafic flows and polymictic conglomerates. Hydrothermal alteration accompanying the mineralization consists of silicification, carbonatization, sericitization, chloritization, hematization, and sulphidation (Figure 7-17). This alteration is commonly marginal to the mineralized quartz-carbonate veins, fractures, and stockworks, and may exceed 50 m wide locally.

Source: Micon (2013).

Figure 7-17: Exposure of the Brookbank Mineralized Corridor showing Intense Hydrothermal Alteration

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 elongated 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/t Au assays. Gold values are highest in the quartz–carbonate veinlets/stringers.

A cross-section view of the Brookbank deposit is presented in Figure 7-18. Sulphide mineralization (pyrite and minor chalcopyrite) is also present within the sheared host rock and quartz veinlets.



Source: After Ontex (2009).

Figure 7-18: Brookbank Deposit—Cross-Section 1500 W

7.5.4 Key Lake Property Local Geology

The Key Lake Property is located within the Beardmore–Geraldton Greenstone Belt of the Wabigoon Subprovince. 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 metavolcanic sub-belt.

Metagreywacke 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-sized clasts may be a matrix-supported metaconglomerate or a vitric lapilli tuff. Magnetite-rich argillite occurs to the north and south of the mineralized zone. BIFs 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 are a useful marker horizon. Gabbro and diorite dykes occur in some areas, and Proterozoic diabase dykes crosscut all other rock units.

Figure 7-19 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, 2019a).

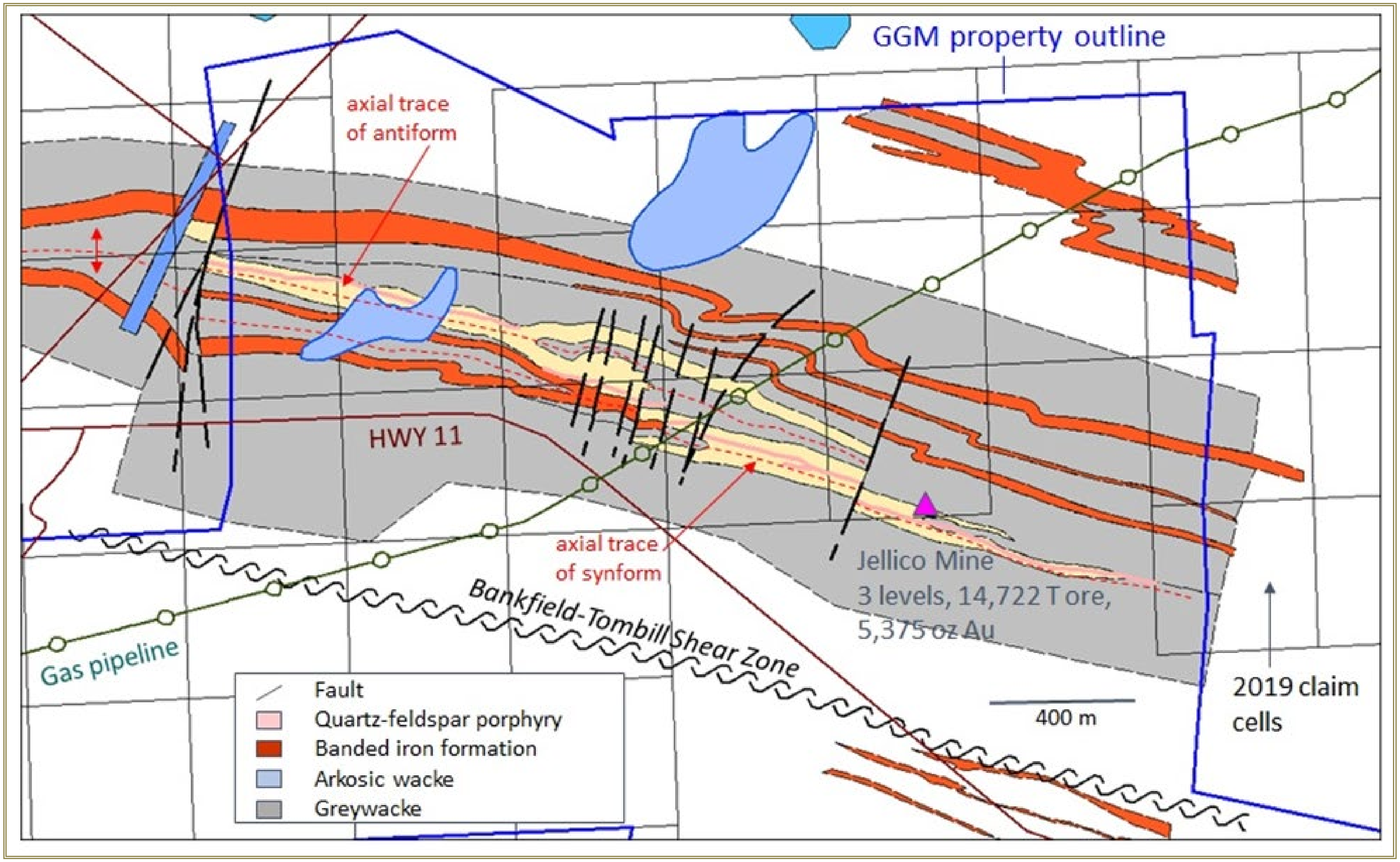


Figure 7-19: Generalized Geology of the Key Lake Property

7.5.5 Key Lake Property Mineralization

Gold occurs in altered metagreywacke (arkose), felsic dykes, and thin veins cutting these rocks. Gold-bearing altered rocks typically have more than trace amounts of pyrite 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 or very fine-grained arsenopyrite. Visible gold occurs in veins in both metagreywacke 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 and ankerite alteration was detected by staining (Gasparetto & Stevenson, 1996).

Zones of greenish, brownish, and rarely yellowish sericitization are more limited, and envelop 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 7-20 and Figure 7-21.

Figure 7-20: Example of Fracture-Controlled Pyrite Mineralization in Sericite-Silica Arkosic Wacke, 0.54 ppm Au

Figure 7-21: Example of Sericite-Altered Quartz-Feldspar Porphyry, 7.75 ppm Au

7.5.6 Kailey Project Local Geology

The Kailey deposit is located at the former Little Long Lac gold mine, about 1.7 km north of the Hardrock Mineralized Corridor. The local geological setting described in Section 7.2 for the Geraldton area is applicable to the Kailey property. It lies within a broad synclinal belt of greywacke, slates, conglomerates, and iron formations 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.

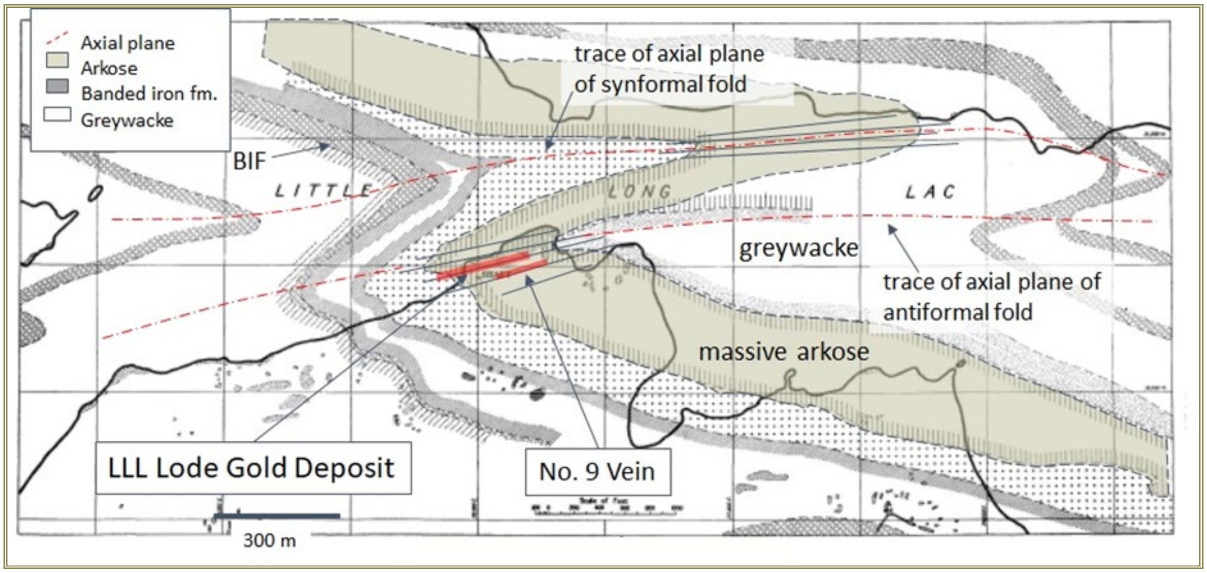


Figure 7-22: Kailey Property Geology

Deposit-forming quartz veining is localized along an N70E-striking, sub-vertical 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. The closure of this antiformal fold is obscured by the waters of Barton Bay and 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 through-going fractures (Burk, 2019b).

An isometric view of the various mineralization and host units is shown in Figure 7-23.

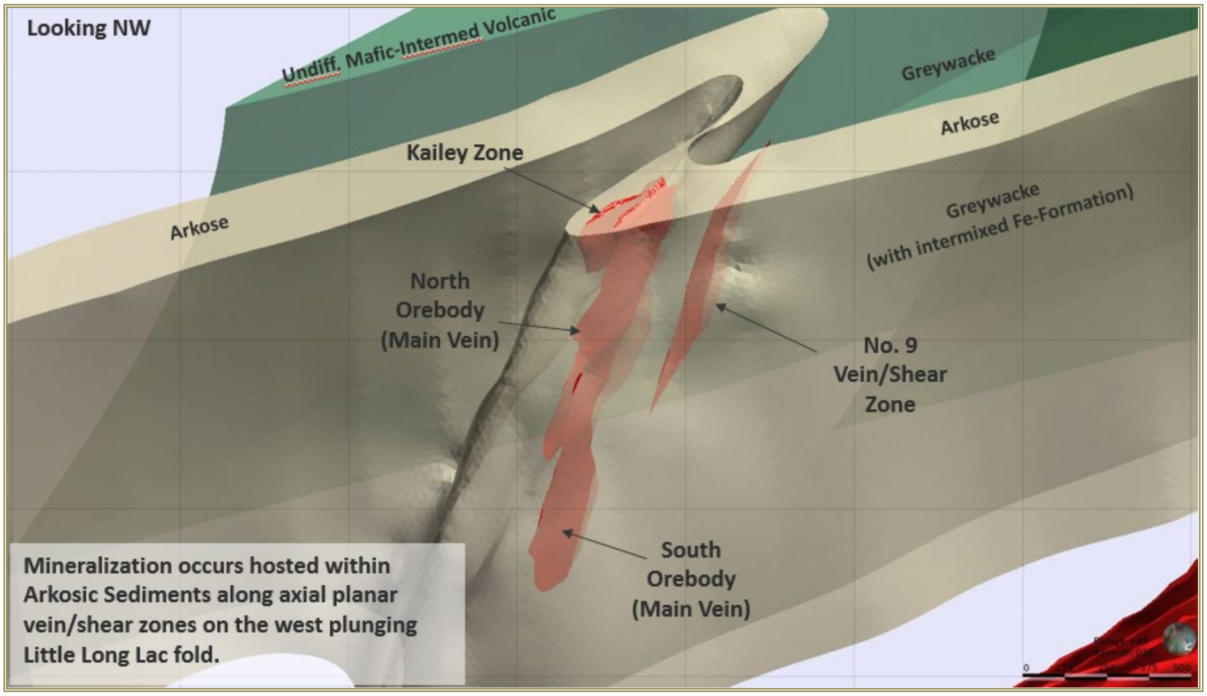


Figure 7-23: Isometric View of the Kailey, North, South, and No. 9 Vein Mineralization with the Folded Arkosic Host Unit

7.5.7 Kailey Property Mineralization

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

The Kailey Zone is a shallow and low-grade mineralized domain north of the 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) containing a lower-grade halo.

The No. 9 Zone is located approximately 150 m south of the Main Vein. 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 by predominantly pyrite, arsenopyrite, and minor base metal sulphides. Scheelite is also present in varying degrees.

8 DEPOSIT TYPES

This section is an update from the 2021 Report, issued January 26, 2021. Some comments have been added to cover available information through June 2024.

8.1 Hydrothermal Iron Formation-Hosted Gold Deposits

The gold ore bodies at the MacLeod–Cockshutt mine are one of the type examples for BIF-hosted gold deposits. Other well known Canadian examples include the Central Patricia mine, Ontario, and the Lupin mine, Northwest Territories. 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. However, it should be noted that the bulk of the iron formation-hosted mineralization at Hardrock 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, potassium, carbon dioxide, sulphur, ± arsenic, and the depletion of calcium and magnesium. This is manifested in the rock as quartz-carbonate veining, silicification, or semi-massive to massive sulphides (pyrite, pyrrhotite ± arsenopyrite) surrounded by a halo of sericite–carbonate–pyrite alteration.

The Hardrock Project recognizes and presents the following subtype descriptions copied from the 2016 feasibility study report, which quotes from Kerswill (1993).

8.1.1 Non-Stratiform Type

In non-stratiform deposits, gold is restricted to late structures (quartz veins or shear zones) or sheared sulphide BIF immediately adjacent to such structures. Mineralization is confined to discrete, commonly small shoots separated by barren (gold- and sulphide-poor) oxide BIF. Mineralized rocks are generally less deformed than associated rocks. Iron-sulphide minerals are, in many cases, relatively undeformed and unmetamorphosed. Pyrite 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 typical of that associated with mesothermal vein gold deposits. Mineralization is relatively silver-poor, and gold grains generally have gold to silver ratios of >8.0. Non-stratiform deposits are relatively common, typically small, and difficult to evaluate and mine compared with stratiform deposits.

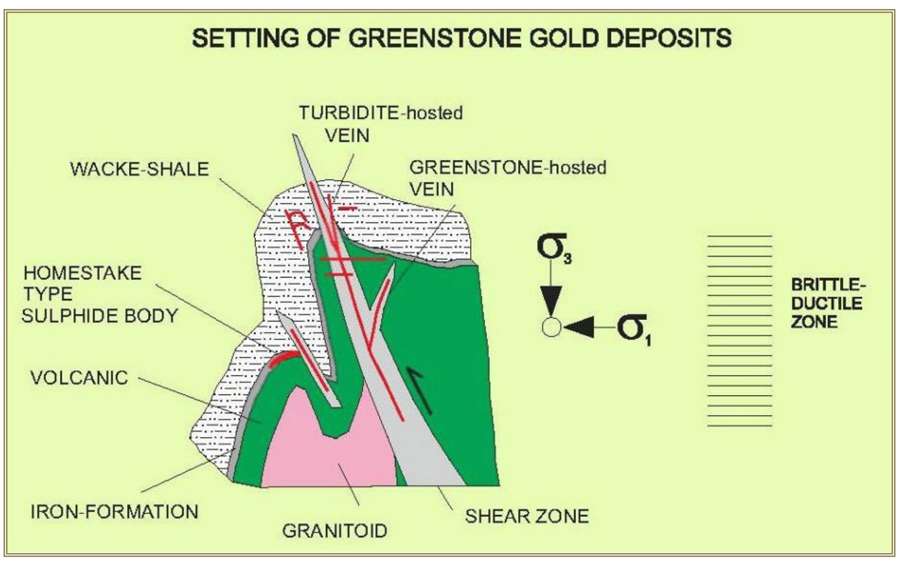
Non-stratiform deposits contain sulphide-rich alteration zones immediately adjacent to late structures and are similar to mesothermal vein-type gold deposits. Late quartz veins or shear zones are present in most known BIF-hosted gold deposits. The distributions of gold-bearing veins and sulphide-rich 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 sulphides and quartz occur in a folded series of greywacke and iron formations in the Hard Rock and MacLeod–Cockshutt mines (Horwood & Pye, 1951). These massive replacement lenses (up to 65% sulphides) 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 sulphides, 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 sulphide replacement of the less-competent wacke laminae in the iron formation.

At the Hardrock Project, examples of this type of deposit are 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 within deformed terranes of ancient to recent greenstone belts commonly metamorphosed at greenschist facies (Dubé & 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 or orogenic gold deposits. 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.



Source: Poulsen et al. (2000).

Figure 8-1: Setting of Greenstone-Hosted Gold Deposits

The crustal-scale faults are thought to represent the main hydrothermal pathways toward higher crustal levels. However, the deposits are spatially and genetically associated with higher-order compressional reverse-oblique to oblique brittle-ductile high-angle shear zones commonly 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.

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 or conjugate sets of veins, breccias, stockworks and associated structures (Dubé et al., 1989; Hodgson, 1989; Robert et al., 1994; Robert & Poulsen, 2001).

Economic grade mineralization also occurs as disseminated sulphides 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 or competent rock type such as iron-rich gabbro (geometric ore shoot); or 2) the slip vector of the controlling structures (e.g., 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 & 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 & Gosselin, 2007). At the deposit scale, the nature, distribution and intensity of the wall-rock alteration are 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 sulphidation of the immediate vein selvages (mainly pyrite and less commonly arsenopyrite).

The main gangue minerals are quartz and carbonate, with variable amounts of white micas, chlorite, scheelite, and tourmaline. The sulphide 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 & Gosselin, 2007).

The structurally controlled, high-grade veins spatially related to the Hard Rock Porphyry in the Hard Rock and MacLeod–Cockshutt mines are like quartz–carbonate–sericite veins that host gold within many gold camps in Ontario (e.g., 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), sulphides (pyrite, pyrrhotite, arsenopyrite, and chalcopyrite) and tourmaline are found to be associated with the quartz. Zones A through H were of this type (Horwood & 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 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 ore body of some 10,000,000 tonnes at 0.15 oz/ton Au (Macdonald, 1983). The F Zone produced the bulk of the tonnage that came from these mines from the 1950s to 1970.

Several DDHs on the current drilling 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.

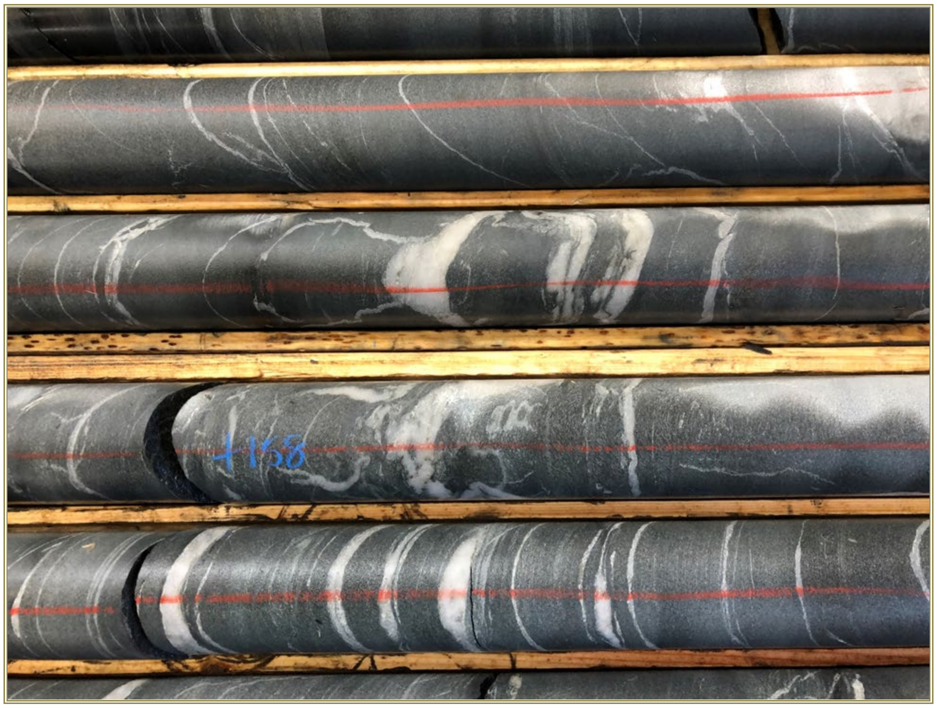


Figure 8-2: Sheeted Quartz-Carbonate Veins Hosted in Greywacke, DDH-19-54

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 in areas of high strain and deformation, with brittle structures providing a pathway for and 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 and stringers, fractures, or stockworks associated with hydrothermal alteration (Figure 8-3).

Source: Micon (2013).

Figure 8-3: Exposure of the Brookbank Deposit Quartz-Carbonate Veining and Fractures

Considering the deposit model discussed above, previous and current exploration activities have 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 mineralization.

8.2.3 Kailey

Kailey is 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 mainly parallel quartz veins and stringers within fracture zones in massive arkose. For the most part, the sulphides 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 narrow selvages and occur at centimetre-scale. The veins are highly sheared and sericitized arkose with small amounts of disseminated sulphides, 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 occurring 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 favourable sediments for deposits of the first type are the belt of Timiskaming sediments of massive and thickly bedded greywacke or arkose that lies between the two northern bands of iron formation. The veins of the Little Long Lac mine are within this belt and are almost exclusively hosted within the Arkose unit.

9 EXPLORATION

This section is an update from the 2021 Report, issued January 26, 2021. Some comments have been added to cover available information through June 2024.

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 Hard Rock Number 1 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. 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 was identified on both the MacLeod–Cockshutt and Hard Rock Properties; it was a low-grade, large-tonnage ore body in greywacke.

Production on the Mosher Long Lac Mine began in 1962 (west of, and immediately down-plunge of the same mineralized zones exploited in the MacLeod–Cockshutt Mine); 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 at 4.9 g /t Au) 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 better understand and assess the open pit potential on the Property.

In 2007, Premier began assembling the current property. The results of 1,629 drill holes were included in the 2016 feasibility study.

In February 2015, Premier and Centerra Gold Inc. formed a definitive 50/50 partnership for the 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 survey 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. The results of this work are detailed in the 2016 technical report.

In 2018, 405 RC holes totalling 19,995 m and 62 blast holes totalling 535 m were drilled to provide further definition of near-surface gold mineralization in five 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.

A significant drilling campaign was conducted during the winter of 2021/2022 targeting inferred blocks at depth and to the east of the pit design outlined in the 2019 feasibility study. Fifty-six DDHs totalling 15,421 m and 67 RCGC holes totalling 4,189 m were drilled.

From September 2022 to the end of June 2024, a total of 16,049 m of RCGC were drilled on the Hardrock deposit. These programs focused on areas that were under-drilled within the original planned Starter pit within the redesigned Phase 1B pit, and to fully define the North Zone voids before mining. None of this RCGC is captured in the 2024 MREs.

9.2 Other Properties

Since Premier started its acquisition of the other projects within the Hardrock Property in 2007, most of the exploration work has been focused on diamond drilling. The details of any historical exploration program performed before Premier owned the property are summarized in Section 8.

9.2.1 Brookbank

Between 2012 and 2013, a two-hole drill program was completed on the Brookbank property 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 DDHs 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 and re-sampling of DDHs.

9.2.2 Kailey and Key Lake

2011 exploration activities focused on in-fill drilling proximal to the historical 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 Premier and GGM completed are described in Section 10.

Very little exploration fieldwork has been conducted on the Brookbank, Key Lake, Kailey, or other satellite properties since the 2021 Report. Work during this time has consisted mainly of data compilation and selected re-logging of DDHs.

10 DRILLING

10.1 Hardrock

Information in this section was provided by the GGM exploration team which has conducted the drilling programs in 2018, 2019, and up to September 2022 under the supervision of G Mining Services Inc. (GMS). GMS has reviewed this information and compiled the following section regarding drilling practices at the Project.

Over the years, different drill-core diameters have been used on the Hardrock deposit. Recent drill holes at the Greenstone Mine are mostly NQ diameter. Table 10-1 summarizes the core diameter used in different years, with recent reverse circulation grade control (RCGC) drilling also shown.

Table 10-1: Number of Drill Holes and RCGC per Core Size per Year

Year Drilled	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	RCGC
2019	76	RCGC
2019	54	NQ
2021–Sep 2022	67	RCGC
2021–Sep 2022	56	NQ
Sep 2022–Jun 2024	491	RCGC
Unknown	29	Unknown
Total	**2,778**	

GCRC holes drilled since September 2022 are not included in the current MRE.

10.1.1 Collar Locations, Orientations, and Downhole Surveys

Collar locations for the drill holes on the Hardrock Project were determined using a cut grid or a hand-held Global Positioning System (GPS). Following completion, the collars were located, depending on the years drilled, using either a GPS receiver or a Trimble, and since 2014 the more-precise Trimble RTK survey instrument. In all, 55% of the holes drilled prior to 2013 have been surveyed using a hand-held GPS receiver.

Whenever it was possible, casings were left in the ground after drilling. In 2014, 536 drill holes for which casing was found were resurveyed using a Trimble RTK. The Trimble resurvey data replaced the original GPS survey data, improving the precision of the collar location for 30% of the drill holes in the database.

Once holes were drilled, the drill-hole azimuth and precise UTM coordinates were determined by placing a Reflex Astronomic Positioning System (APS) unit on the drill casing. The downhole dip and drill-hole orientations were surveyed using a gyroscope unit (REFLEX Gyro) (UTM, Zone 16N; NAD 83) to record the easting, northing, and elevation of the drill collars.

Since 2018, the site surveyor and geologists spotted the RCGC and blastholes using a Trimble RTK system 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 APS™. Downhole 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), or 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 to 1.5 m. In clearly mineralized zones where visible gold is present, the geologist will place a piece of coloured 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 to Activation Laboratories Ltd. (Actlabs Geraldton) facility in Geraldton.

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 fibre-optic microscope. Information on lithology, alteration, and mineralization was recorded in DataShed by the on-site geologists.

The core, RCGC duplicate samples, assay pulps, and rejects are stored off-site at the Magnet Mine.

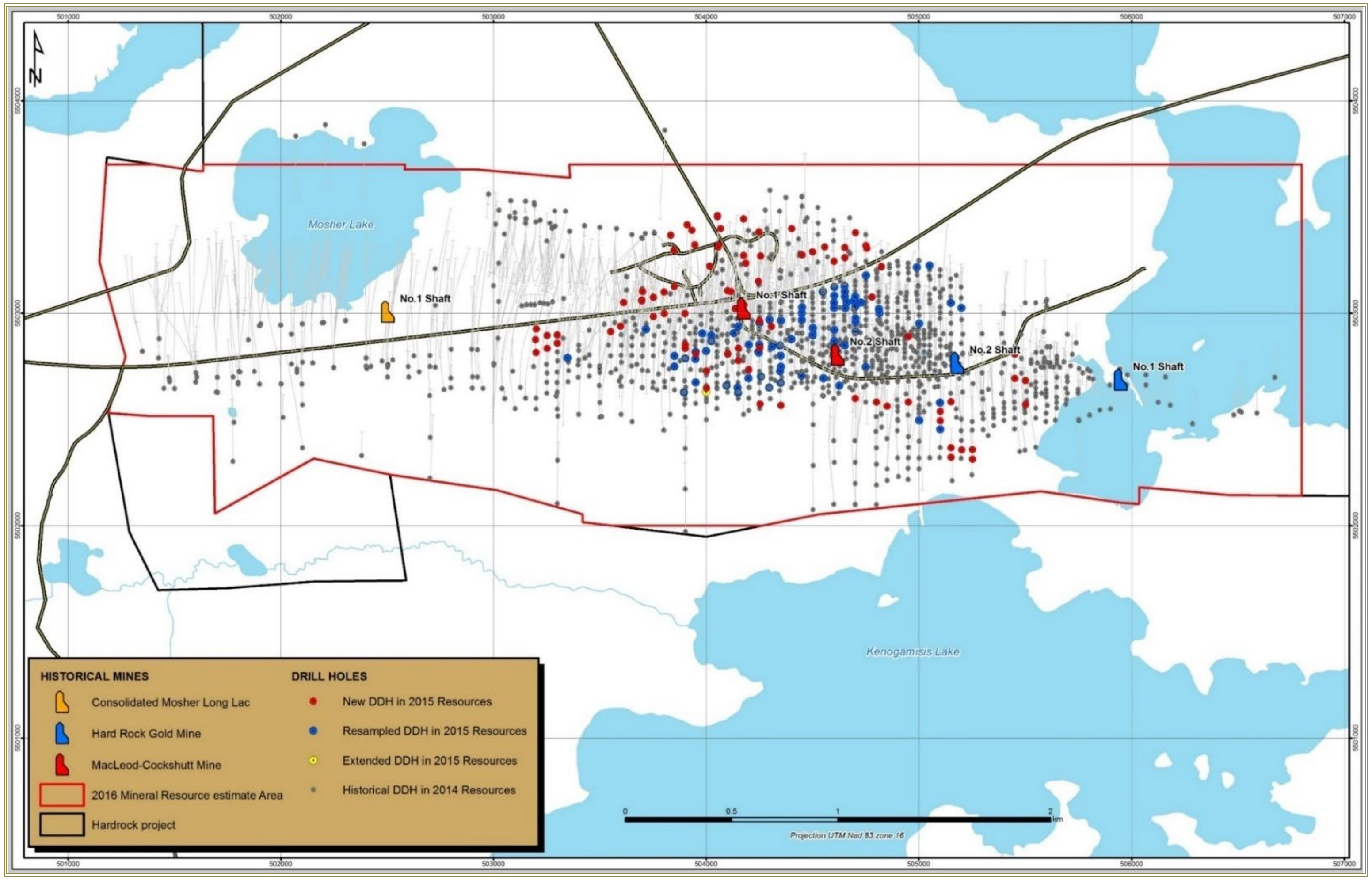
10.1.3 Drilling and Re-Sampling Included in the 2016 Mineral Resource Estimate

Between May 26, 2014, and November 18, 2015, GGM added 157 DDHs on the Hardrock deposit for a total of 54,027 m. One DDH—MM043—included in the 2014 MRE was also deepened, from 456 to 655 m, representing a total of 199 m of new metreage.

Seventy-nine historical DDHs 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.

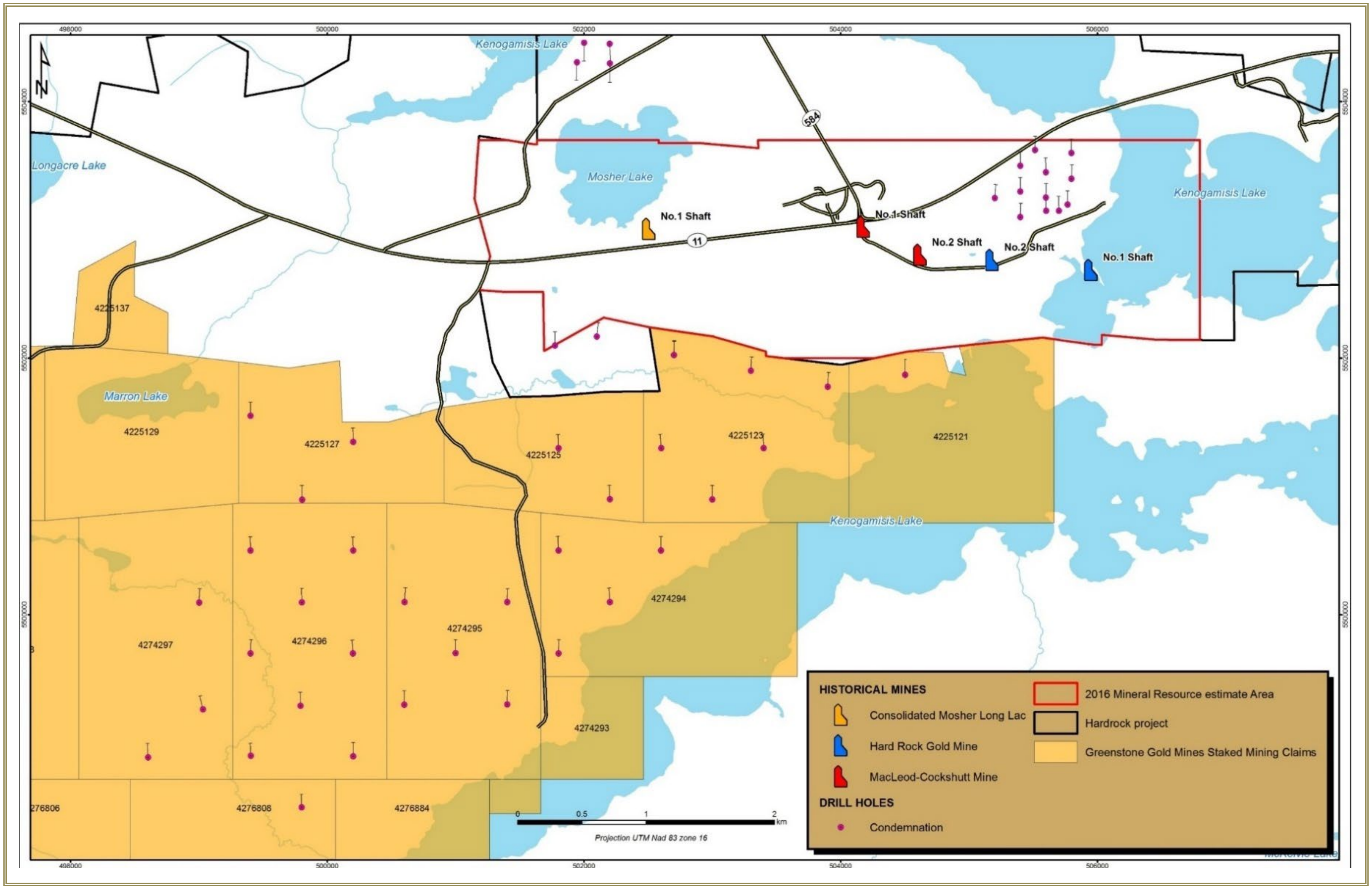
Figure 10-1 shows the locations of the drill holes included in the 2016 MRE. The new drill holes (red), the re-sampled DDHs (blue) and extended drill holes (yellow) that are included in the 2016 MRE are presented in Figure 10-1.

Figure 10-2 shows the locations of the condemnation drill holes drilled in the area of the Hardrock deposit. A total of 55 condemnation DDHs totalling 8,512 m were drilled by GGM.



Source: Innovexplo (2015) with modifications by GGM (2016).

Figure 10-1: Location of Drill Holes used in the 2016 Mineral Resource Estimate, prior to the 2018 and 2019 Drilling Programs



Source: GGM (2016).

Figure 10-2: Location of Condemnation Drill Holes in the Hardrock Deposit Area

10.1.4 2018 RC Grade Control and Blasthole Drill Program

The 2018 RCGC and down-the-hole (or blasthole) drilling campaigns were resource definition programs, designed to de-risk the project and 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 Down-the-Hole Drilling at Hardrock

Area	Zone	Lithology
Area 2	Headframe	Interbedded BIF and greywacke
Area 3	F	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	BIF, porphyry, and greywacke

From May 24, 2018, to September 6, 2018, 405 RCGC drill holes, totalling 19,995 m, were completed, on the Property. All RCGC drilling was completed by NPLH Drilling, in Timmins, Ontario. The program targeted five areas defined by their geographic and lithological properties (Table 10-2).

All RCGC drill holes were planned by Mr. Réjean Sirois of GMS and QP of this current MRE. RCGC holes were planned 10 m apart north–south and 20 m apart east–west. On average, the RCGC holes were oriented true north or south, and drilled with inclination −50° to a depth of 50 m. The results obtained from the RCGC drilling program confirmed the continuity of grade 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.

Additionally, Epiroc drilled 62 blastholes totalling 535 m. 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 (Table 10-2), and to test the performance and viability of blasthole drilling for the Hardrock deposit.

The blastholes were planned with a tighter spacing of approximately 6 m north–south and east–west. The blastholes were on average 10 m deep and drilled vertically. Figure 10-3 presents the surface plan of RCGC drilling and blastholes from 2018.

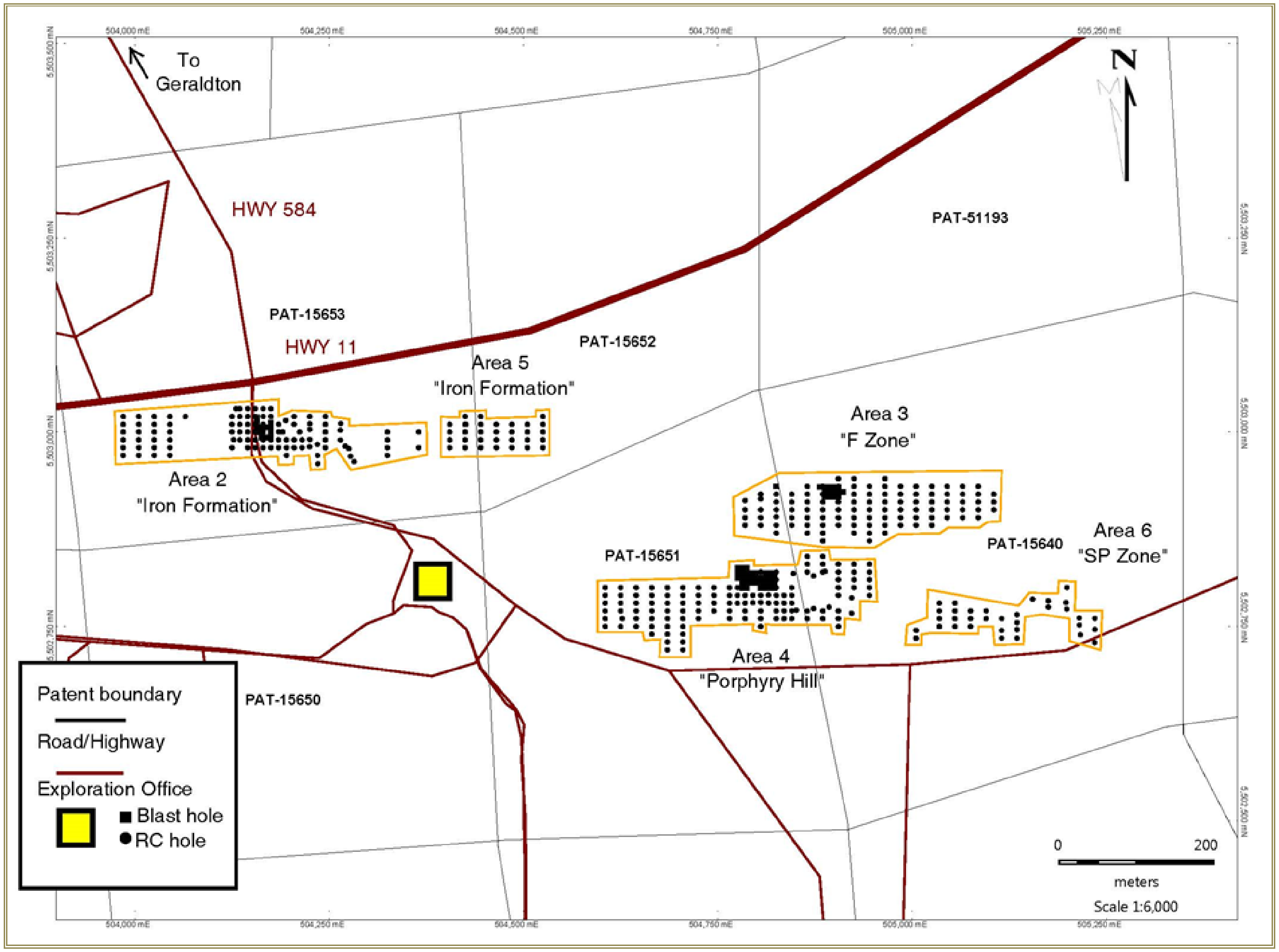


Figure 10-3: 2018 RCGC and Blasthole Drilling Locations

10.1.5 2019 Drill Program

The 2019 drilling program consisted of 76 RCGC drill holes totalling 5,946 m, of which 5,527 m were assayed, and 54 NQ-size DDHs for 12,108 m, of which 10,470 m were assayed. The first drill was mobilized on February 12 ,2019; drilling commenced February 19, 2019, and continued through April 25, 2019. Both RCGC and DDH drilling were contracted to NPLH Drilling. 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 north–south and 20 m east–west. On average, the RCGC holes were 100 m deep and had a dip of −50°, oriented true north or south. Figure 10-4 illustrates the RCGC and DDH programs performed by GGM in 2019 with the 2016 feasibility study pit design in the background for scale.

The 2019 drilling program outcomes are detailed below:

- RCGC drilling was spatially limited to the SP and F Zones to confirm grade continuity for benches 4 to 7.
- 70 m vertical (or 7 benches) were drilled at an average spacing of 20 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.

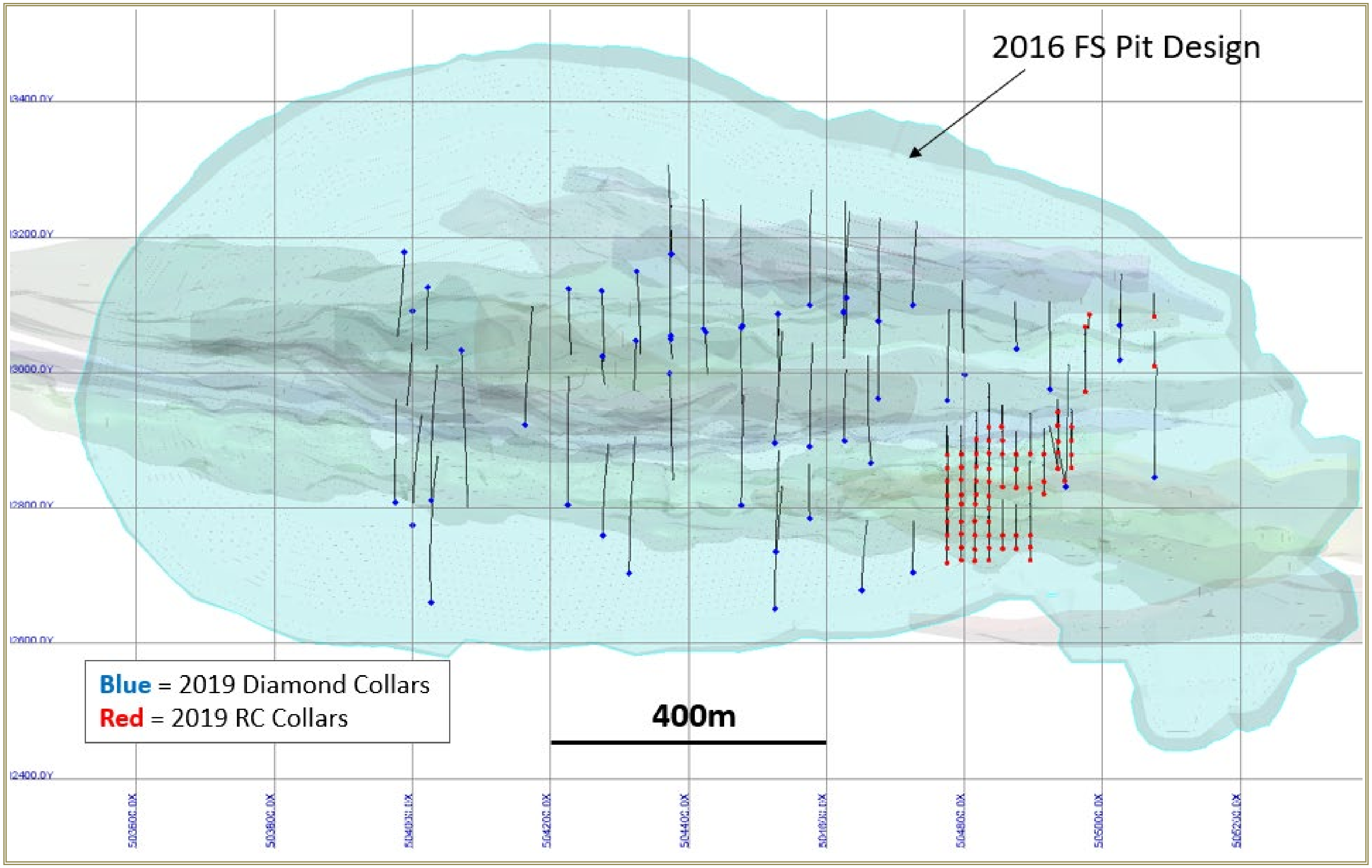


Figure 10-4: 2019 Drilling Programs

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 2021—September 2022 Drill Program

A significant drilling campaign was conducted during the winter of 2021/2022, targeting inferred blocks at depth and to the east of the pit design outlined in the 2019 feasibility study. Fifty-six diamond drill holes totalling 15,421 m, and 67 RC drill holes totalling 4,189 m were drilled.

G Mining Services Inc. (GMS) was retained to update the mineral resource block model for the Hardrock deposit, incorporating new drilling undertaken since the release of previous block model as part of the 2019 feasibility study. The scope of work for the 2022 block model update is described below:

- Incorporate new drilling undertaken in 2021 in the eastern portion of the deposit.
- Identify coherent zones of mineralization in the external grade shell domain to model manually to reduce the ratio of external grade shells to principal domains.
- Update capped gold-grade estimate using a similar approach as 2019 MRE.
- Update arsenic, sulphur, and total carbon model.
- Update void model using information obtained from recent drilling.
- Update lithology model, and use to recode bulk density.

The current MRE is based on these models. Small adjustments were made to the resource categorization using the latest surfaces. Those block models were filtered by the current topography, pit designs, whittle shells, and latest optimization parameters up to June 30, 2024.

10.1.7 September 2022–June 2024 Grade Control and Blasthole Drill Programs

Significant RCGC drilling campaigns were conducted since September 2022. The goal of these campaigns was to improve the final gold estimate of the mineralized zones to be mined inside the current open pit operation. In all, 496 RCGC holes totalling 28,002 m were drilled. Figure 10-5 shows their location within the 2024 design pit.

Since the beginning of the operation a total of 18,255 vertical blastholes of 10 m length were sampled for gold. This information helped the grade-control department to better delineate additional mineralization outside of the main wireframes. Figure 10-5 shows clearly the east–west trend of the Hardrock deposit.

RCGC and blasthole gold assays are not used in the current MRE.

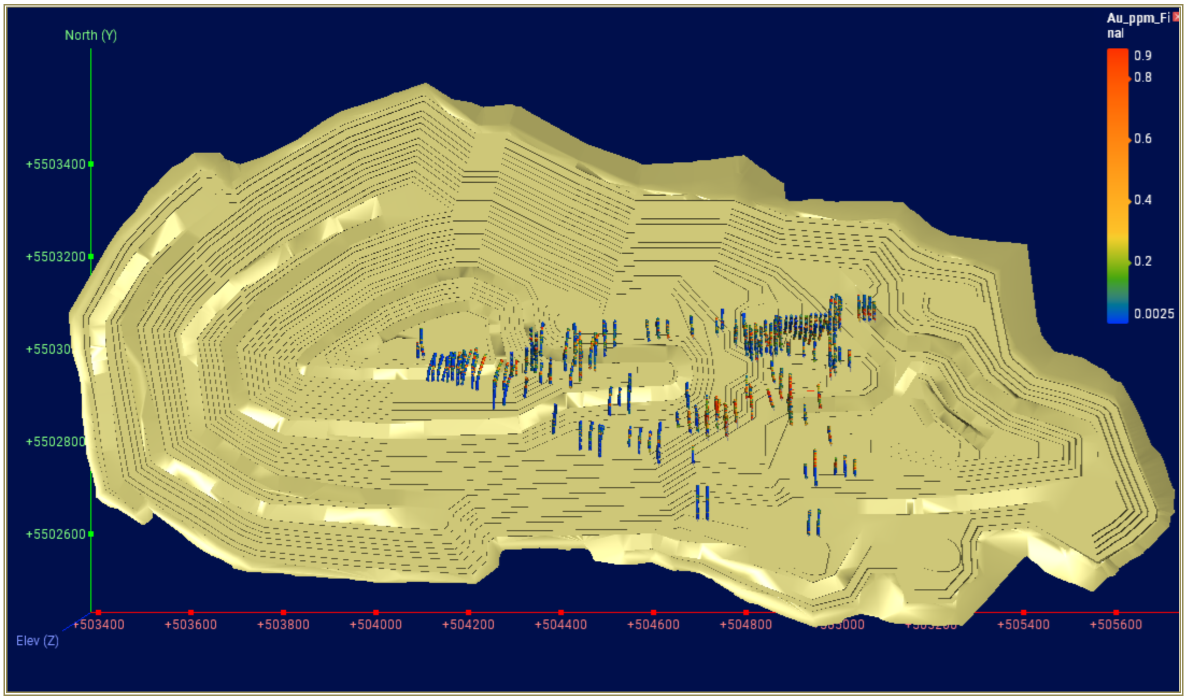


Figure 10-7: Plan View of RCGC Drilling and Gold Assay Results from September 2022 to June 2024 within the 2024 Pit Design

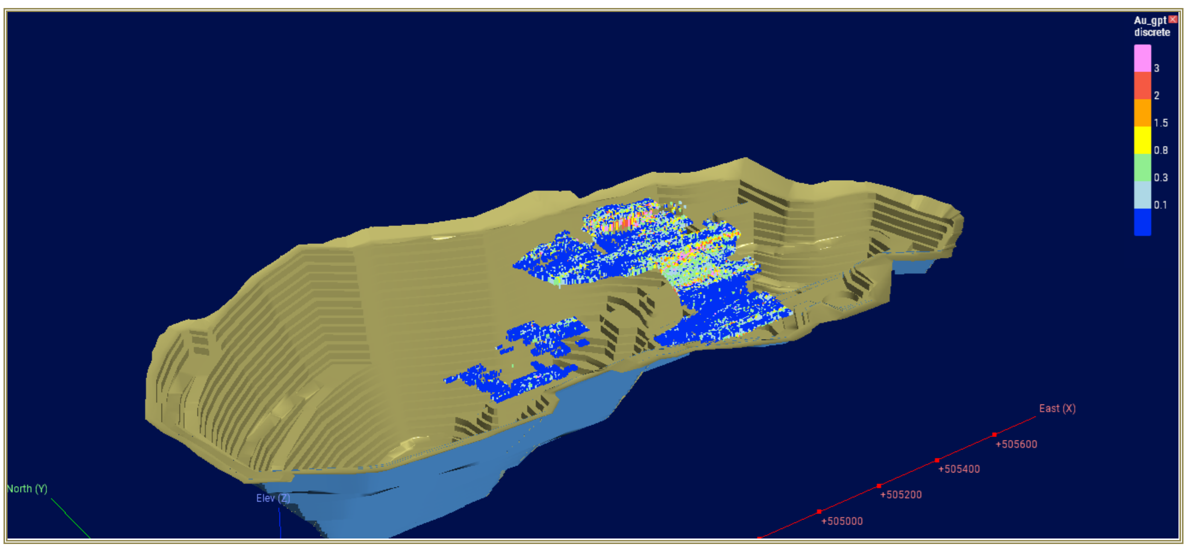


Figure 10-8: View Northeast, Blasthole Drilling and Gold Assay Results up to June 2024 within the 2024 Pit Design

10.1.8 QP Opinion on Drilling—Greenstone Mine

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

Overall, GMS considers the data obtained from the exploration and drilling programs carried out at the GGM to be reliable, meet commonly adopted industry standards, and be suitable for Mineral Resource estimation.

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 and from the previous Technical Report prepared by GMS in 2021.

10.1.9 Drilling Procedures

All Ontex diamond drilling was completed from surface using NQ wire-line-stabilized hexagonal core barrels with an 18 inch-long shell. For deep holes hexagonal core barrels are first used. 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, of Moncton, 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 hole was drilled by Confederation College and the other 13 holes by ForageG4 drilling. Drill holes were surveyed approximately every 10 m using a REFLEX TN14 instrument for single-shot surveys, and a REFLEX EZ-GYRO for multishot surveys.

Core diameter for Premier drilling was 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 depths were labelled by the drill crews. Upon receipt at the Premier core shack, the boxes were labelled with permanent metal tags according to drill-hole number, box number, and metres downhole. 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.

10.1.10 Collar Locations, Orientations, and Downhole 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 NAD 83 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 NAD 83 using the prior surveys to control the grid transformation.

Collars for the Premier drilling were located using a cut grid or by using a hand-held GPS receiver. 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 downhole survey tool provided by the drill contractor was used with surveys typically taken every 50 m. A REFLEX MAXIBORE tool was also used for downhole surveys starting in November 2007. 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 historical holes and 310 holes drilled by Premier.

10.1.11 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 metre downhole using the MPP susceptibility meter from GDD Instruments in Quebec City.

The drillers provided the “ori-marks,” and core was then oriented by technicians 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 (HFZ) 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, 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.

10.1.12 Brookbank

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 are summarized in Section 6.

During 2016, the Brookbank deposit was explored by the Centerra and Premier joint venture.

Two drilling programs were completed on the Brookbank property between July 17 and December 16, 2016: the Brookbank deposit infill program and Brookbank East exploration drilling program. In total, 13 holes were drilled during the two drilling programs (one additional hole was abandoned before completion) (Figure 10-9).

Between October 24, 2016, and December 4, 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 defining targets at depth.

Exploration drilling at Brookbank East, approximately 1 km east of the second exploration program, was performed between November 8 and December 16, 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.

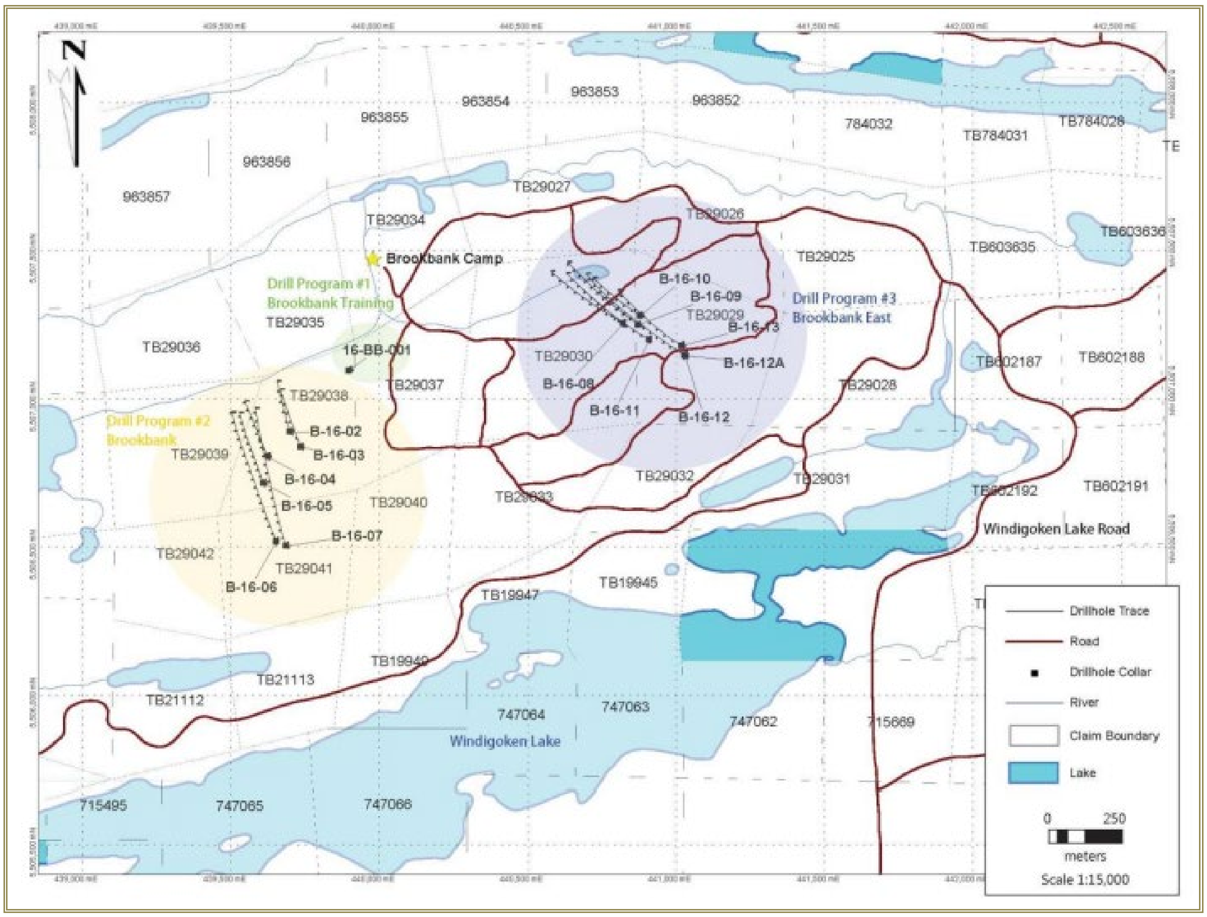


Figure 10-9: Six Drill Traces of Two Drilling Campaigns Performed at Brookbank in 2016 (Green = Brookbank training; Blue = Brookbank East; Yellow = Brookbank)

A summary of all the diamond drilling completed on Brookbank property between 1944 and 2016 is shown in Table 10-3.

Table 10-3: Summary of Brookbank Property Drilling Programs

		Zone								Total			
		Brookbank		Cherbourg		Foxear		Other					
Year(s)	Company	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**

10.1.13 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 digitized from hard-copy records by GGM in 2019 and 2020. Only the more-recent drilling undertaken by Premier is considered as verifiable and used in the MRE described below.

Drilling Campaigns

In late 2007, Premier focused their drilling program in the area around the historical Little Long Lac mine. Eight holes were drilled approximately 200 m southeast of the old mine headframe. All the holes were oriented N334, and they had as target the undeveloped Vein No. 9. While the mine was in production, this mineralized structure was mined out on Levels 2, 4, and 16. The vein is sub-vertical for about 50 m along strike, and steeply dips to the west-southwest at around 60°. Six of the eight drilled holes intercepted the structure, returning anomalous gold values. The initial program also able 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 on two zones, the first target aimed to define mineralization on Veins No. 9, 10, and 11; the second was focused on the newly discovered Kailey Zone. Drilling successfully extended gold mineralization in Veins No. 9 and 10 down plunge.

The Kailey Zone is a low-grade bulk tonnage target proximal and parallel to the historical 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 historical, high-grade resource areas previously mined. The latest drilling programs has demonstrated that the Kailey mineralized zone is a shallow and low-grade potential open pit Mineral Resource. Various drilling programs are summarized in Table 10-4.

Table 10-4: Summary of the Kailey Property Drilling Programs

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

10.1.14 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.

Drilling Campaigns

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

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 of which 1,189 m were assayed.

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 drilling metres for the 1974 to 2011 drilling programs on the Key Lake Property.

Table 10-5: Summary of the Key Lake Property Drilling Programs

		Zone—Key Lake			
Year(s)	Company	No. of Holes	Metres	No. of Samples	Metres Assayed
1974	Jelex Mines Ltd.	2	251.2	-	-
Mid-1980–1990	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	8	3,190.0	896	1,188.8
Total		**312**	**63,919.1**	**21,703**	**23,087.7**

10.1.15 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.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

This section describes GGM sample preparation, analysis, and security procedures for the RCGC and diamond drill hole (DDH) drilling programs performed at the Project since 2012. Data pertaining to sampling, analytical, security, and quality assurance and quality control (QA/QC) protocols were supplied to G Mining Services Inc. (GMS) from GGM's geology team and reviewed accordingly.

11.1 Greenstone Mine

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 was an independent commercial laboratory. GGM purchased the Geraldton facility from Actlabs in March 2024 and has been operating it since.

All re-assaying of batches (pulps) was undertaken at Australian Laboratory Services (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

RCGC Sampling

GGM samplers assisted NPLH Drilling (NPLH) drillers in sampling the RC drill material. The drill rig (Figure 11-1) 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 technicians 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.

A QC 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 Actlabs Geraldton and AGAT Laboratories in Thunder Bay. Samples were tested for 50 g fire assay (excluding QA/QC and re-run by gravimetric analytical method if the sample ran over 5 g/t). Approximately 10% to 15% of these samples were chosen for additional ICP-MS analyses for other elements at Actlabs Thunder Bay.

Figure 11-1: Principal Sample (Small) and Metallurgical Sample (Large) from a 2 m Interval—RCGC Sample

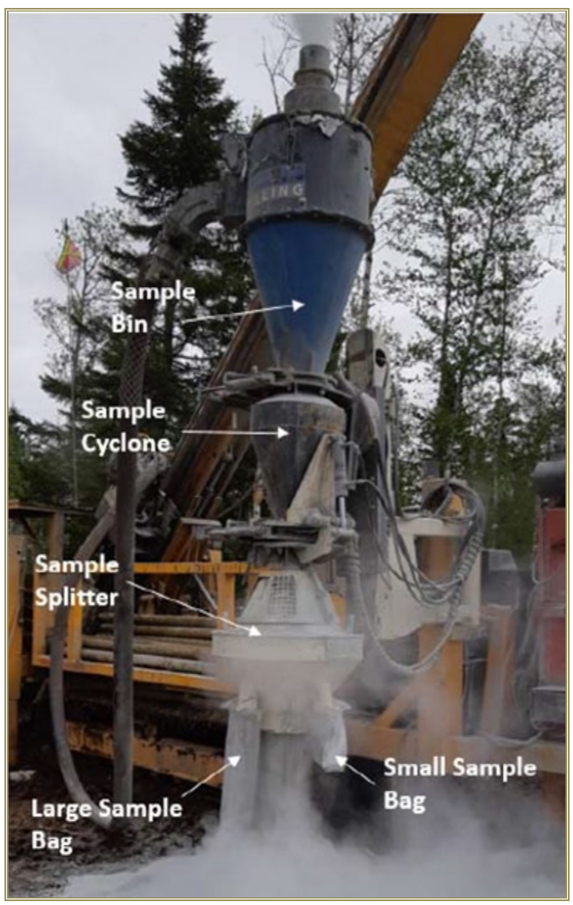


Figure 11-2: Hardab 7000 Maxidrill RC Drill Rig and Splitter in Operation

DDH Sampling

DDH core sample intervals are typically 1.5 m in length, but vary and may be as small as 0.5 m if warranted by the occurrence of veining, intense sulphide mineralization, or the presence of visible gold. Certified reference materials (CRM or standard) 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 sample rejects are stored at the Magnet Mine site.

Figure 11-3: Drill Core—Sawing Shack

Quality Control Sample Preparation by GGM

All QA/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 to the box. One half of each QC sample ticket is placed in the appropriate type of control sample bag, which was prepared beforehand. A list of QC samples and their numbers and 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; no third party is involved in transportation.

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

- 30 regular samples
- 1 field duplicate sample
- 1 field blank
- 1 CRM with a low gold value
- 1 CRM with a high gold value.

As a QC check, Actlabs Geraldton adds a 35th sample to every field batch received—a coarse duplicate of the last regular sample (i.e., 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 more QC samples (two analytical blanks, two CRMs, and three pulp duplicates), bringing the fusible batch to 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

Fire Assay Sample Preparation (Actlabs Geraldton)

Fire-assay samples are received at Actlabs Geraldton, then sorted and bar-coded. They are then dried in the sample drying room 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 (P_{90}) 10 mesh and split with a Jones riffle, and a 250 g split is pulverized to P_{95} 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 samples 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.

Metallic Sieve Sample Preparation (Actlabs Geraldton)

All sieve 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 µm) 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 greater the likelihood of gold loss by plating out on the mill.

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 1 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 gold (doré bead) + Au. The entire silver doré bead is dissolved in aqua regia and the gold content is determined by atomic absorption (AA) finish (1A2-50 code).

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

All samples assaying grades over 5.0 g/t Au with AA were re-run with gravimetric finish to ensure accurate values. After the fire assay procedures, gold is separated from the silver 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.

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 temperatures 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 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.

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, then inquarted with 6 mg of gold-free silver and 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 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 weight.

At the ALS-Chemex laboratory, the batch size for all fire assay methods is 84, including six internal QCs. Therefore, 78 client samples can be taken 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 to Table 11-3 summarize QC results prior to the 2018 and 2019 drilling programs. 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—Analytical Method FA/AA by Actlabs Geraldton Laboratory

Standard (CRM)	Standard Supplier	Certified Gold Value (g/t)	Amount of Results	Lower Process Limit (±10%)	Upper Process Limit (±10%	Outliers	Passing Quality Control (%)
CDN-GS-5F	CDN	5.300	228	4.770	5.830	11	95.2
CDN-GS-5K	CDN	3.840	376	3.456	4.224	27	92.8
CDN-GS-7A	CDN	7.200	2	6.480	7.920	2	0.0
CDN-GS-7B	CDN	6.420	583	5.778	7.062	40	9.31
CDN-GS-8A	CDN	8.250	201	7.425	9.075	16	92.0
SF67	Rocklabs	0.835	227	0.752	0.919	18	92.1
SG40	Rocklabs	0.976	227	0.878	1.074	5	97.8
SJ53	Rocklabs	2.637	131	2.373	2.901	5	96.2
SN60	Rocklabs	8.595	204	7.736	9.455	15	92.6
Total			**2,179**			**139**	**93.6**

Notes: CDN = CDN Resource Laboratories Ltd.; CRM = certified reference material; Rocklabs = Rocklabs Ltd.
Source: Innovexplo (2013).

Table 11-2: Results for Standards Used by Premier During the Drilling Program on the Hardrock Deposit from August 12 to December 31, 2013—Analytical Method FA/GRAV

Standard (CRM)	Standard Supplier	Certified Gold Value (g/t)	Amount of Results	Lower Process Limit (−2 SD)	Upper Process Limit (+2 SD)	Outliers	Passing Qualify Control (%)
CDN-GS-5K	CDN	3.85	1,191	3.33	4.37	11	99.08
CDN-GS-6C	CDN	5.95	477	4.99	6.91	12	97.48
CDN-GS-7B	CDN	6.37	555	5.43	7.31	22	96.04
CDN-GS-8A	CDN	8.25	3	7.05	9.45	0	100
SF67	Rocklabs	0.835	256	0.793	0.877	85	66.80
SN60	Rocklabs	8.318	249	7.694	8.942	16	93.57
Total			**2,731**			**146**	**94.65**

Notes: CDN = CDN Resource Laboratories Ltd.; CRM = certified reference material; Rocklabs = Rocklabs Ltd.; SD = standard deviation.
Source: Innovexplo (2015).

Table 11-3: Results for Standards used by Premier during the Drilling Program on the Hardrock Deposit from January 2 to May 26, 2014

Standard (CRM)	Standard Supplier	Certified Gold Value (g/t)	Analytical Method	Count of Results	Lower Process Limit (−2 SD)	Upper Process Limit (+2 SD)	Outliers	Passing Qualify Control (%)
CDN-GS-5K	CDN	3.85	FA/GRAV	207	3.33	4.37	3	98.55
CDN-GS-5K	CDN	3.85	FA/AA	160	3.33	4.37	2	98.75
CDN-GS-6C	CDN	5.95	FA/GRAV	114	4.99	6.91	4	96.49
CDN-GS-6C	CDN	3.85	FA/AA	26	4.99	6.91	0	100
CDN-GS-7B	CDN	6.37	FA/GRAV	111	5.43	7.31	22	94.59
CDN-GS-7B	CDN	6.37	FA/AA	53	5.43	7.31	0	100
SF67	Rocklabs	0.835	FA/GRAV	20	0.793	0.877	1	95.00
SN60	Rocklabs	8.318	FA/AA	66	7.694	8.942	9	86.36
Total				**757**			**25**	**96.70**

Notes: CDN = CDN Resource Laboratories Ltd.; CRM = certified reference material; Rocklabs = Rocklabs Ltd.; SD = standard deviation.
Source: Innovexplo (2015).

Blanks

The field blank material 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/t Au (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/t Au (Figure 11-4).

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/t Au (Figure 11-5).

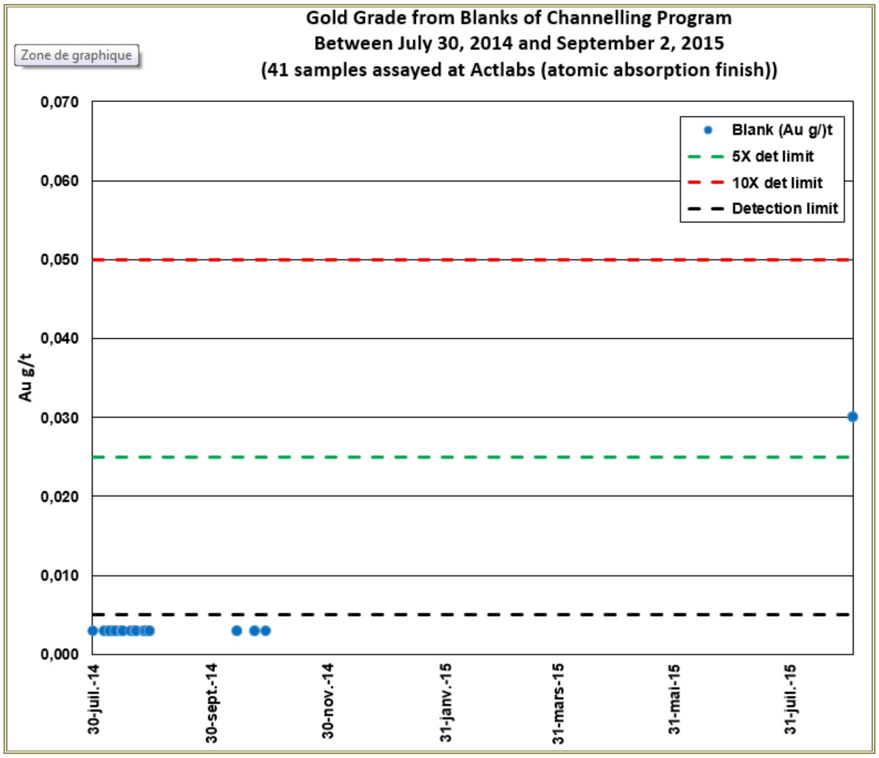


Figure 11-4: Results of Blank Samples Used for Quality Control During Channelling Program Hardrock Deposit between July 30, 2014, and September 2, 2015. Detection Limit = 0.005 g/t Au for AA Finish

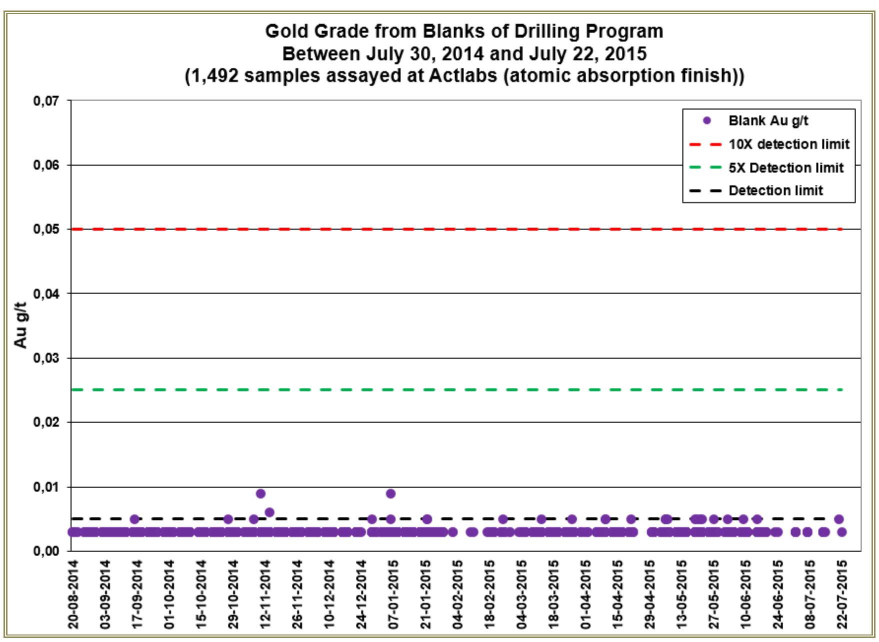


Figure 11-5: Results of Blank Samples used for Quality Control during the Drilling Program on the Hardrock Deposit between July 30, 2014, and July 22, 2015. Detection Limit = 0.005 g/t Au for AA Finish

Certified Reference Material (Standards)

Two CRMs 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/t Au as follows:

- CDN-GS-P4B with a theoretical value of 0.417 ± 0.023 g/t Au
- CDN-GS-P7J with a theoretical value of 0.722 ± 0.036 g/t Au
- CDN-GS-1L with a theoretical value of 1.160 ± 0.050 g/t Au
- CDN-GS-2P with a theoretical value of 1.990 ± 0.075 g/t Au
- CDN-GS-5K with a theoretical value of 3.840 ± 0.140 g/t Au
- CDN-GS-6C with a theoretical value of 6.030 ± 0.280 g/t Au
- CDN-GS-7B with a theoretical value of 6.420 ± 0.230 g/t Au
- SF67 with a theoretical value of 0.835 ± 0.021 g/t Au
- SN60 with a theoretical value of 8.595 ± 0.223 g/t Au.

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

The results of all standards used in the Hardrock deposit 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 inTable 11-5.

Overall, more than 97.50% of the available results for standards passed the QC 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—Analytical Method FA/AA

Standard (CRM)	Standard Supplier	Certified Gold Value (g/t)	Count of Results	Lower Process Limit (−2 SD)	Upper Process Limit (+2 SD)	Outliers	Passing Qualify Control (%)
CDN-GS-2P	CDN	1.99	2	1.765	2.22	0	100
CDN-GS-5K	CDN	3.84	39	3.46	4.24	2	94.87
CDN-GS-6C	CDN	6.03	40	5.31	6.75	0	100
CDN-GS-7B	CDN	6.42	1	5.73	7.11	0	100
Total			**80**			**2**	**97.50**

Notes: CDN = CDN Resource Laboratories Ltd.; CRM = certified reference material; SD = standard deviation.

Table 11-5: Results for Standards used by Premier during the Drilling Program on Hardrock Deposit from July 30, 2014, to July 22, 2015—Analytical Method FA/AA

Standard (CRM)	Standard Supplier	Certified Gold Value (g/t)	Count of Results	Lower Process Limit (-2 SD)	Upper Process Limit (+2 SD)	Outliers	Passing Qualify Control (%)
CDN-GS-P4B	CDN	0.417	474	0.348	0.486	21	95.57
CDN-GS-P7J	CDN	0.733	70	1.01	1.31	2	97.14
CDN-GS-1L	CDN	1.16	71	1.01	1.31	1	98.59
CDN-GS-2P	CDN	1.99	114	1.77	2.22	3	97.37
CDN-GS-5K	CDN	3.84	804	3.46	4.24	18	.97.76
CDN-GS-6C	CDN	6.03	589	5.47	6.59	12	97.96
CDN-GS-7B	CDN	6.42	531	5.72	7.12	8	98.48
SF67	Rocklabs	0.835	177	0.772	0.898	1	99.44
SN60	Rocklabs	8.595	145	7.926	9.264	7	98.17
Total			**2,975**			**73**	**97.55**

Note: CDN = CDN Resource Laboratories Ltd., CRM = certified reference material; SD = standard deviation.

Coarse Reject Duplicates

The QC 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 are 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/t Au, no outlier is observed on the graph because no duplicate pair is outside the lines marking a ±20% relative difference.

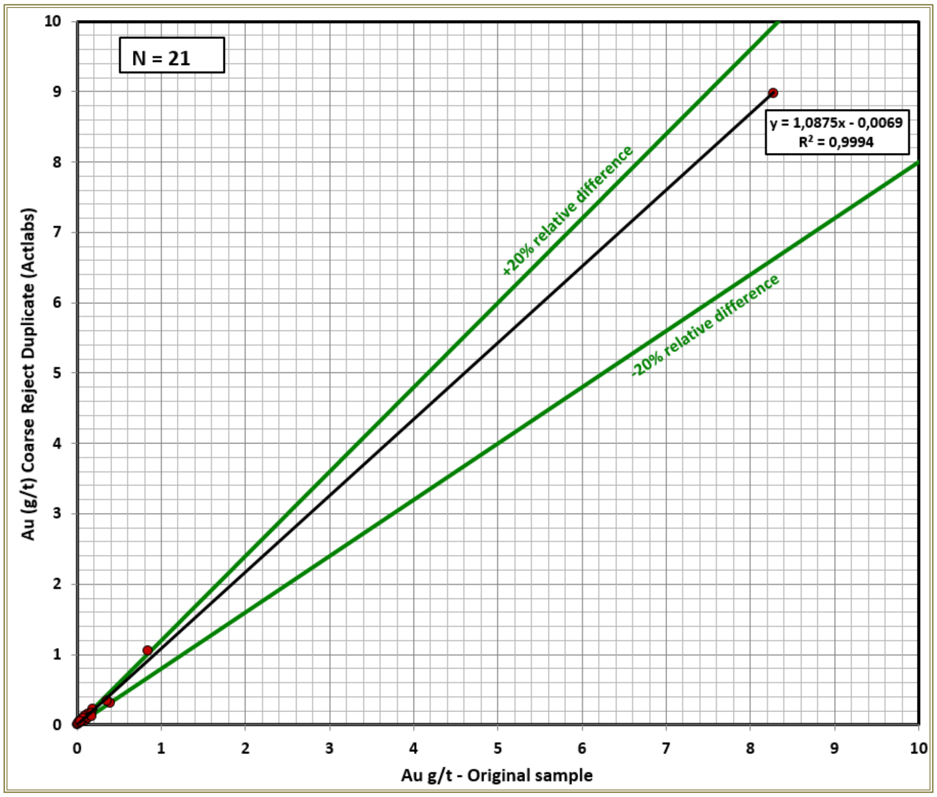


Figure 11-6: Linear Regression Comparing Original Samples and Crush-Coarse Duplicate Samples (Duplicate Pairs) between July 30, 2014, and September 2, 2015 (Channelling)

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/t Au, 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 deposit coarse duplicates between 2012 and 2016 are reliable and valid.

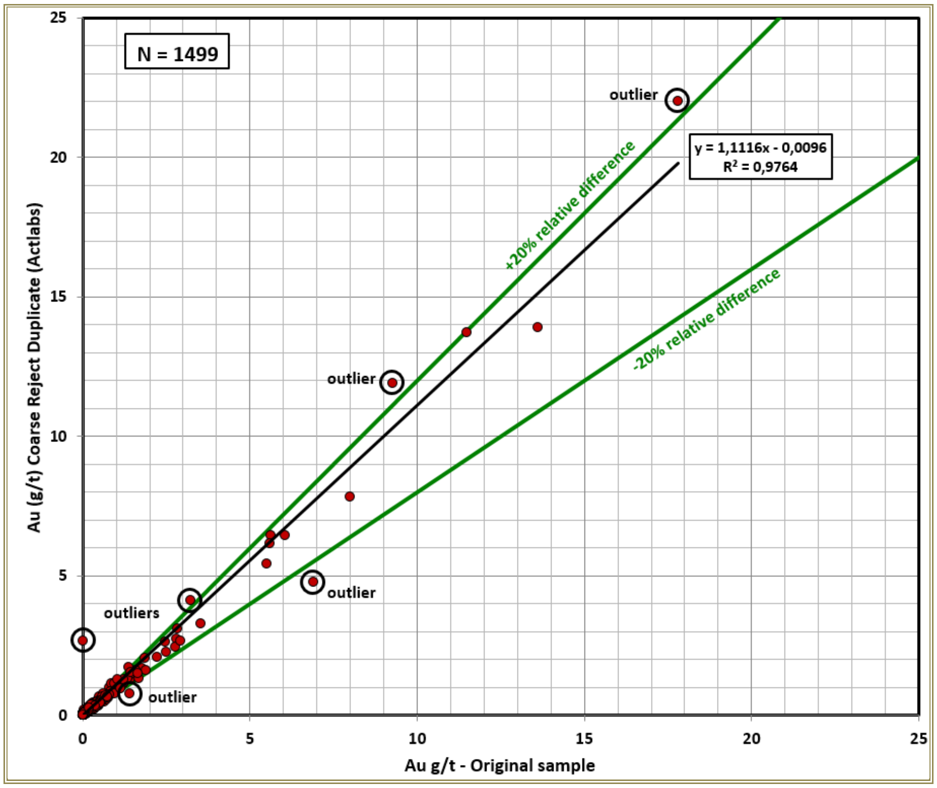


Figure 11-7: Linear Regression Comparing Original Samples and Crush Coarse Duplicate Samples (Duplicate Pairs) for the Period between July 30, 2014, and July 22, 2015 (Drilling)

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.

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/t Au (10X detection limit [DL] for gravimetric finish), the blank is re-run, as well as 10 samples before and after the failed QC sample. All re-runs 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.

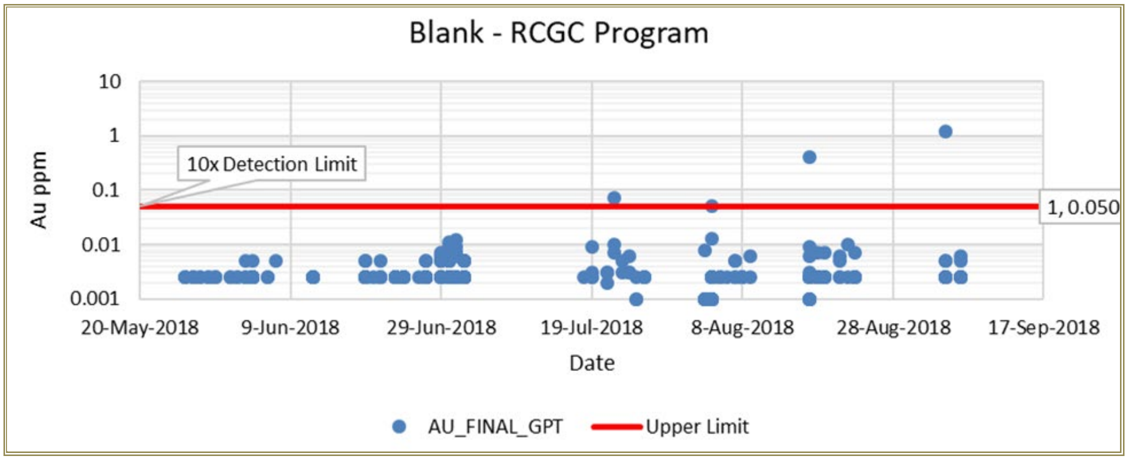


Figure 11-8: QA/QC—2018 Blank Results of RCGC Samples

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.

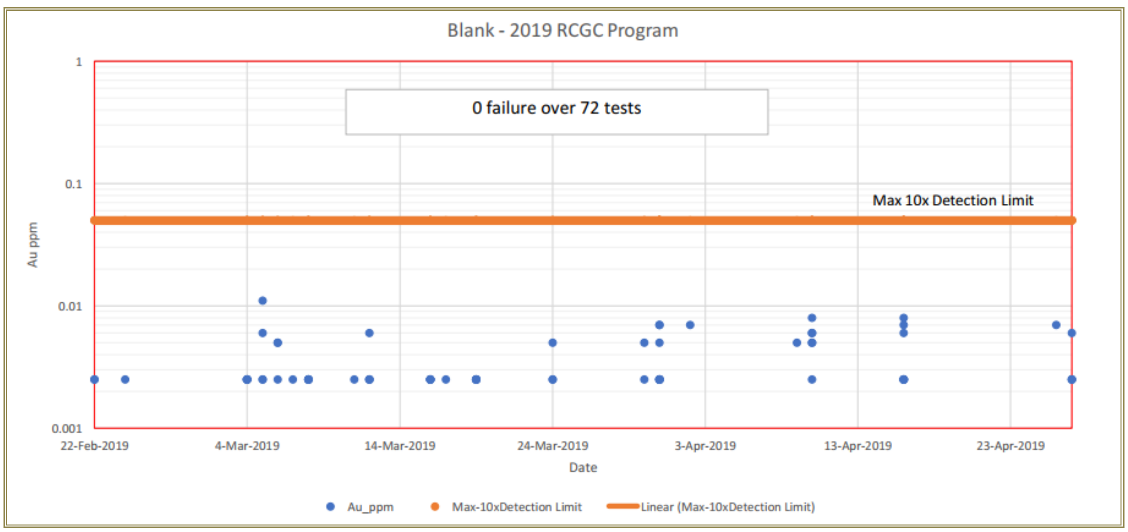


Figure 11-9: QA/QC—2019 Blank Results of RCGC Assays

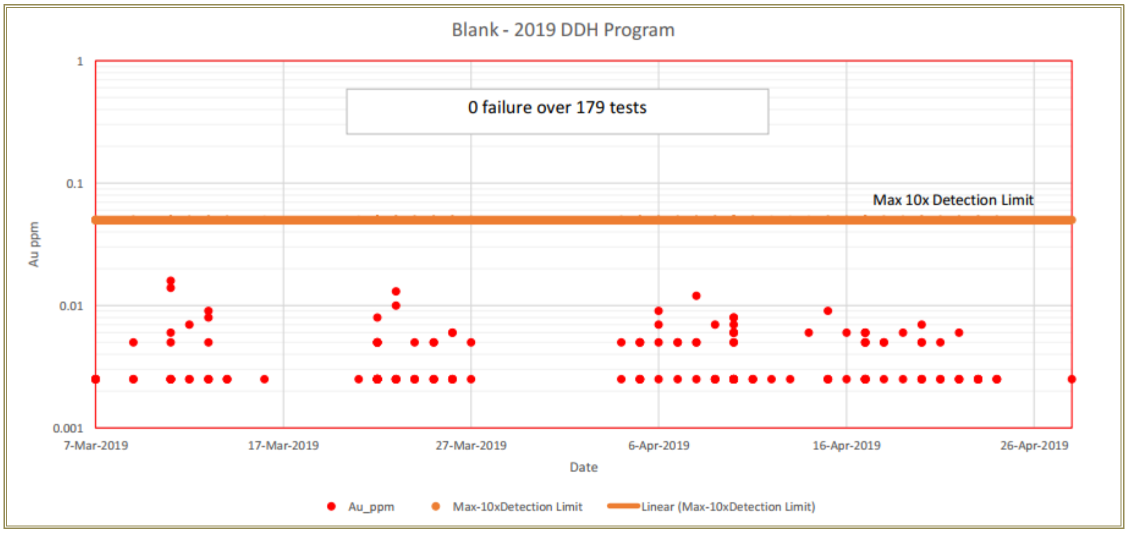


Figure 11-10: QA/QC—Blank Results of DDH Assays

Certified Reference Material

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 OREAS (www.ore.com.au/oreas-reports/) and CDN Labs (www.cdnlabs.com/Cu-Au-standards.htm) website.

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

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

Internal laboratory standards were also used by Actlabs:

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

GGM QC protocol stipulates that if any analyzed standard yields a gold value above or below three standard deviations (3 SD) of the certified grade for that standard, then the CRM is re-run with 10 samples before and after. The re-run material is analyzed and compared to the 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.

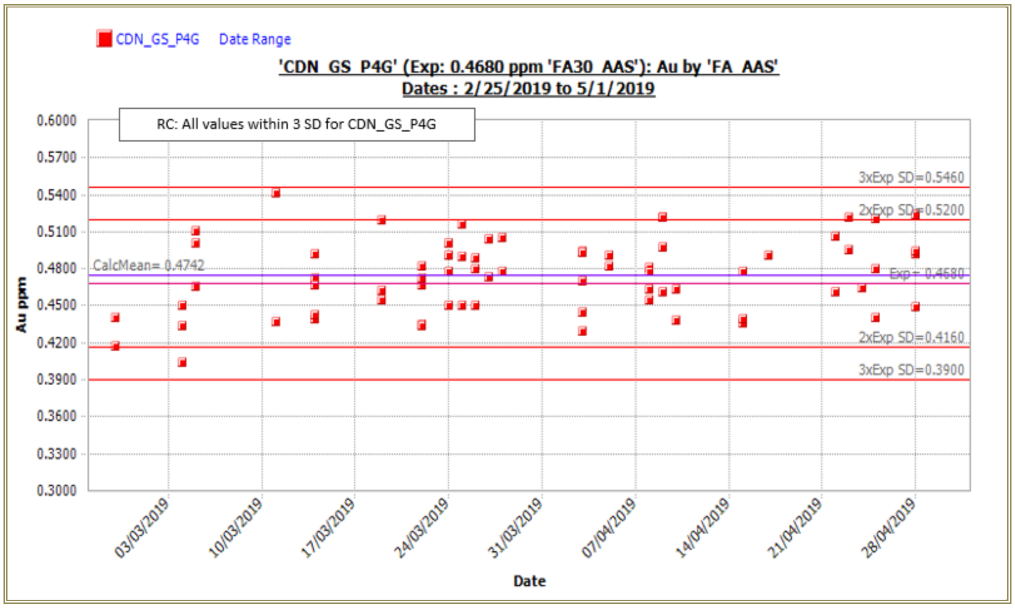


Figure 11-11: Standard CDN-GS-P4G Results—RC Assays

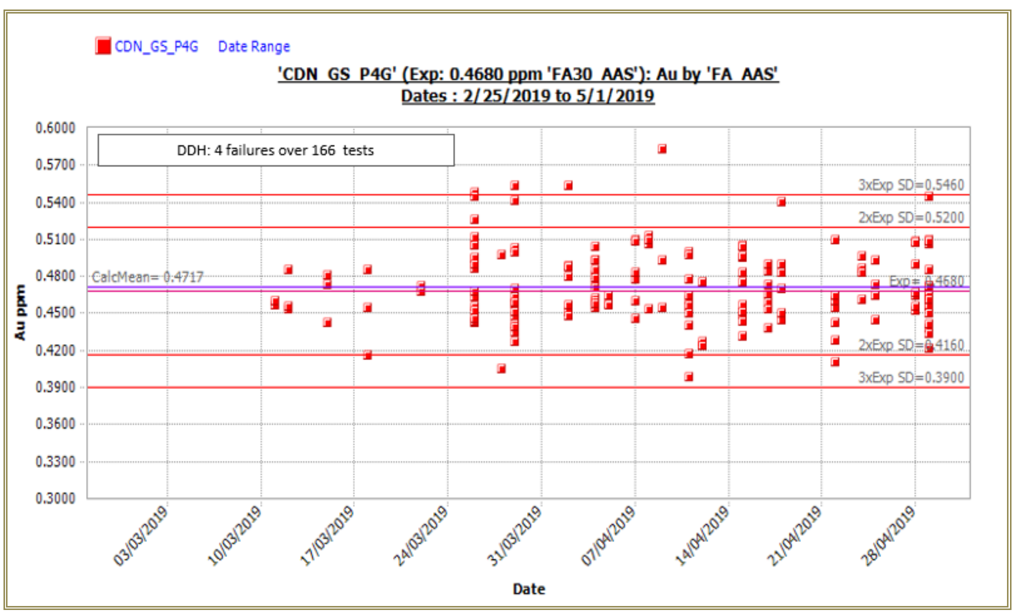


Figure 11-12: Standard CDN-GS-P4G Results—DDH Assays

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					Calculated Values			
Standard (CRM)	Analytical Method	Certified Value	-3 SD	+3 SD	No. of Samples	No. of Failures	% Failure	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**

Notes: CRM = certified reference material; CV = Coefficient of variations; FA_AAS = fire assay–atomic absorption spectroscopy; FA_GRAV = fire assay–gravity; QC = quality control; SD = standard deviation.

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 inTable 11-8. More than 98.2% of the available standard's results passed the QC 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					Calculated Values			
Standard (CRM)	Analytical Method	Certified Value	−3 SD	+3 SD	No. of Samples	No. of Failures	% Failure	Mean Gold	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**

Notes: CRM = certified reference material; CV = Coefficient of variations; FA_AAS = fire assay–atomic absorption spectroscopy; FA_GRAV = fire assay–gravity; QC = quality control; SD = standard deviation.

Table 11-8: Standard Result Summary (Excluding Internal Lab Standards)—DDH Samples

Au (g/t) Standard(s)			Limits					Calculated Values			
Standard (CRM)	Analytical Method	Certified Value	−3 SD	+3 SD	No. of Samples	No. of Failures	% Failure	Mean Gold	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%**

Notes: CRM = certified reference material; CV = Coefficient of variations; FA_AAS = fire assay–atomic absorption spectroscopy; FA_GRAV = fire assay–gravity; QC = quality control; SD = standard deviation.

RC Field Duplicates and Quarter-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, quarter-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 gold values (i.e., <0.3 g/t Au). This is not uncommon, because as the gold values approach the limit of detection, determinations become less accurate. For this reason, a precision limit of ±20% where the value was 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.

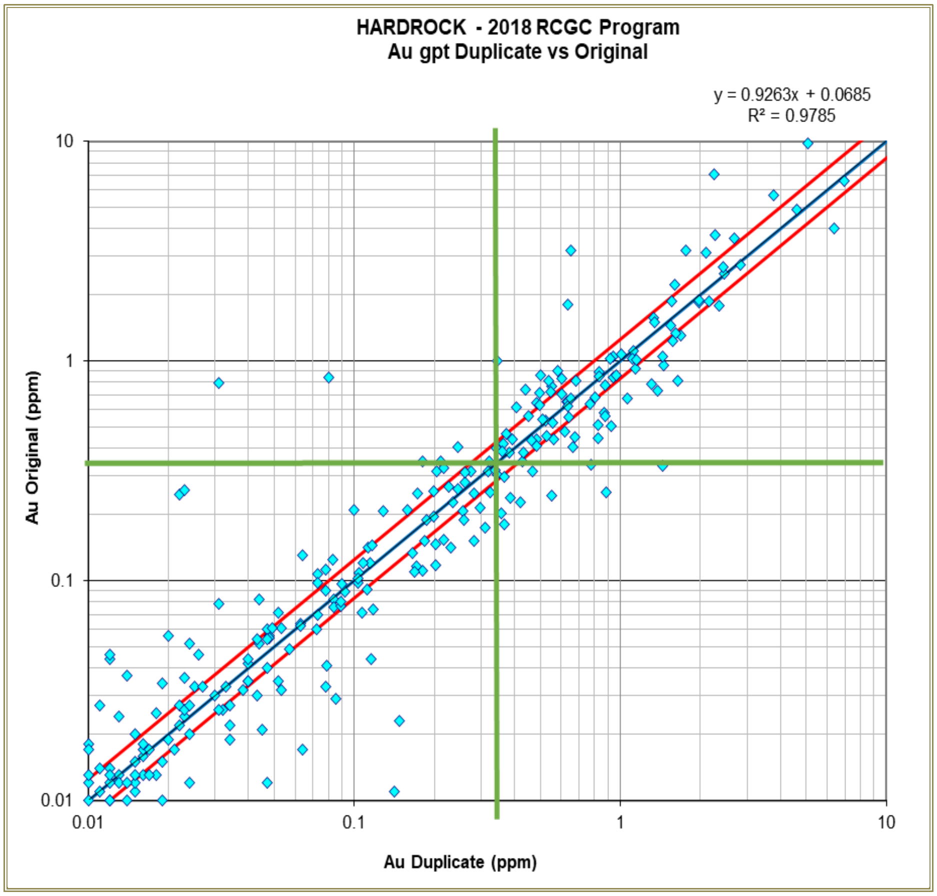


Figure 11-13: 2018 Field Duplicates for Gold Values—RCGC Samples

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.

A total of 172 quarter-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.

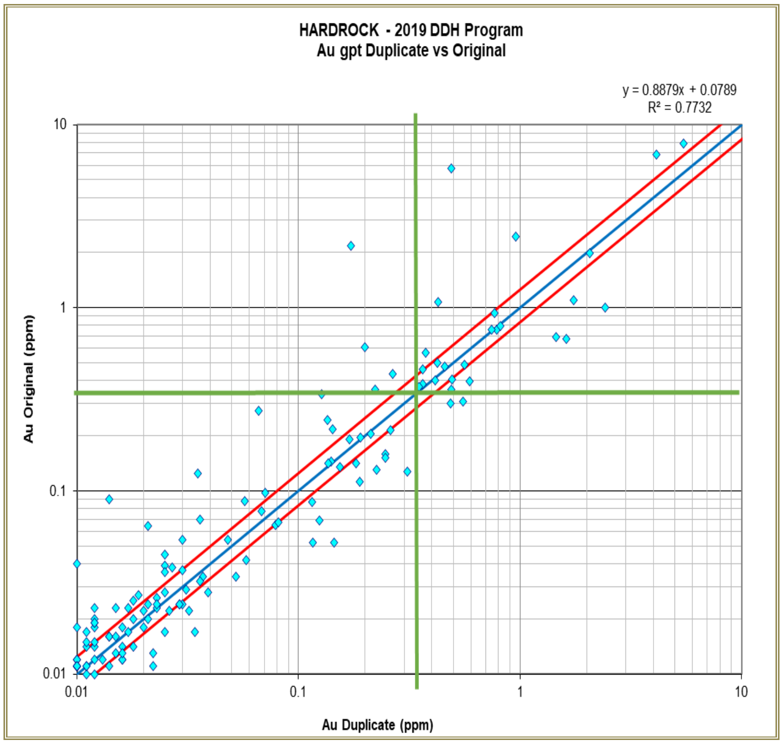


Figure 11-14: 2019 Quarter-Core Duplicates for Gold Values—DDH Samples

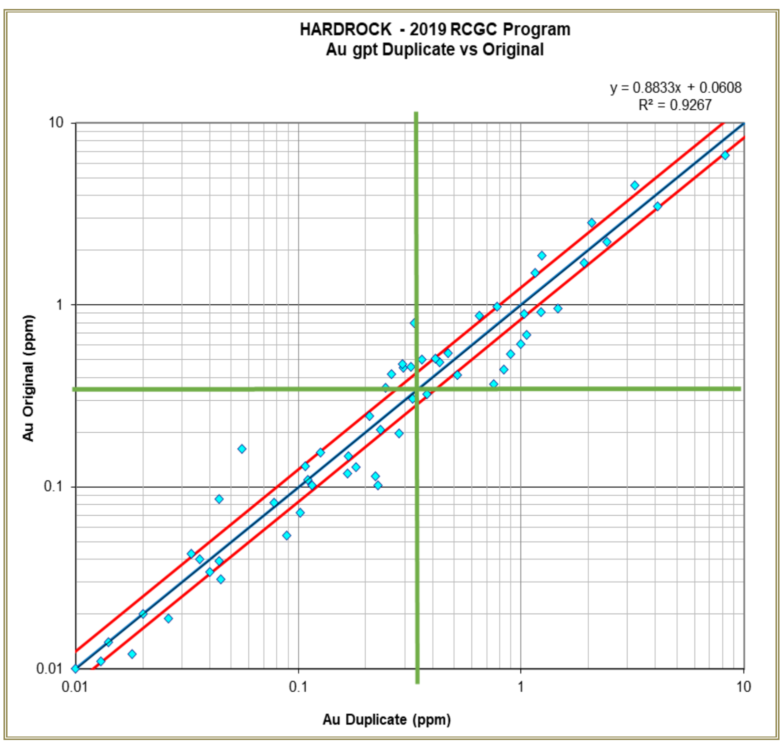


Figure 11-15: 2019 Field Duplicates for Gold Values—RCGC Samples

11.1.6 Qualified Person's 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 are 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 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)

Brookbank

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

During 1999 to 2009 drilling campaigns, Ontex mainly used either Actlabs Geraldton or Swastika Laboratories Ltd. (Swastika) in Swastika, Ontario for analysis of drill core samples. 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. Accurassay is an accredited by the Standards Council of Canada Laboratory 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.

Sample Preparation and Analyses

Swastika Laboratories

Each sample shipment was checked for the count of samples, the condition of the packaging, the integrity of the sample seals, and the customer's analysis instructions. Any damage, evidence of altering original batches, 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, P_{80} 10 mesh (1,700 µm). The crushed material was split successively in a riffle divider to arrive at a subsample of 300 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 P_{90} to P_{95} 100 mesh (150 µm). 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 from pulverized material, weighed, and placed into a 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 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 5 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/t Au.
- All grades obtained are reported in either parts per million or grams per tonne.

The internal QC at Swastika was carried out by using in-house or Canmet certified standards and 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.

Accurassay

The description below, excerpted from the RPA 2009 technical report, discusses 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 (Low Intensity Magnet Separation [LIMS]). The samples were dried, crushed to approximately eight mesh (2,360 µm) and then a 250 to 500 g subsample was taken. The subsample was pulverized to P_{90} 150 mesh (106 µm), 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 that 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/t Au to be verified by re-assay (assayed in triplicate) to ensure reproducibility.
- Ontex samples grading greater than 30 g/t Au would have been verified by Accurassay internally, but not reported. Gravimetric analysis is offered only by request.
- Accurassay employed an internal QC 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.

<u>Actlabs</u>

The details of sample preparation and analysis performed for gold at Actlabs in 2009, are similar to the description discussed in Section 11.1.3.

Ontex QA/QC Program

<u>Blanks</u>

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

- 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 and August 2009.

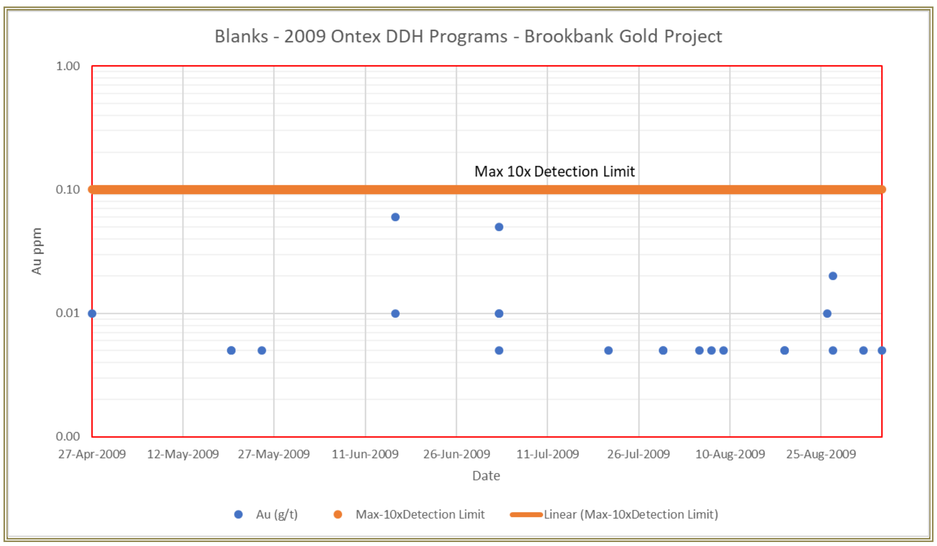


Figure 11-16: Blanks at Swastika Laboratories Ltd.—Ontex 2009 Drilling Campaign

Certified Standards

Ontex inserted two reference materials into the sample stream at a rate of one for each 20-sample batch submitted at Swastika. 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 and 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**	

Notes: CRM = certified reference material.

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 11-17 and Figure 11-18.

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

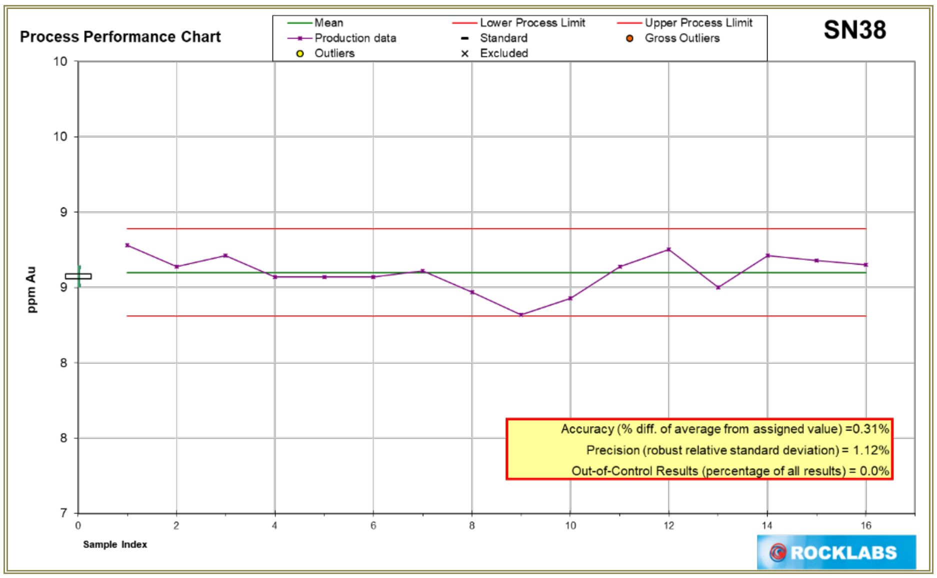


Figure 11-17: Control Chart of Certified Standard (SN38)—2009 QA/QC Program

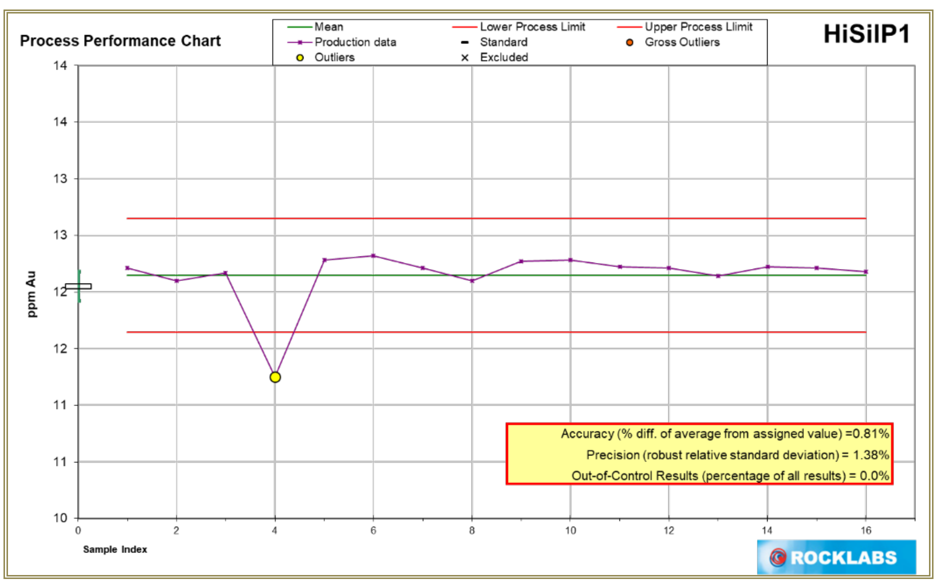


Figure 11-18: Control Chart of Certified Standard (HiSilP1)—2009 QA/QC Program

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.

In addition, 44 pulp duplicates were assayed by Actlabs in 2009 as an additional QC, shown in Figure 11-20. The comparison shows excellent reproducibility of the original assay results.

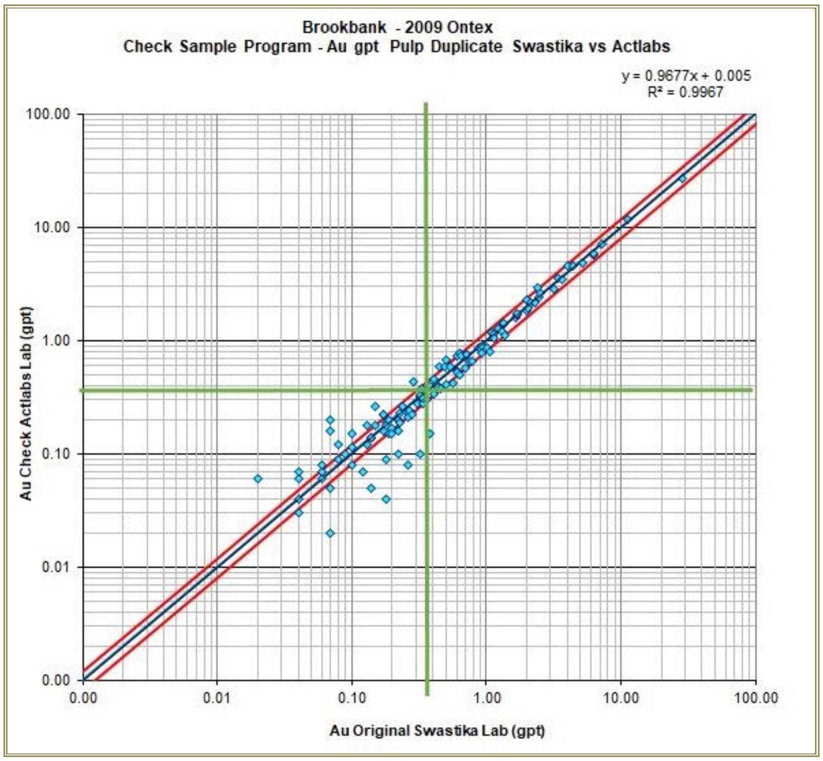


Figure 11-19: Pulp Duplicate Results Comparing Swastika and Actlabs Laboratories—2009

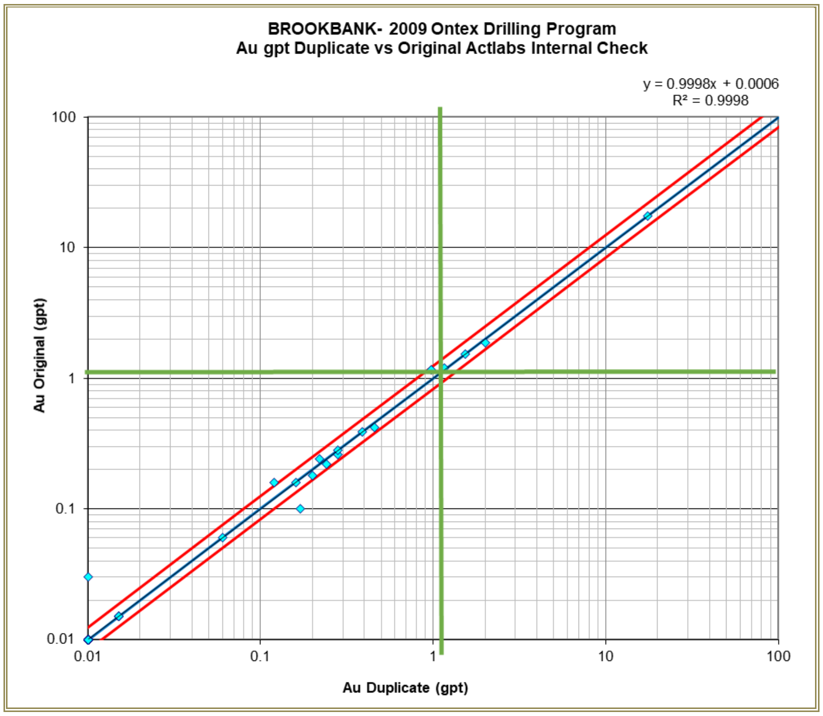


Figure 11-20: Actlabs Internal Quality Control Chart—Brookbank (2009)

Kailey

Descriptions of sample preparation, analysis, or security are not available for exploration work carried out on the Kailey property before the time of Premier. All historical data integrated by GGM's 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.

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 on 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.

Sample Preparation and Analyses

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

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

Goldstone QA/QC Program (2010—2011)

This section discusses the QA/QC program performed at the Key Lake Property during 2010 and 2011.

Goldstone implemented a thorough QC program for the drill program at the Key Lake Property, with the insertion of two CRM samples (standards)and 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 was sent to Actlabs for analysis. This number includes the QC samples mentioned 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, and duplicate samples.

Blanks

All blank material data for gold were plotted, using an upper tolerance limit of 10X DL—0.005 ppb.

Figure 11-21 shows that the blanks performed well, with only one result falling outside the 10X DL. The failure value was identified by the lab's internal QC, and has no impact on the resource database.

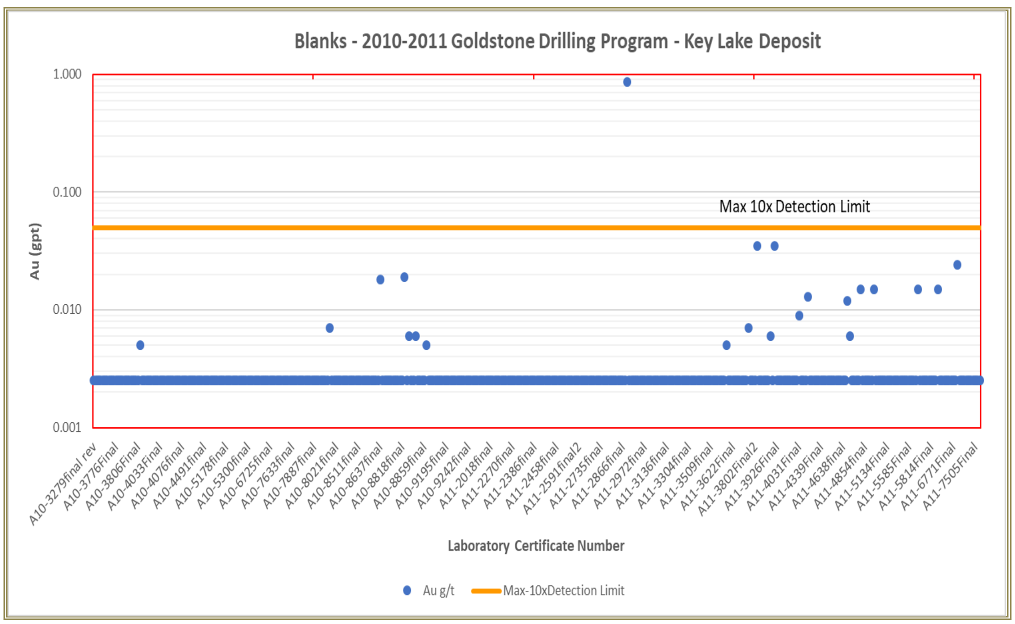


Figure 11-21: Performance of Blank for Gold

Certified Standards

A total of four different CRMs were used throughout the 2010 and 2011 Key Lake drill programs. The OREAS CRMs (OREAS 2Pd, 6Pc, 54Pa, and 62d) were purchased from Analytical Solutions Ltd. (ASL) in Toronto, and the standard supplier was also OREAS.

Standards OREAS 2Pd and OREAS 6Pc are plotted in Figure 11-22 and Figure 11-23. All results outside the ±3 SD 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 re-run 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.

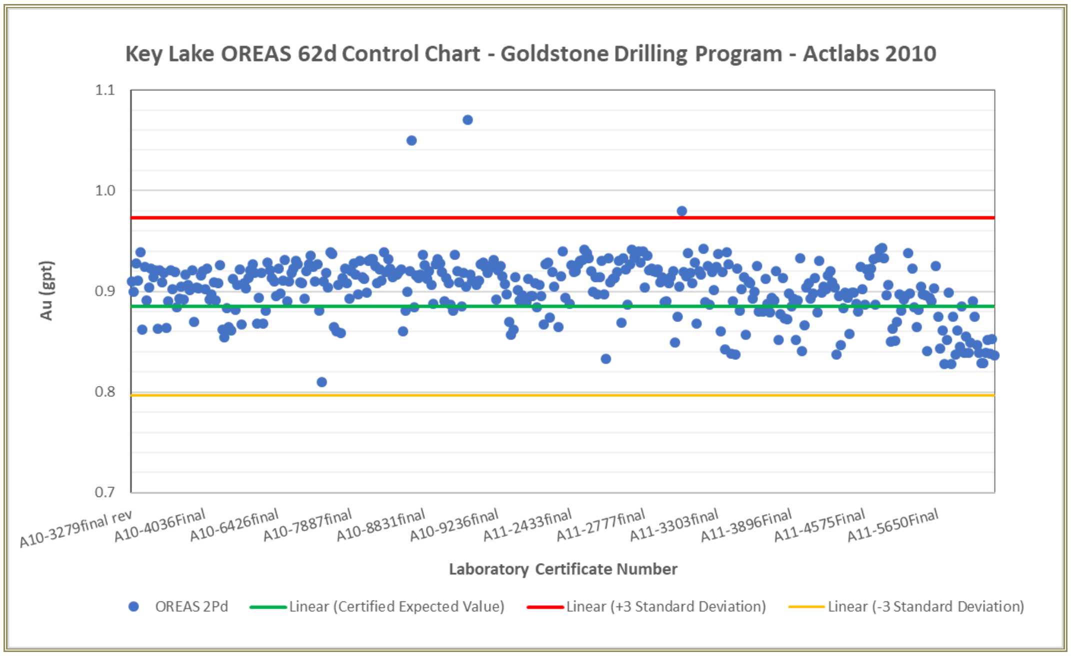


Figure 11-22: Performance of OREAS 2Pd Reference Material for Gold

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.

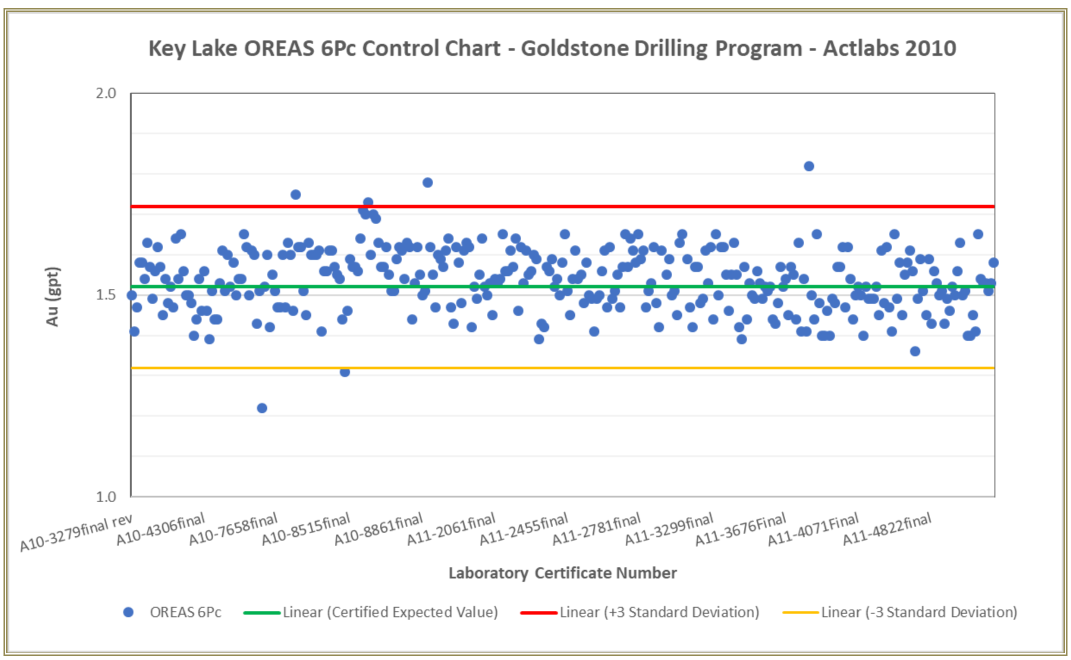


Figure 11-23: Performance of OREAS 6Pc Reference Material for Gold

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.

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; the pulp duplicates had the best precision (due to fineness of grain size and homogenization).

Figure 11-24 shows the results of field duplicates. The comparison has poor reproducibility, with an R^2 of 0.5566, which is mostly affected by the results of 2011 field duplicates.

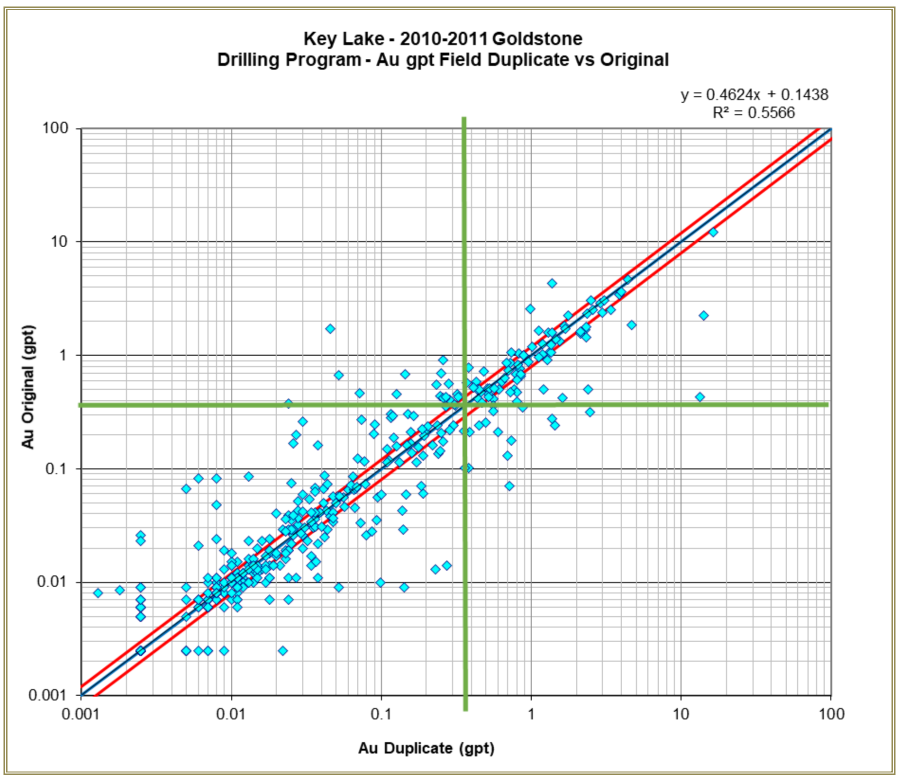


Figure 11-24: Field Duplicates Control Chart—Key Lake (2010 to 2011)

Figure 11-25 and Figure 11-26 illustrate a good reproducibility with R^2 close to 0.9918 and 0.9836 respectively.

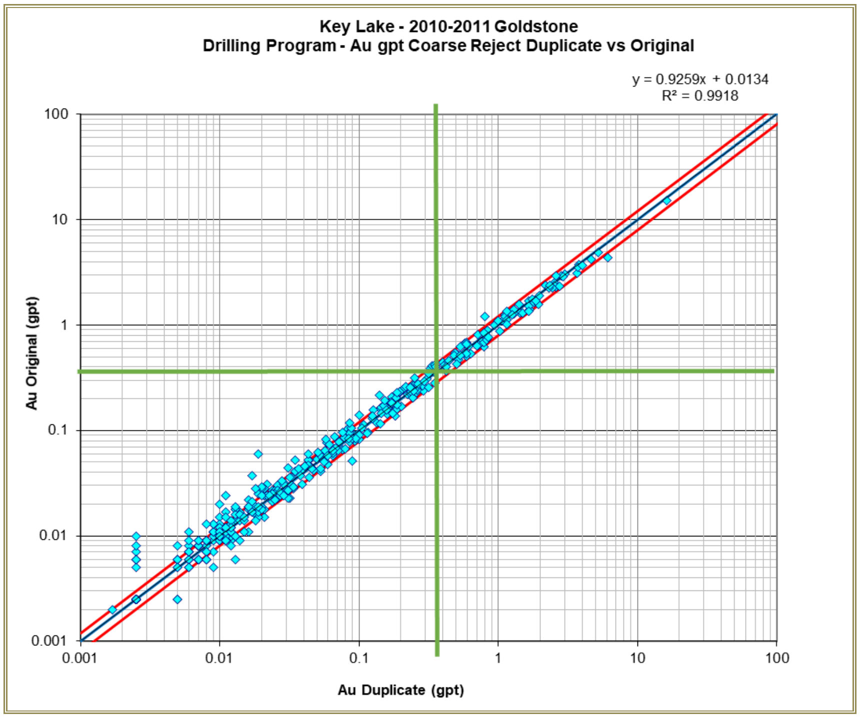


Figure 11-25: Coarse Reject Duplicates Control Chart—Key Lake (2010 to 2011)

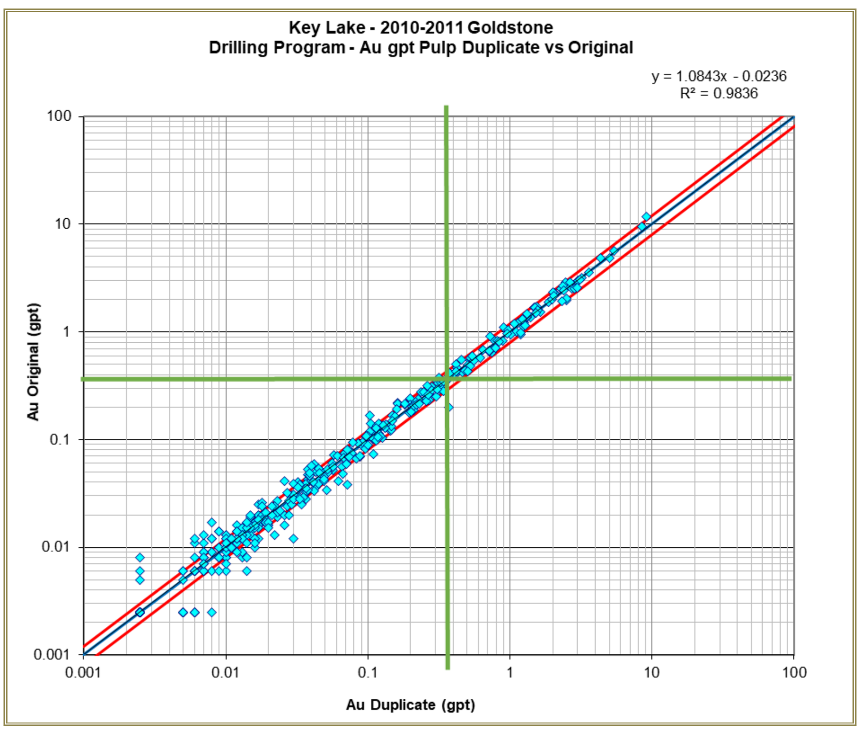


Figure 11-26: Pulp Duplicates Control Chart—Key Lake (2010 to 2011)

11.2.2 Premier Sampling Procedures and QA/QC

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

Protocols Before Sample Dispatch

Drill core sampling protocols are described below. Sample batches including the core drill and QC samples were placed into rice bags, sealed, and transported to Actlabs Geraldton in trucks by Premier staff. Sample pulps were shipped to Actlabs Thunder Bay for analytical work. Actlabs was 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.

Sample Preparation and Analysis

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

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 into the 2010 to 2016 exploration programs.

During the drilling programs executed by Premier, a QC 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.

Analytical Standards

From 2007 to 2016, 16 different CRMs 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 of the reference materials were purchased from CDN Labs (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 from ROCKLABS (SH35, SL34, SG40 and SJ53), and two of the materials were standards 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 to 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					Calculated Values			
CRM Code	Analytical Method	Certified Value	−3 SD	+3 SD	No. of Samples	No. of Failures	% Failure	Mean Gold	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**

Notes: CRM = certified reference material; CV = Coefficient of variations; FA_AAS = fire assay–atomic absorption spectroscopy; FA_GRAV = fire assay–gravity; QC = quality control; SD = standard deviation.

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					Calculated Values			
CRM Code	Analytical Method	Certified Value	−3 SD	+3 SD	No. of Samples	No. of Failures	% Failure	Mean Gold	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**

Notes: CRM = certified reference material; CV = Coefficient of variations; FA_AAS = fire assay–atomic absorption spectroscopy; FA_GRAV = fire assay–gravity; QC = quality control; SD = standard deviation.

Table 11-12: Results for Standards Used by Premier During the Drilling Program on the Key Lake Deposit from 2011

Au Standard (s)			Limits					Calculated Values			
CRM Code	Analytical Method	Certified Value	−3 SD	+3 SD	No. of Samples	No. of Failures	% Failure	Mean Gold	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**

Notes: CRM = certified reference material; CV = Coefficient of variations; FA_AAS = fire assay–atomic absorption spectroscopy; QC = quality control; SD = standard deviation.

The analytical results were graphed to illustrate the performance of the QC samples by using the convention of ±3 SD control limit within which the standard values should fall. In case of failures, all reanalyzed batches (pulps) were sent to ALS Minerals in Vancouver, British Columbia.

Figure 11-27 and Figure 11-28 present examples of a standards control chart for the Kailey and Key Lake deposits.

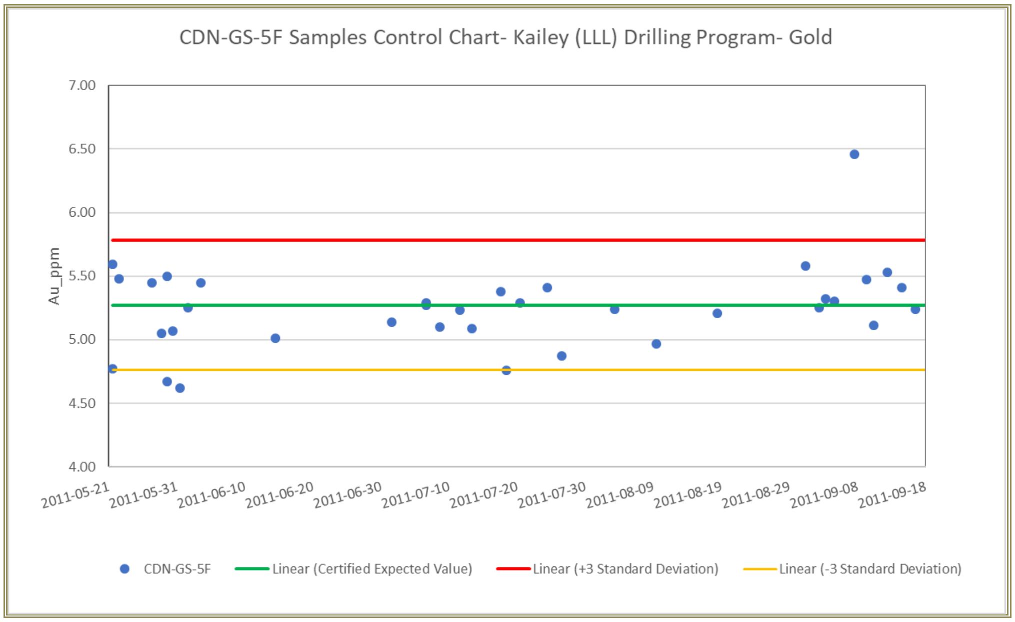


Figure 11-27: Control Chart—Standard CDN-GS-5F: Outliers Included—Kailey Drilling Program (2011)

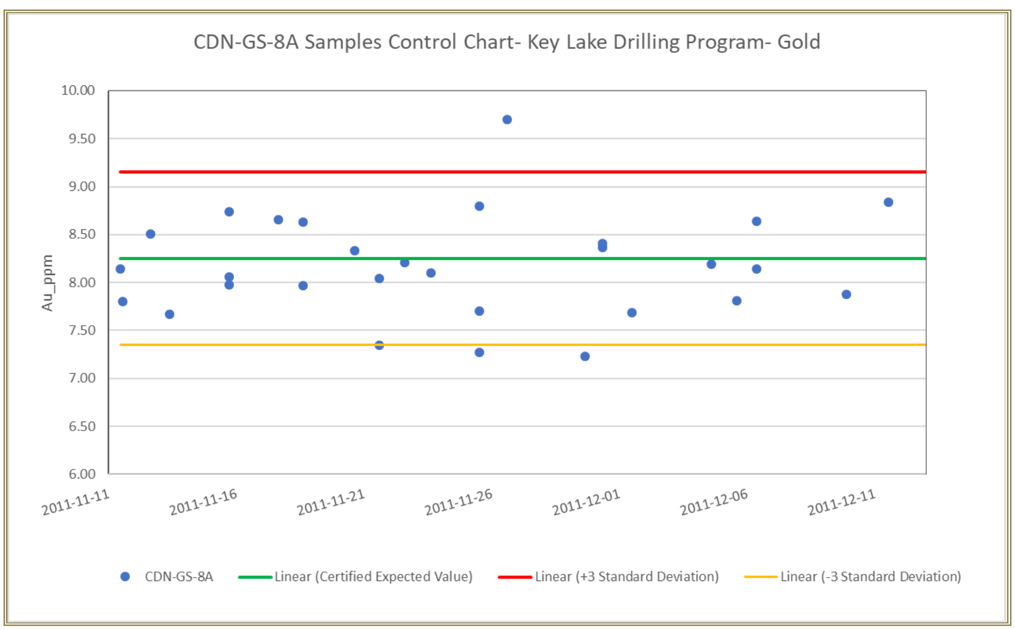
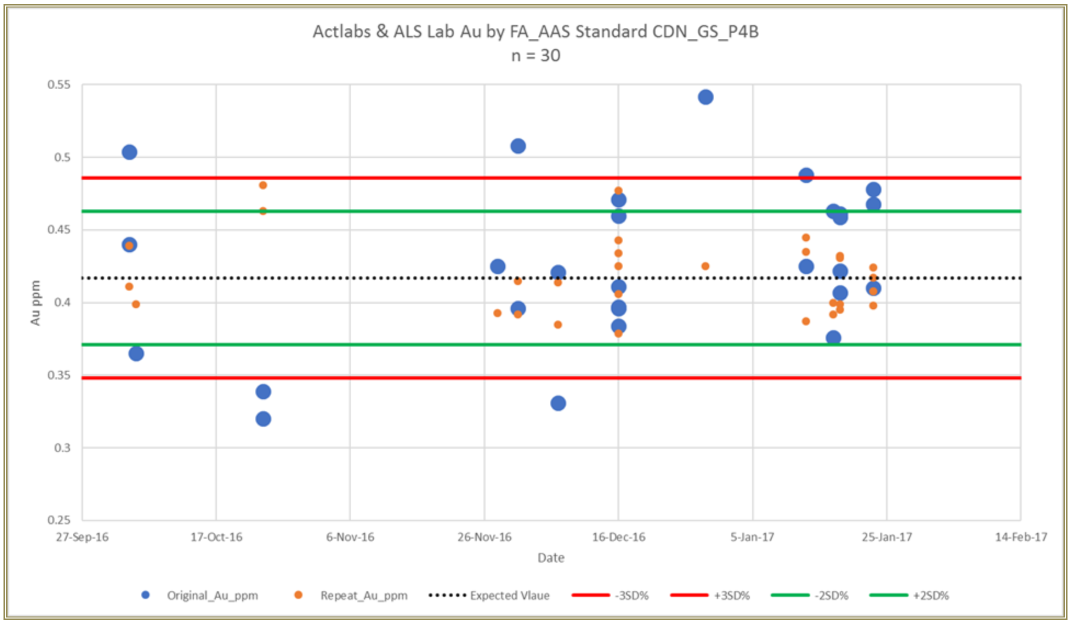


Figure 11-28: Control Chart—Standard CDN-GS-8A: Outliers Included—Key Lake Drilling Program (2011)

A representative number (15%) of Brookbank assay batches with standards that exceeded ±3 SD limits were selected and sent for re-assay at GGM's Umpire laboratory (ALS Minerals). The certified reference material and 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 FA by AA finish analysis, and to confirm the precision of the original results. All results returned from ALS laboratory were within the recommended limits of ±3 SD of the expected value for all standards submitted.



Source: GGM QA/QC Report (2016).

Figure 11-29: Control Chart of Certified CDN-GS-P4B—ALS vs. Actlabs Laboratory Checks

Analytical Blanks

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

In 2016 at Brookbank, 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/t Au (10X DL), 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 blank samples analyzed for gold produced a result over 10X 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 QC 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 DL**				**Mean Gold**	**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**					

Notes: CV = Coefficient of variations; FA_AAS = fire assay–atomic absorption spectroscopy; SD = standard deviation.

For all the Brookbank historical resampling, channel sampling, and development and 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/t Au. 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 was 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 re-run results returned from the ALS Mineral repeats within the expected range, well above the upper limit of 0.05 g/t Au, except for sample 265195.

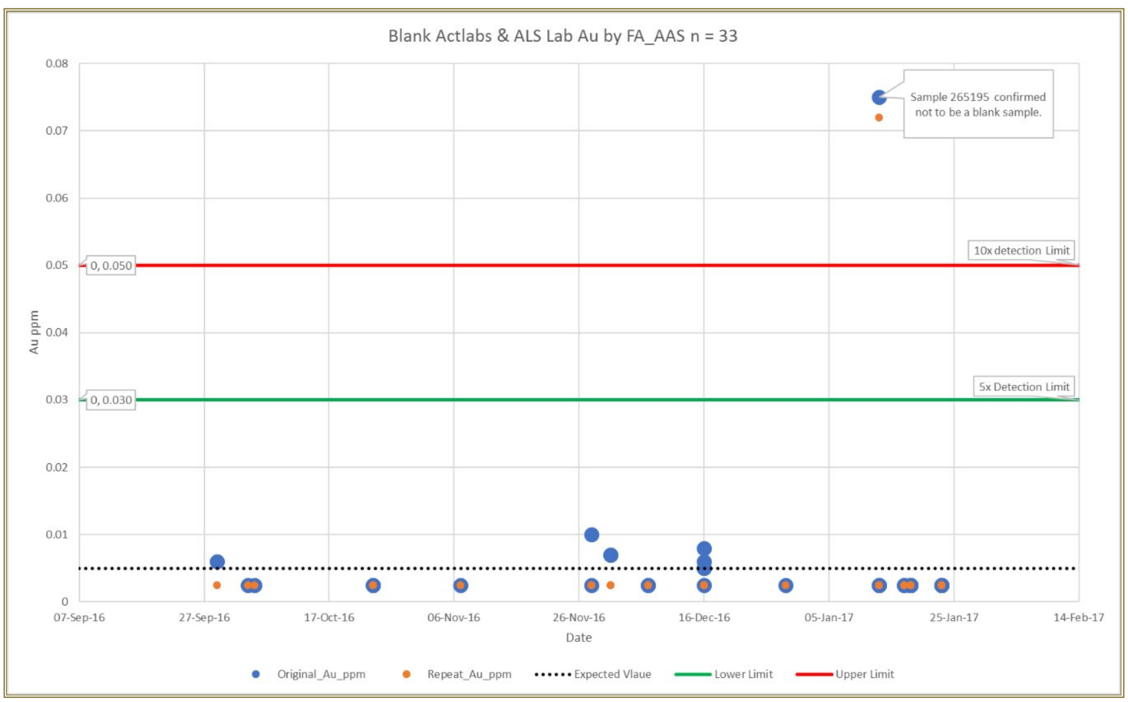


Figure 11-30: Umpire Repeat Assays—Actlabs vs. ALS—Brookbank Property

Analytical Duplicates

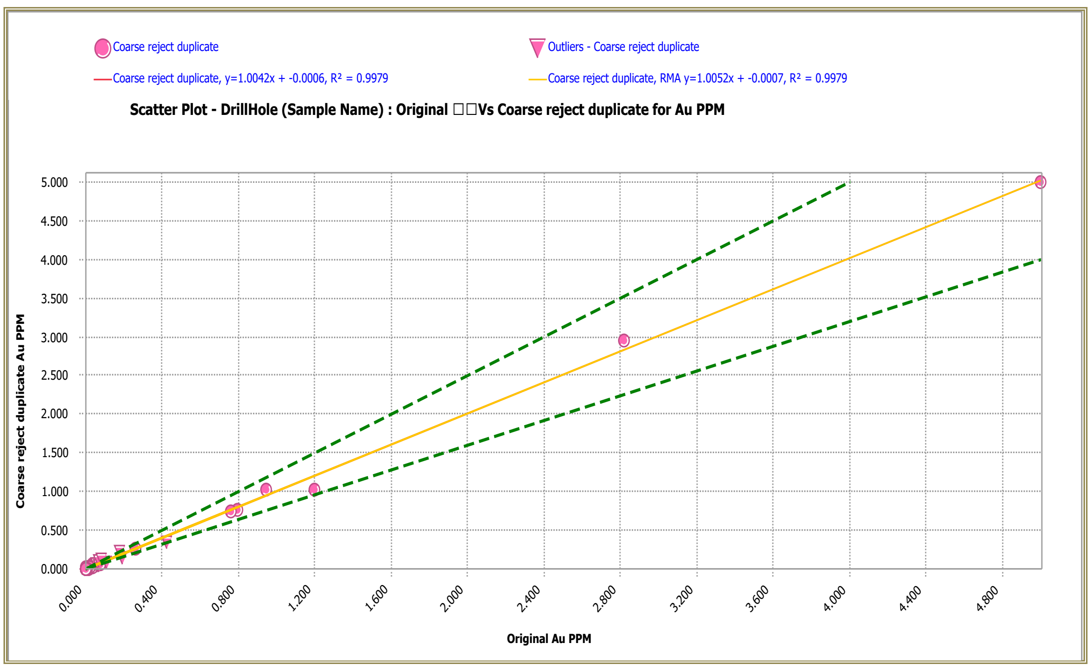
Premier used split core for its duplicate samples until October 3, 2012. After this, the duplicate samples consisted of two samples taken from the same piece of core which was crushed and riffle split at the assay laboratory. A combined total of 5,906 check assays (6.1%) were conducted by Premier for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

From 2012 to 2016, GGM QC 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 sub-samples taken after primary crushing is adequate for the given crushed particle size to ensure a representative sub-split.

As mentioned in Section 11.1.4, 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 reject duplicates, showing 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.



Source: GGM QA/QC Report (2016).

Figure 11-31: Scatter Plot—Drill Hole (Sample Name: Original vs. Coarse Reject Duplicate for Au (g/t) FA AAS Analytical Method

Of the 231 coarse reject duplicates, eight 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 g/t Au. For gold values >0.5 g/t there are no outliers, and they show good reproducibility.

Table 11-14: Duplicate Gold Results with a Precision >20% (Control Limit)—Brookbank QA/QC Program

Sampled Batch No.	Area	Sample No.	Duplicate Sample No.	Duplicate Type	Analytical Method	Original Gold Value (g/t)	Duplicate Gold 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

Notes: FA_AAS = fire assay–atomic absorption spectroscopy.

The duplicate sample program performed at Kailey from 2007 to 2011 is 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.

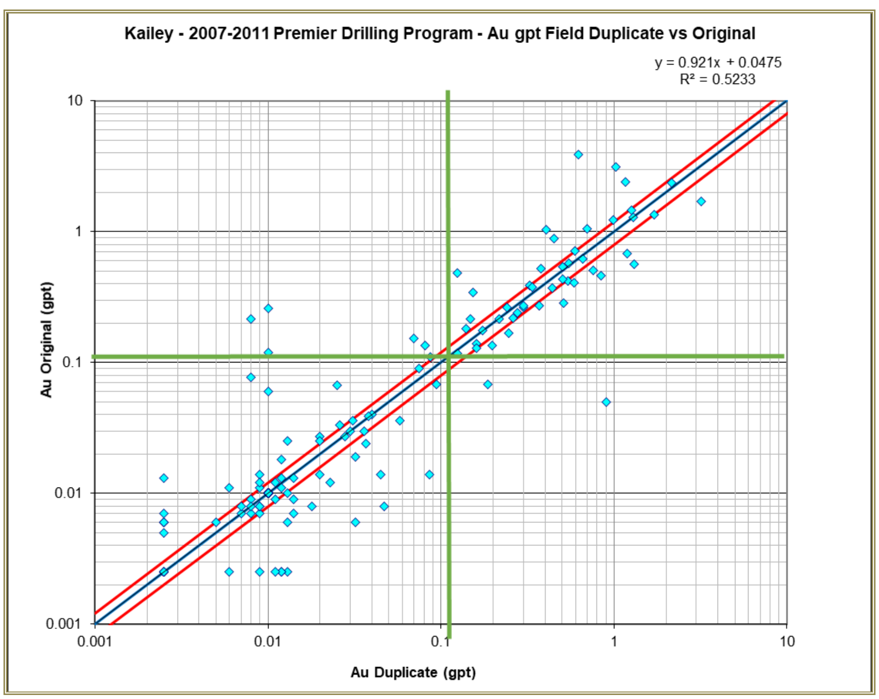


Figure 11-32: Field Duplicates Control Chart—Kailey (2007 to 2011)

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.

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., and Réjean Sirois, P.Eng., of GMS inspected sampling facilities and core storage areas, which is discussed further in Section 12.

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 lengthwise) using a diamond saw at its core logging and cutting facility in Geraldton. Samples of halved drill core were sealed in labelled plastic sample bags and securely packed for shipping. Bags of samples have been shipped by road to Actlabs 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 Properties, and considers that the drilling and assay database are suitable for use in the estimation of a Mineral Resource.

12 DATA VERIFICATION

This section is an update from the 2021 Report, issued January 26, 2021. As the 2021 MRE database formed the basis of the 2024 MRE reporting, this information is still valid and has been included for the sake of completeness.

12.1 Hardrock Data Verification

12.1.1 Data Verification for the 2016 Mineral Resource Estimate

The DDH database used for the 2016 MRE presented in this Technical Report was provided by 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 MRE 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 DDHs. These were added to the GGM database for the MRE update in this Technical Report. The 2014–2015 stripping program is also included in this update.

GMS's 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 1 and 4, 2016; and more recently July 23 to 25, 2024.

Historical Work

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

Greenstone Gold Mines Database

GMS was granted access to the certificates of assays for all holes in the latest drilling programs, 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.

Greenstone Gold Mines Diamond Drilling

The historical surface drill-hole 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.

Greenstone Economic Development Corporation (GEDC) compiled underground drill holes. However, these holes were excluded from the current MRE because the location data are considered unreliable, and the assay results could not be verified.

Downhole surveys were conducted on the majority of the surface holes using REFLEX survey instrumentation. GMS verified the survey data 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.

Figure 12-1: Drill-Hole Collars Surveyed during GMS 2016 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

	Check		Database		Difference	
Hole-ID	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

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

Independent Re-Sampling

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 random order inside relevant mineralized intercepts. For each zone and drill hole, one sample was collected at around the 20 m interval, when possible. Only samples grading more than 1.0 g/t Au were selected, and 50 cm of quarter-core splits were collected randomly in the sample interval.

In all, 16 samples were assayed for gold using fire assay with AA finish. Samples assaying more than 5 g/t Au with AA were re-run 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/t Au)	Lab Check (g/t Au)	Check Sample No.
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

DDH	Zone	From (m)	To (m)	Length (m)	Original Grade (g/t Au)	Lab Check (g/t Au)	Check Sample No.
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 comparison of the original samples and the field duplicate samples for all 16 samples. This graph shows 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 eight samples all show a significant decrease in gold grades, ranging from 75% to near 100%. Since one-sixth 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.

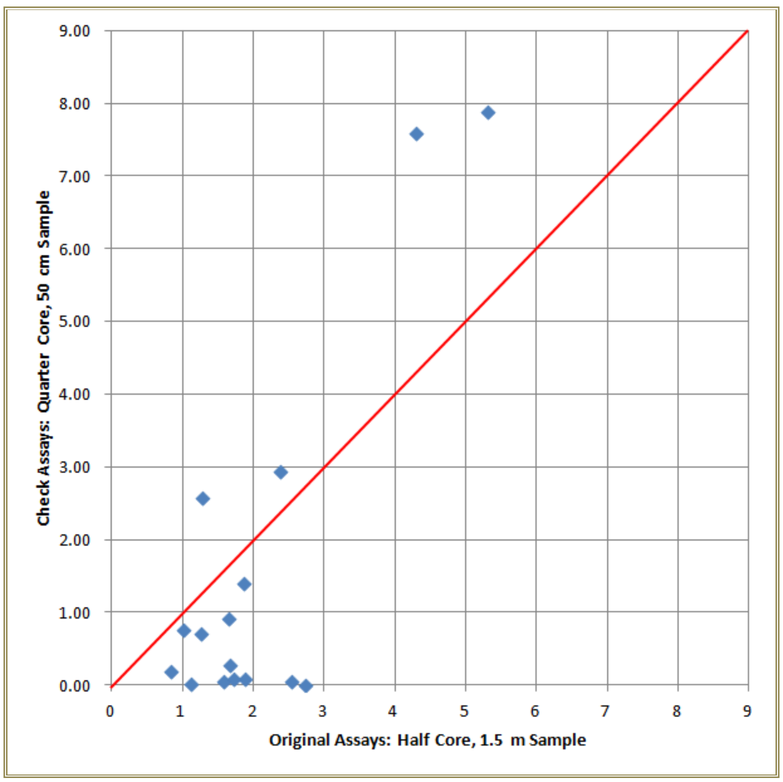


Figure 12-3: Original Assays Compared to Check Assays

Mined-Out Voids

Considerable effort has been made to improve the accuracy of the stope and drift 3-D 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 3-D 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 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.

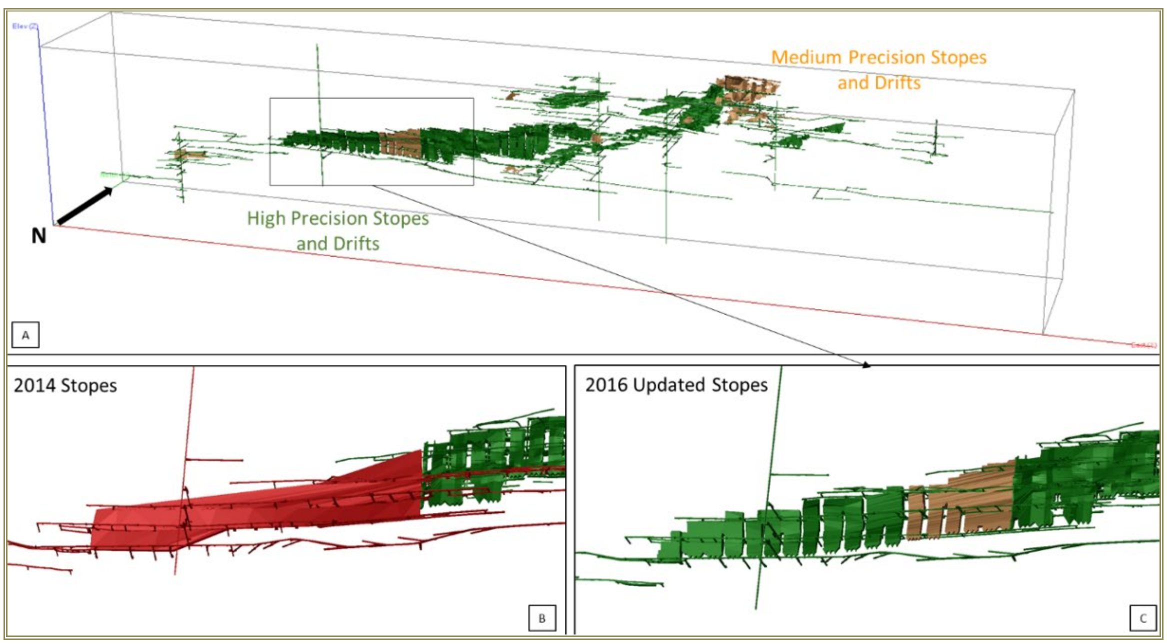


Figure 12-4: Isometric View Looking NNW Showing a Compilation of the Mined-out Underground Voids: A) Overall View of Stopes and Drifts by Level of Precision; B) Close-up View of the Stopes Modelled in 2014; C) Close-up View of the Stopes Updated in 2016

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.

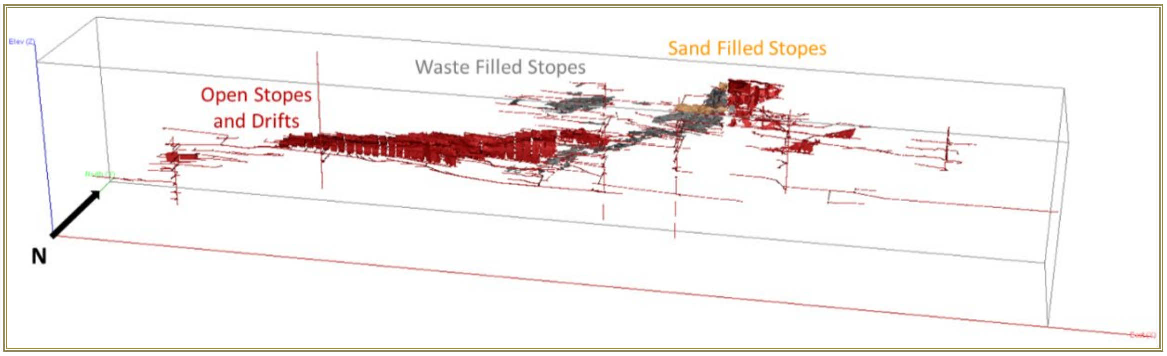


Figure 12-5: Isometric View Looking NNW Showing a Compilation of the Mined-out Underground Voids Based on their Backfill Type

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.

Conclusion

Overall, GMS believes that the data verification process demonstrated the validity of the data and protocols for the Greenstone Mine. GMS considers the GGM database to be valid and of sufficient quality for use in mineral resource estimation.

12.1.2 Data Verification for the 2019 Mineral Resource Estimate

Drilling activities at the Hardrock deposit resource area ended on April 25, 2019, and the database close-out date for the MRE 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 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 DDHs (12,009 m) were drilled in the Project during 2019. A single failed DDH was excluded from the database. All data collected from these drill holes were incorporated into the drilling database for the current MRE update.

GMS's data verification for the Greenstone Mine consisted of numerous site visits to monitor drilling activities, reviewing new drill-hole data merged into the 2016 MRE database, reviewing new voids, and new lithology, alteration, and structural data. Finally, the verification also included a comparison of the RCGC assay grades to 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:

- May 24–25, 2018
- October 03–04, 2018

- February 25–26, 2019
- April 10–11, 2019
- May 08–09, 2019
- July 29–30, 2019.

2019 Mineral Resource Estimate Drilling Database

The 2019 MRE database contains 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 2019 MRE.

The database includes drilling totalling 536,850 m assayed for gold taken from 722,086 m of RC and DDH programs (Table 12-3). The 26 channel samples totalling 1,323 m that were used in the 2016 MRE have now been excluded and are not used in the 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
DDH Subtotal	**1,682**	**696,125**	**513,245**
2018 RCGC	405	20,015	18,050
2019 RCGC	76	5,946	5,555
RCGC Subtotal	**481**	**25,961**	**23,605**
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 IDs and collar surveys for the 2018 RCGC drilling. Of the 52 collars that were revisited, only six 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 do not have a material effect on the 2019 MRE, but recommend that they be corrected for future MREs if drill collars are still available. 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. 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.

No significant errors or issues were 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 Mineral Resource estimation.

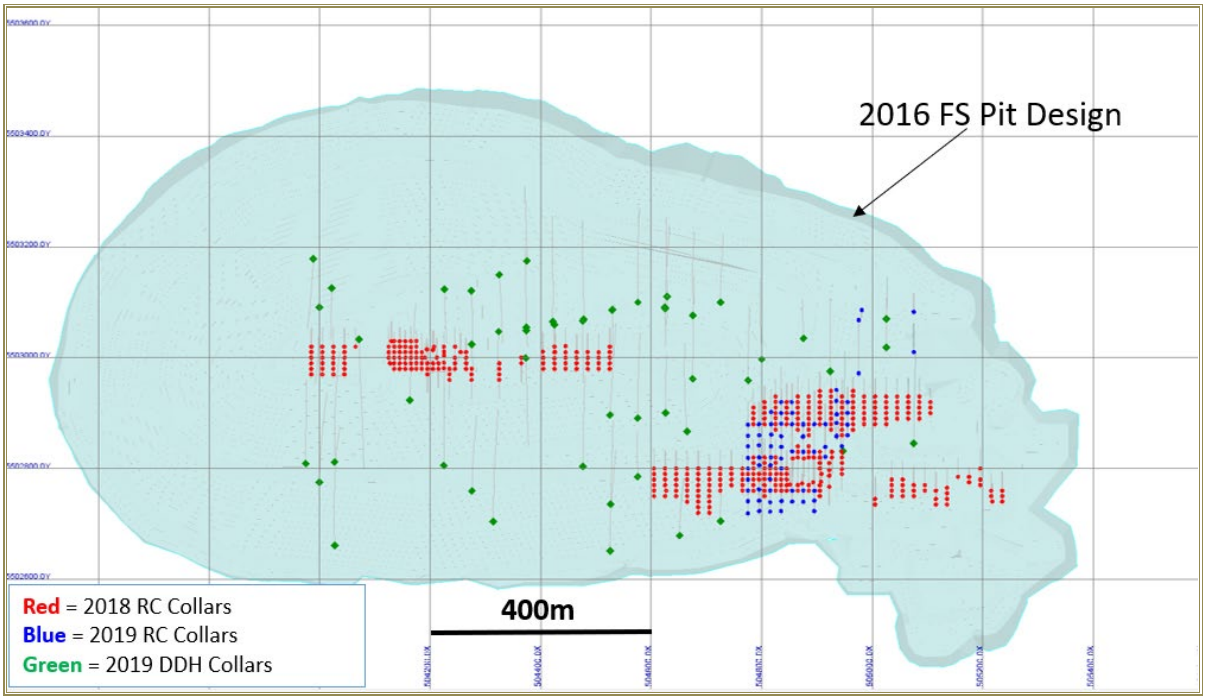


Figure 12-6: 2018–2019 Drilling Programs—Greenstone Mine

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, 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.

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 working wireframes. The wireframes representing mined-out areas were last updated during the 2016 MRE. Since then, no additional work has been done to provide a more accurate representation of the mined-out volumes in the historical underground workings.

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

- 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 historical 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. Figure 12-9 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.

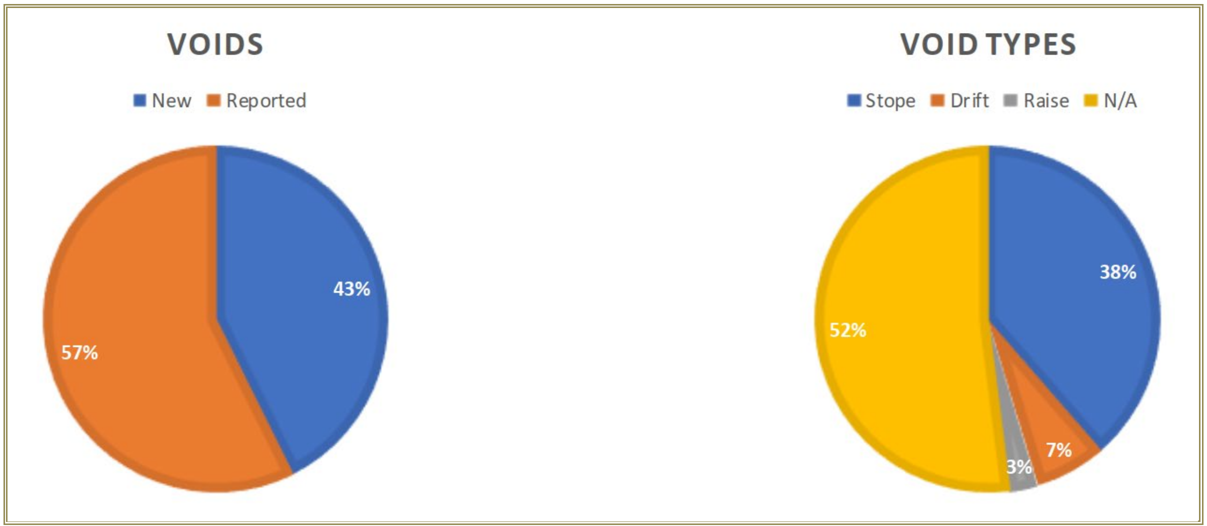


Figure 12-7: Mined-out Voids Status as Intercepted by RC and DDH Drilling Programs

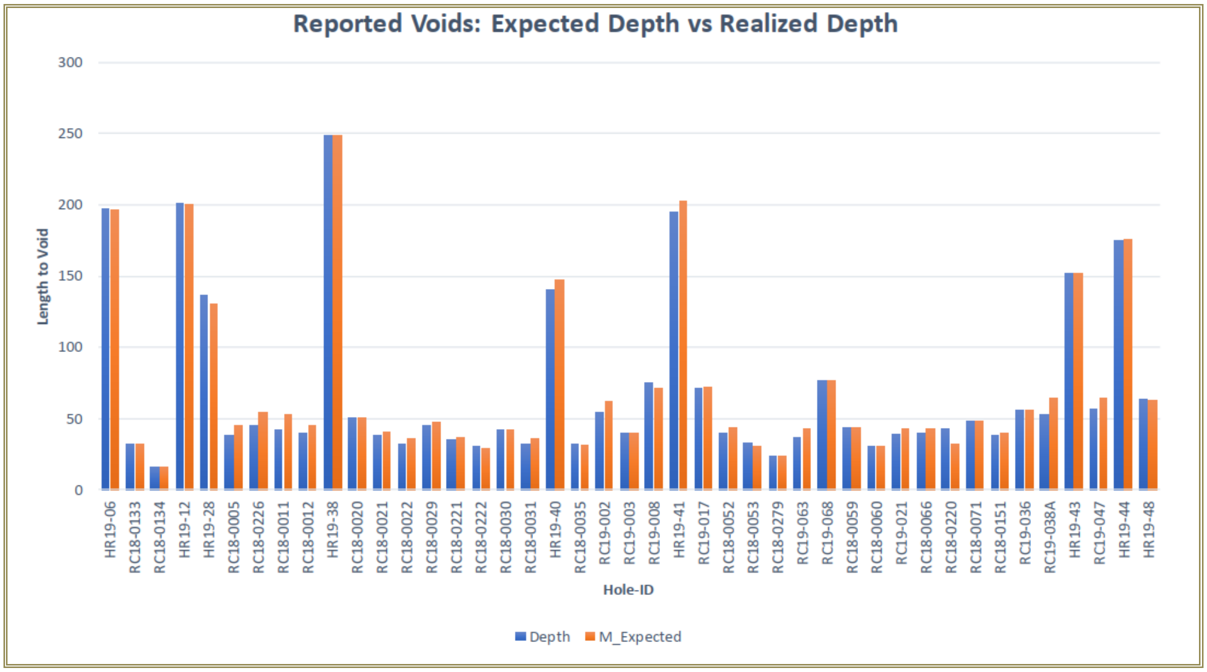


Figure 12-8: ***Reported Voids: Expected Length vs. Actual Length***

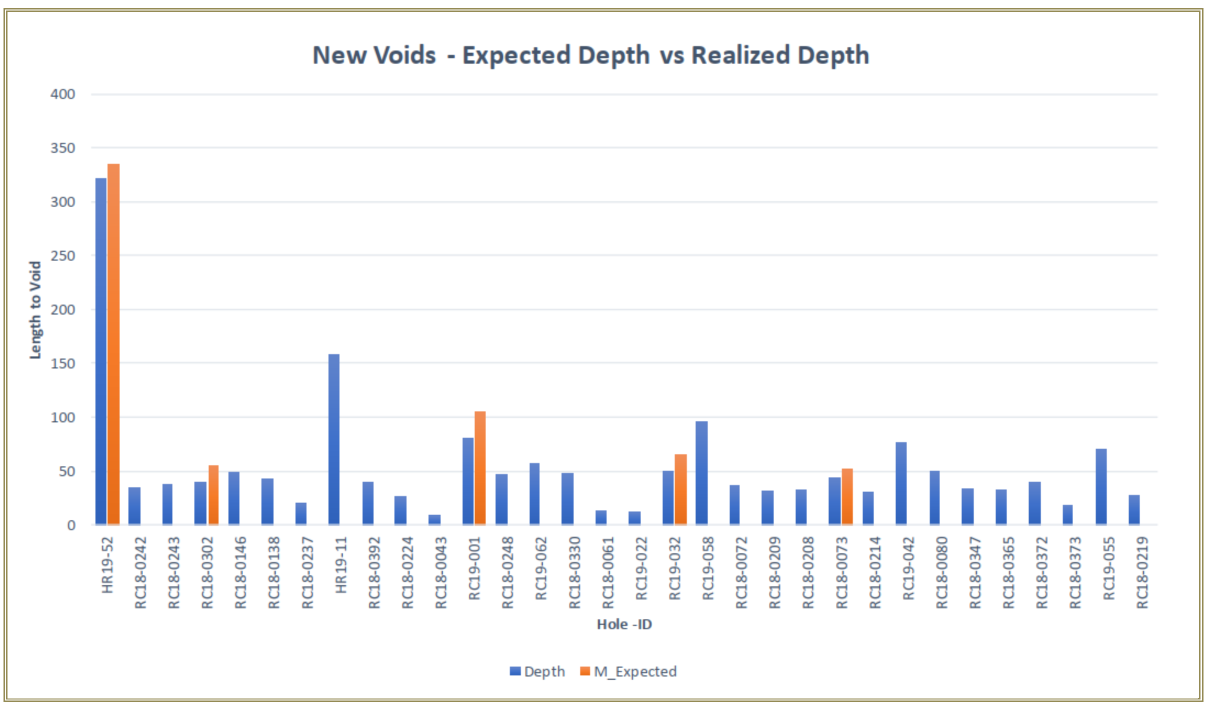


Figure 12-9: ***New Voids: Expected Length vs. Actual Length***

2019 Voids Model Update in Hardrock Gems Project

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

GMS's observations regarding the drill-hole intercepts in relation to the historical underground workings are as follows:

- For 2018 RCGC drilling, 47 holes intersected historical openings (or mined-out voids). Some 26 of the 47 RC holes are classified as "new voids," since the intercepted interval was not expected (Figure 12-10).
- For 2019 RC drilling, 22 holes have crossed over historical openings. 12 of the 22 RC holes are now classified as new voids, and 6 out of the 22 are classified as "to expand" since the intercepted interval had a different depth compared to the expected depth in the GGM database.
- 2019 diamond drilling has a total of 12 DDHs that have intersected historical workings. Only one DDH has been classified as "to expand" because this opening has not intersected the historical void as expected. All the other DDHs have intercepted a nearby historical opening.

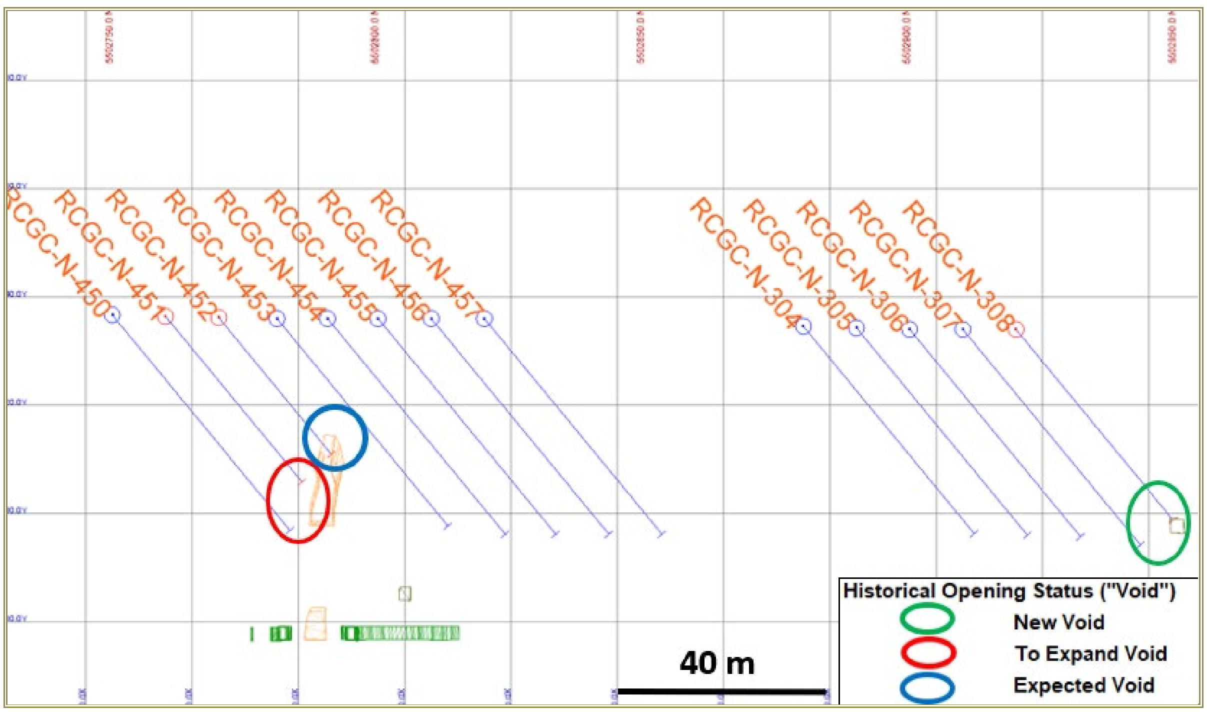


Figure 12-10: 2018 RC Holes vs. Historical Openings—Section 4805E (Looking West)

Further iterations should be undertaken to improve the overall accuracy of the historical underground workings. Several drill holes encountered voids near 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 underground voids was removed from the 2019 MRE.

12.1.3 Data Verification for the 2022 and 2024 Mineral Resource Estimate

A new drilling database export was supplied to GMS on March 23, 2022. The previous database used by GMS was a patchwork of the original 2016 drilling database along with two subsequent drilling campaigns (separated into RC and DDH databases) which became cumbersome and difficult to work with. GMS excluded un-assayed geotechnical holes, blastholes, abandoned holes with redrills, and channels from the MRE, consistent with the 2019 approach.

QA/QC results were reviewed pertaining to the winter 2021–2022 drilling campaign with GGM geologists on site; no material issues were found. Any QA/QC failures resulted in the reanalysis of the batch according to the GGM internal QA/QC protocols.

A summary of the drilling database by drilling type supplied to GMS is shown in Table 12-4, and a plan view is shown in Figure 12-11.

Table 12-4: Drilling Database Statistics—All drilling

Drilling Type	Number of Drill Holes	Total Length (m)	Assayed
Borehole (vertical)	90	965	684
Channel	31	1,513	1,498
Diamond	1,846	738,232	462,540
Reverse Circulation	549	30,183	27,389
Total	**2,516**	**770,893**	**492,110**

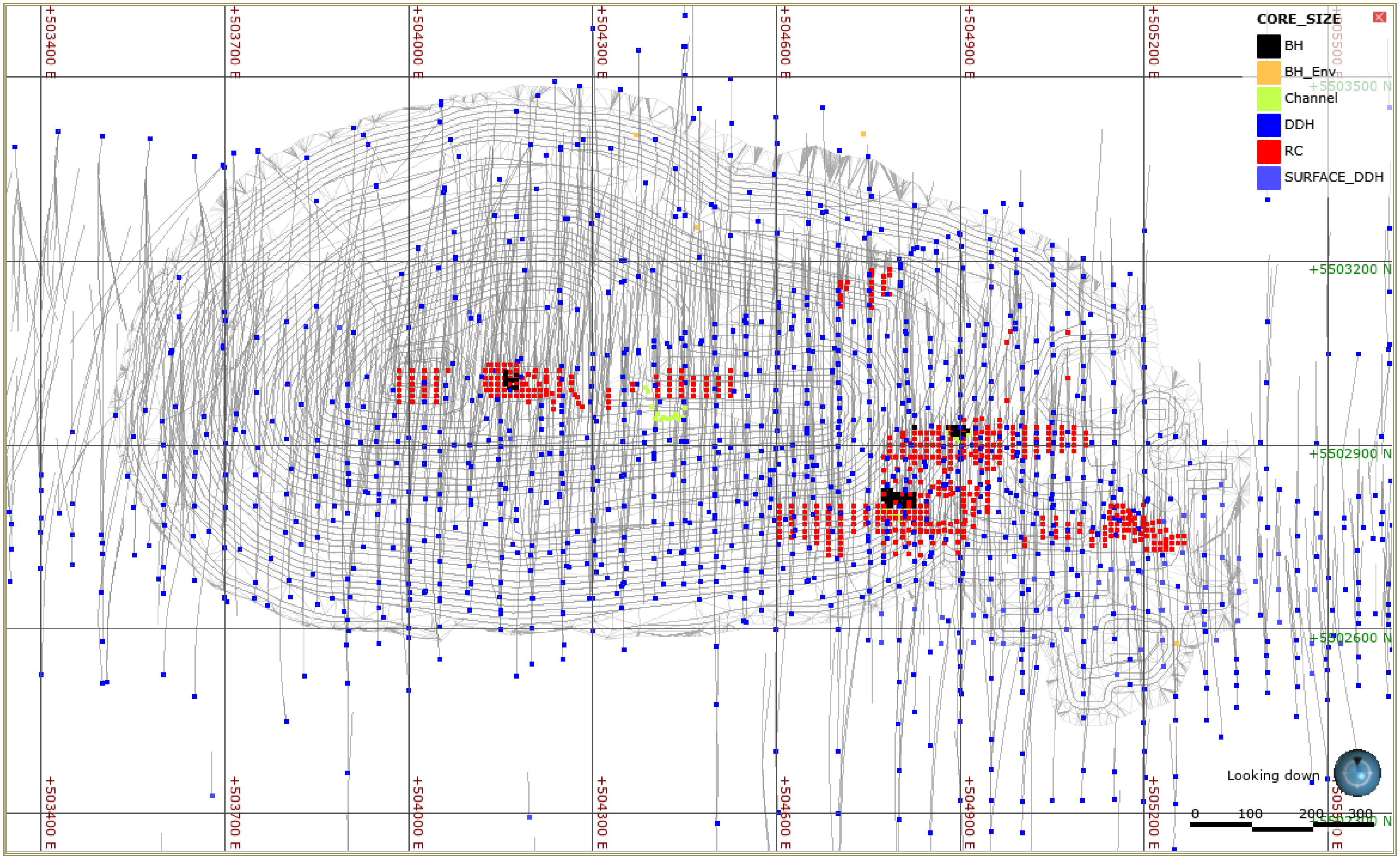


Figure 12-11: Plan View of all Drilling used in the 2022 MRE Update Coloured by Drilling Type—Background is the 2019 Feasibility Pit Design

12.1.4 Data Validation Conclusions and Recommendations—Hardrock Deposit

Overall, GMS believes that GGM's protocols for drilling, sampling, analysis, security, and database management meet industry standards. The 2022 and 2024 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 Mineral Resource estimation.

12.2 Brookbank, Kailey, and Key Lake Data Verification

12.2.1 2020 Site Visit

For the 2020 update of the Brookbank, Kailey, and Key Lake MREs, James Purchase, P.Geo., of GMS conducted a site visit to the three properties from July 27 to 30, 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 and 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 was able to locate collars for drill holes 11-85, -86, -87; PLL 08–13, -32, -30; 16-BB-001; B99-03, -04. The collars were found to be within ±5 m accuracy of the database coordinates when using a handheld GPS. Figure 12-12 shows the Key Lake Property and a drill collar found on the Kailey Property.

Figure 12-12: 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 removed 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 QA/QC 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 2024 Site Visit

The current MRE QP, Réjean Sirois, P.Eng., of GMS, conducted a site visit to the three properties from July 23 to 25, 2024. The following activities were undertaken:

- Review of core storage and sampling facilities
- Verification of a few drill-hole collars for Brookbank and Key Lake deposits
- Walk on a few outcrops and observe geological features
- Examination of drill core and 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 a few drilling logs and assay result sheets.

GMS was able to locate collars for drill holes BB99-04, BB06-03, and 16-BB-001. The collars were found to be within ±5 m accuracy of the database coordinates when using a handheld GPS.

Figure 12-13: Drill-Hole Collar at the Brookbank Deposit (Photo Taken on July 23, 2024)

Figure 12-14: Mine Workings—Capped Jellicoe Shaft, Key Lake Property

12.2.3 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 and repeat assays in Figure 12-15 confirm the laboratory's high degree of accuracy (lack of bias) and precision, with the exception of one mismatch, which is attributed to mistaken sample switch.

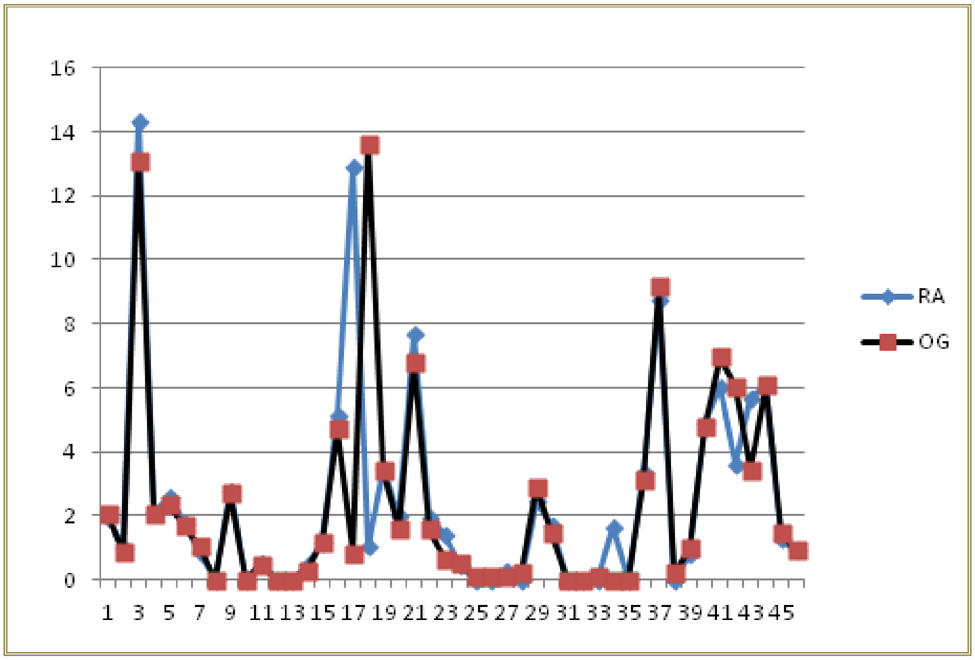


Figure 12-15: Comparison of Original (OG) and Repeat Analyses (RA)

12.2.4 Database Validation

Since the 2012 MRE, there has been a significant effort by GGM staff to increase 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 several 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 in 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 in 2022 and 2024 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, suffer from selective, incomplete sampling of drill core, which has likely resulted in intervals of low-grade mineralization remaining unsampled. In all such zones, GMS has assigned a detection limit assay value of 0.001 g/t Au. This will likely result in an understatement of the ounces, but it is an industry-standard approach when faced with an 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 applying a 0.001 g/t grade.

12.2.5 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 3-D 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 digitized and pinned to the collar coordinate. At Kailey, underground drifts were digitized 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 12-16.

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

12.2.6 Data Verification Conclusions and Recommendations—Brookbank, Kailey, and Key Lake Deposits

Considering the data verification that has been undertaken on the three properties by GMS, and previously by Micon and Scott Wilson, GMS believes that the GGM database is suitable for use in the current MREs. 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/t Au 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.

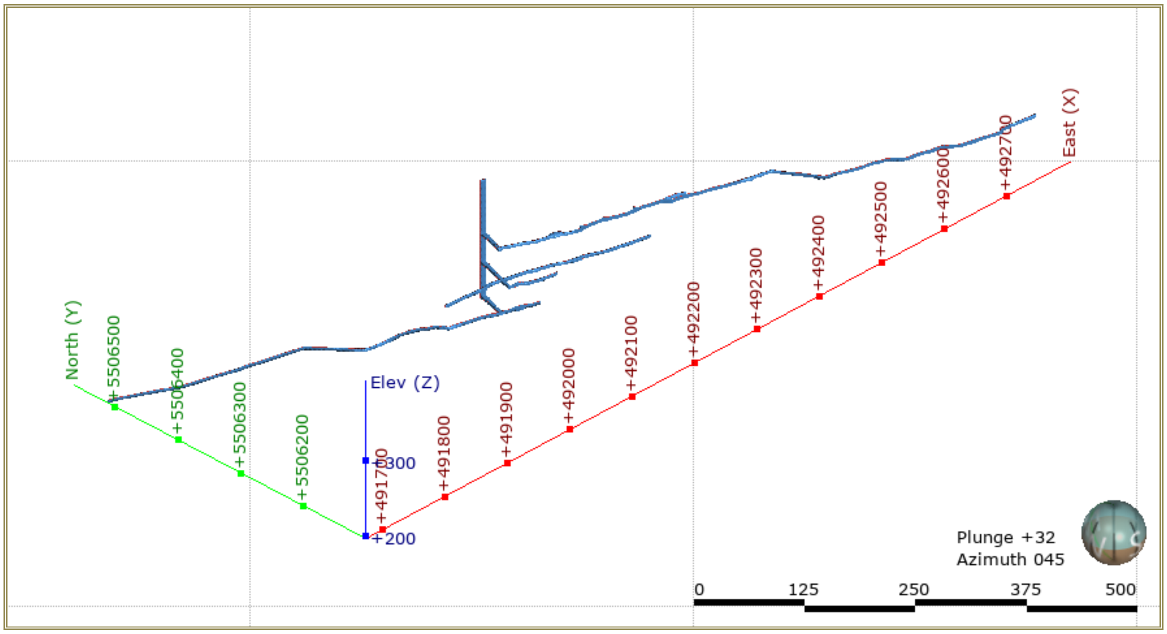


Figure 12-16: Top, Jellicoe Shaft and Underground Drives at Key Lake; Bottom, Little Long Lac (Kailey) Shaft and Underground Drifts, with the Mined Vein Wireframe

13 MINERAL PROCESSING AND METALLURGICAL TESTING

This section summarizes all the relevant testwork performed on the Hardrock deposit: all testwork completed after 2011, including that completed during the GGM Feasibility Study (Pierre Roy, 2016) and during the detailed engineering phase.

13.1 Previous Testwork

Some mineralogy, grindability, and gold recovery testwork was performed prior to the start of the feasibility study. The key reports from 2011 to 2013 are summarized in this section. The reference documents are:

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

13.1.1 Gold Recovery Testwork at SGS Lakefield (Phase 1)

Samples were sent to SGS Lakefield Research Limited in March 2010. Composites 1 and 2 were prepared and subjected to head analyses; mineralogy; Bond work index determination; gravity separation; gravity-tailings flotation and whole ore; and gravity-tailings and flotation-concentrate cyanidation.

Head Assays

Composites 1 and 2 were submitted for gold analysis according to the metallic sieve protocol; each provided two 1 kg samples of each composite—1 (A) and 1 (B); 2 (A) and 2 (B)—which were submitted for coarse gold analysis (±106 µm or 150 mesh fractions). The fine fraction was assayed in duplicate (in Table 13-1 under "−106 µm Gold Analysis, Assay a and Assay b").

Table 13-1: Gold Head Analyses by Metallic Sieve

Sample	Head Grade Gold (g/t)	+106 µm			−106 µm Gold (g/t)		
		Distribution		Gold (g/t)			
		Mass (%)	Gold (%)		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 the coarse fraction gold content between Composite 2 (A) and Composite 2 (B) suggest the presence of fine free gold.

Mineralogy

A sample of each composite underwent an analysis of the rock-forming components using transmitted-light microscopy, XRD, chemical analysis, and SEM techniques. Table 13-2 lists Composite 1 and Composite 2 constituents. Other trace constituents include iron-titanium 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

Grindability Testwork

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. According to SGS Lakefield's database, Composite 1 falls into the moderately hard category while Composite 2 can be considered medium hardness.

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

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.

Flotation

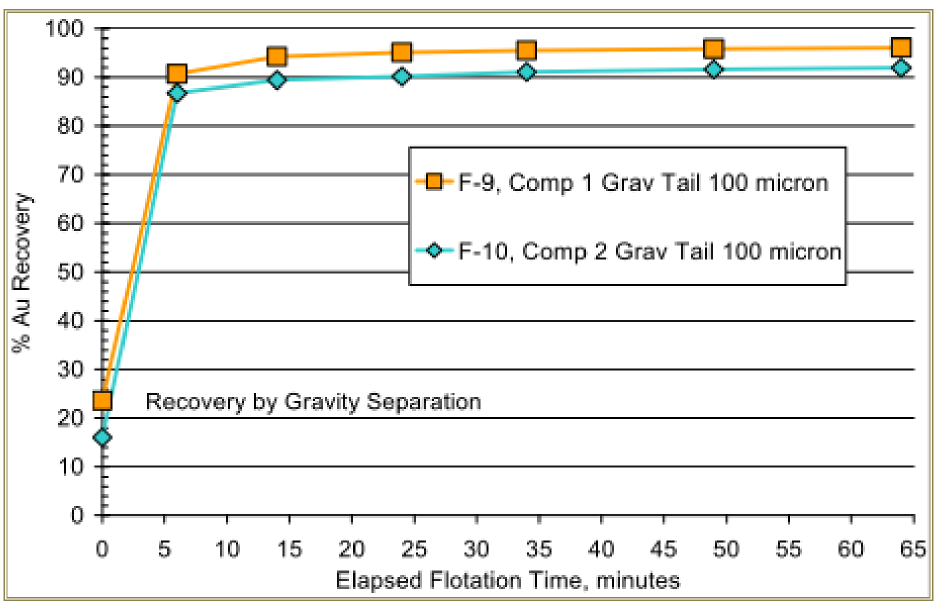
The gravity tailings were subjected to flotation testing. The objective of the initial kinetic rougher flotation tests was to evaluate the impact of grind size on gold recovery and determine the test conditions required to generate bulk concentrate for further testwork. The purpose of the tests was to recover gold in a sulphide rougher concentrate.

Flotation tests were carried out at grinds of P_{80} 47, 70, and 95 µm for Composite 1. A concentrate mass recovery of 17% to 41% was achieved with gold grades ranging from 16.4 g/t Au at 95% overall recovery (coarsest grind) to 7.3 g/t Au at 98% overall recovery (finest grind). The tailings gold grade ranged from 0.19 g/t Au (P_{80} 95 µm) to 0.15 g/t Au (P_{80} 47 µm).

For Composite 2, flotation tests were performed at grinds of P_{80} 51, 75 and 108 µm. Approximately 23% to 33% mass was recovered to the concentrate. Concentrate gold grades ranged from 11.3 g/t Au at 93% overall recovery (P_{80} 108 µm) to 9.1 g/t Au and 95% overall recovery (P_{80} 51 µm). The tailings gold grade ranged from 0.29 g/t Au (P_{80} 108 µm) to 0.22 g/t Au (P_{80} 51 µm) .

Bulk flotation tests were conducted to generate concentrate for cyanidation. The results of the 10 kg bulk tests on the P_{80} 100 µm gravity tailings were comparable to the 2 kg flotation tests on similar feed. The correlation between gold (non-gravity recoverable) and sulphide sulphur recovery indicates an association.

Figure 13-1 presents the results of the bulk flotation tests on Composite 1 and Composite 2.



Source: SGS Lakefield (2011, March 1).

Figure 13-1: Gravity Tailings Bulk Rougher Kinetics Results

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 P_{80} >≈70 µm. Below this size, the gain in recovery was less significant.

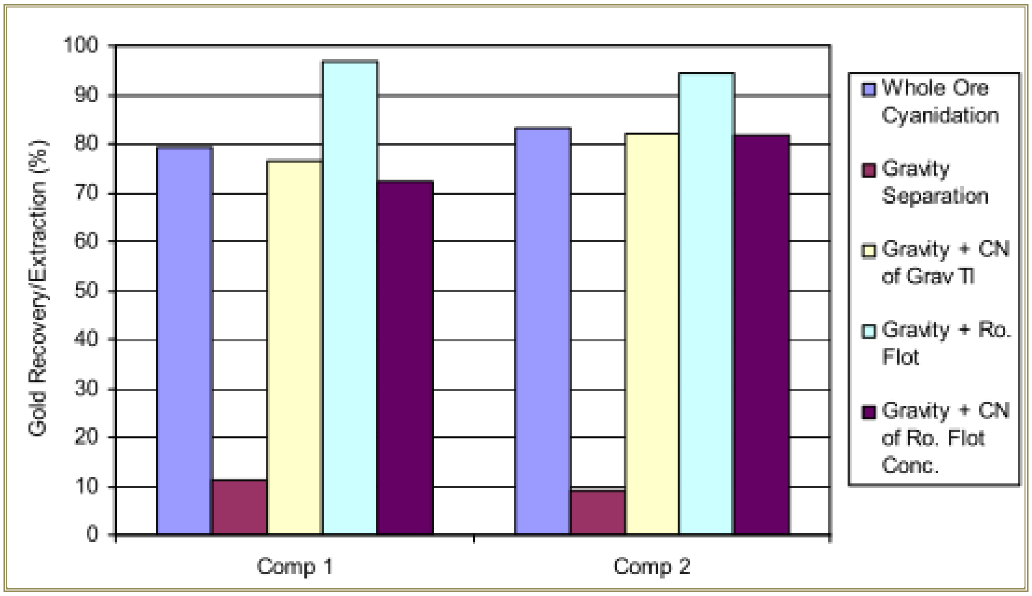
Gravity-tailings cyanidation tests aimed to determine ore amenability to cyanide leaching and also examined the effect of regrind on gold extraction. Bottle roll tests were performed under the same conditions as the whole-ore tests. Cyanide extraction ranged from 64% to 70% for Composite 1 and between 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 (P_{80} 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 summarizes the results of the whole ore, gravity tailings, and flotation-concentrate cyanidation tests. 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 test 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/t Au)
		Reagent Consumption of Whole Ore (kg/t)		Grind P_{80} (µm)	Recovery (%)		
		NaCN	CaO				
CN-2	Composite 1 Whole Ore	1.5	0.4	59	79.2		1.01
CN-8	Composite 1 Gravity Tailings	0.8	0.5	68	62.1	73.3	0.99
CN-14	Composite 1 Flotation Concentration	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 Tailings	0.8	0.8	68	72.9	82.1	0.53
CN-16	Composite 2 Flotation Concentration	1.0	1.0	11	65.8	81.9	0.52



Source: SGS Lakefield (2011, March 1).

Figure 13-2: Comparison of Combined Results

Gravity separation followed by gravity-tailings cyanidation achieved similar results to whole-ore cyanidation. Gravity separation followed by flotation yielded the highest recoveries but assumed that the flotation concentrate could be sold as smelter feed.

13.1.2 Gold Recovery Testwork at SGS Lakefield (Phase 2)

Samples were sent to SGS Lakefield in May 2011. This second phase of work followed the previous testwork 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. In addition, gravity-tailings flotation and flotation rougher-concentrate cyanidation tests were included in the program.

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 (in Table 13-5 under "−106 µm Gold Analysis, Assay a and Assay b").

The variations in gold content in the coarse fraction between Composite IF1 and Composite P2 and in gold content in the coarse fraction of Composite IF1 (A) and IF1 (B) and between P2 (A) and P2 (B) suggests the presence of fine free gold.

Table 13-5: Gold Head Analyses by Metallic Sieve

Sample	Head Grade (g/t Au)	+106 µm			−106 µm (g/t Au)		
		Distribution		Au (g/t)	Mean	a	b
		Mass (%)	Au (%)				
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

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 scanning electron microscopy–energy dispersive spectroscopy (SEM–EDS). The majority of gold in the samples occurred 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.

Grindability Testwork

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.

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 P_{80} 570 µm. A 14% recovery was 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.

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.

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/t Au and an 85% recovery. After regrinding to P_{80} 45 µm, the mass pull was 14%, gold grade was 27.0 g/t Au, and recovery was increased to 89%.

For Composite P2, the second cleaner concentrate showed 7% mass pull, a grade of 54.8 g/t Au, 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/t Au and 22.0% sulphide was achieved with a 92.5% gold recovery and a 94.8% sulphide 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 Concentrate Gold Grade (g/t)
				Recovery (wt %)	Gold Grade (g/t)	Gold Recovery (overall) (%)	
Composite IF1	Whole Ore	F1	n/a	12.5	30.3	84.6	n/a
		F3	45	13.8	27.6	88.7	n/a
	Gravity Tailing	F5	n/a	8.5	28.8	89.1	8.32
		F6	23	7.9	28.7	88.6	
		F7	10	8.1	31.8	91.3	
Composite P2	Whole Ore	F2	n/a	7.3	54.8	81.3	n/a
		F4	22	8.5	56.8	87.8	n/a
	Gravity Tailing	F8	n/a	5.9	48.9	88.6	1.21
		F9	19	4.9	54.2	87.4	
		F10	9	4.8	58.6	86.2	

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 Tailings	1,291.1	16.1	0.72	1.14	2.2	4.3
Rougher Tailings	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

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 testwork program. Only 70% of the sulphides 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 P_{80} 123 µm, pressure oxidation increased gold extraction to 97%, with a 94% overall gold recovery (including the flotation stage).

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 methods were varied in the flotation-concentrate cyanidation tests. The effects 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)	72 hour Au Extraction (%)	Recovery		Residue/Tailings Grade		CN Consumption (kg/t)	
				Flotation (%)	Overall (%)	CN Au[1] (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: [1] Average of duplicate residue assays.

13.1.3 Whole-Ore Cyanidation Testing at McClelland

Thirteen drill-hole reject composites 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.

Results

The tests consisted of standard bottle roll tests with or without carbon addition. The direct head assays of the 13 samples ranged from 0.40 to 7.37 g/t Au with an average of 3.20 g/t Au. The cyanidation tests were performed on three different grind sizes: P_{80} 125, 75, and 37 µm. The tests performed at 75 µm and without carbon addition are summarized in Table 13-9.

Table 13-9: Whole-Ore Cyanidation Test Results

Sample	COrg (%)	S (%)	Au Recovery (%)	g/t Au Ore				Reagent Requirements kg/t Ore	
				Extracted	Tailings	Calc'd Head	Head Assay	NaCN Consumption	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

Note: Calc'd = calculated.

All 13 composites were amenable to cyanidation under the tested conditions. Gold recovery was between 85% and 95% for the composites with low sulphide-sulphur content (less than 2.5%). Three composites showed higher sulphide-sulphur 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 and 75 µm, and 4.3% between 75 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 2013. The results are presented in Table 13-10.

Table 13-10: QEMSCAN Modals on Global Composite

Survey Project/Low Intension Magnet Separation 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

Survey Project/Low Intension Magnet Separation Sample	Global Composite Mineral Mass (%)
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 Testwork

This section includes any testwork program that was performed during the preliminary economic assessment (PEA) and during the feasibility study. As the feasibility study progressed, additional testwork was initiated and is described here. Primarily, high-pressure grinding roll (HPGR) tests were required to confirm the ore amenability for high-pressure grinding, to select the equipment and to estimate the operating costs. The key reports from 2014 and 2015 are summarized in this section. The reference documents are:

- *An Investigation into the Grindability Characteristics of Samples from the Hardrock Deposit, Report 1 (Grindability)-14117-001, August 26, 2014* (SGS Canada Inc., 2014, August 26)
- *An Investigation into The Hardrock Deposit, Final Report 14117-001, October 8, 2014* (SGS Canada Inc., 2014, October 8)
- *The HPGR Amenability of Samples from The Hardrock Deposit, Report 2–Rev 1-14117-001, March 6, 2015* (SGS Canada Inc., 2015, March 6)
- *Revised SMC Test Report, April 2014* (JKTech Pty Ltd., 2014, April)
- *Thickening and Rheology Tests on Gold Ore Composite, June 2014* (FLSmidth, 2014, June).

13.2.1 Grindability Testwork

Five dilution samples, three PQ core samples and 53 core interval samples were submitted for comminution testing at SGS Canada Inc. in Lakefield. 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 (ROM) 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.

Grindability Tests Results

The grindability test 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 Sample Comminution Tests Results

Type	Name	Interval Number	CWI (kWh/t)	Relative Density	JK Parameters			Ball Mill Work Index (kWh/t)	Modified Bond Ball Mill Work Index (kWh/t)	Abrasion Index (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
	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

Notes: DWT = Drop Weight Test; SMC = SMC Testing Pty Ltd.

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	Modified Bond Ball Mill Work Index 90th Percentile (kWh/t)	Drop Weight Index 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 were submitted to comminution testing, using material from various lithologies representing the entire Hardrock deposit. 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		Bond Rock Work Index (kWh/t)	Ball Mill Work Index (kWh/t)	Modified Bond Ball Mill Work Index (kWh/t)
	Relative Density	Axb			
Average	2.98	29.2	16.5	14.9	14.4
Standard Deviation	0.21	3.4	0.2	1.0	1.2
Relative Standard Deviation	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 RWIs 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 Testwork

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 testwork. 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 Sample 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	-	-	-

Characterization and Recovery Tests Results

Head Grade Determination

Composite head grades were determined by metallic sieve analyses and a weighted average was calculated from the testwork (Table 13-15). The direct and calculated head grades all correlate well, except for Composites C and F.

Table 13-15: Composite Samples Direct and Calculated Head Grade

Sample Name	Direct (g/t Au)	Calculated (from Testwork) (g/t Au)
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

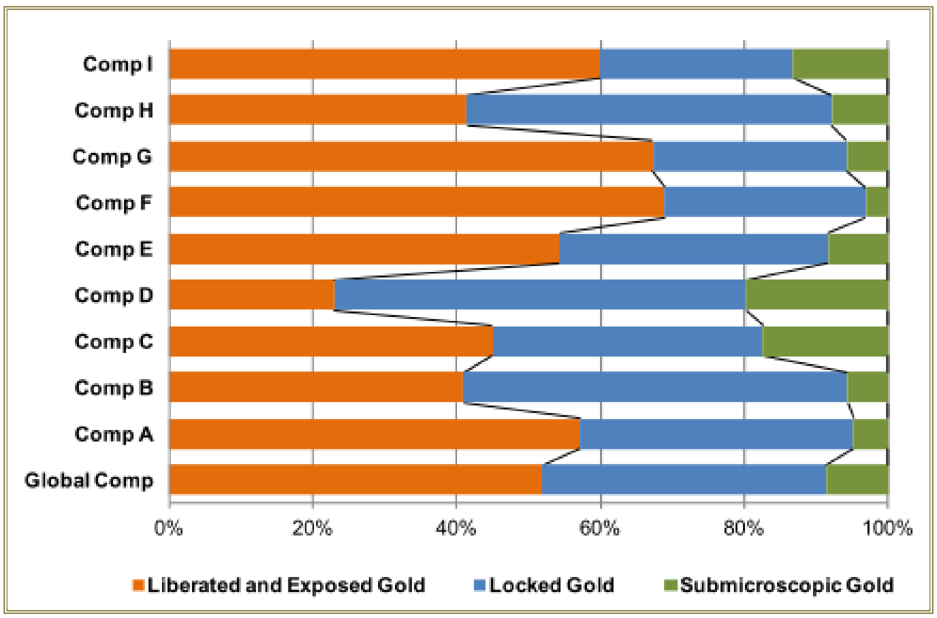
Mineralogy

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 P_{80} 300 µm is shown in Figure 13-3.

Table 13-16: Gold Deportment Results

Composite	Gold Distribution (%)		Gold Associated Minerals (%—Normalized to 100%)							
	Submicroscopic Gold	Microscopic Gold	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

Notes: 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.



Source: SGS Canada Inc. (2021, July 14).

Figure 13-3: Gold Occurrence (by Distribution)

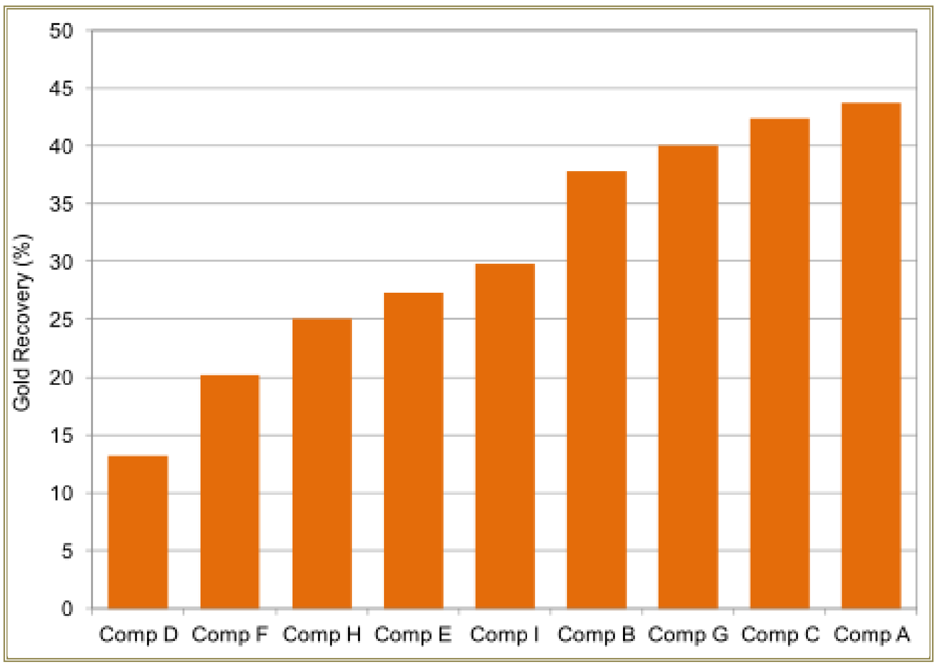
Magnetic Separation

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 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.

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 with 80% passing various grind sizes, 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 P_{80} 80 µm. The gold recovery varied from 13% to 44% (Figure 13-4).



Source: SGS (2008).

Figure 13-4: Composite Gravity Recovery Results

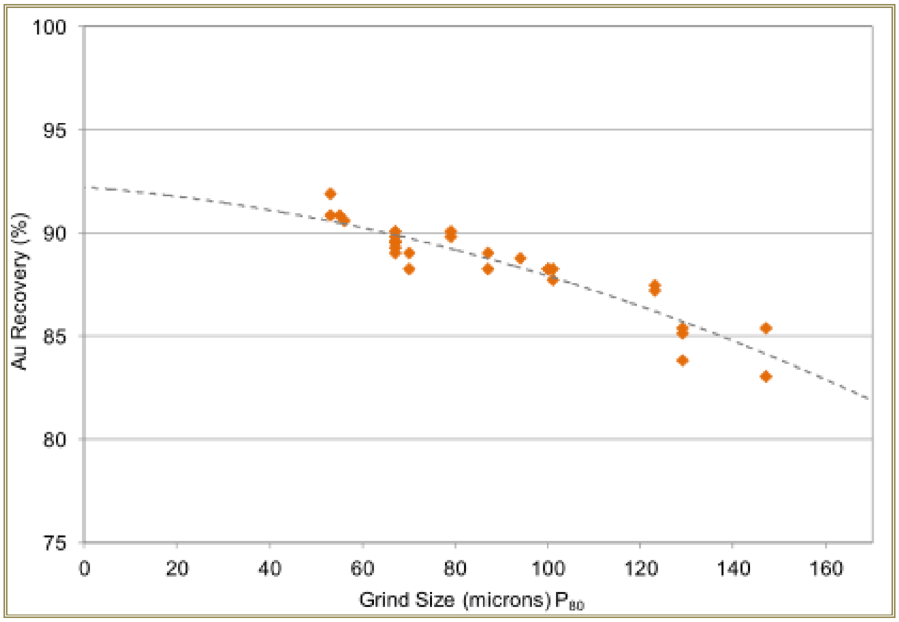
The global composite was also submitted for an extended gravity-recoverable gold test. The amount of gravity-recoverable gold in the sample was assessed at 47.2%.

Cyanidation Testing

The global composite was subjected to developmental cyanidation testing. The program included whole ore versus gravity tailings leaching, the effect of pre-aeration, grind size, and percent solids. The results can be summarized as follows:

- Whole ore leach extraction: 85% to 93%.
- Gravity tailings leach recovery: 81% to 90%.
- Leach kinetics increased with a finer grind size and oxygen sparging.

- Oxygen sparging yielded lower cyanide consumption 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).



Source: SGS, (2008, October 14).

Figure 13-5: Gold Recovery as a Function of Grind Size (Global Composite)

Gravity tailings of the global composite and the variability composites underwent cyanidation testing at P_{80} ≈80 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 improved at finer grind sizes, ranging from 80% to 95%.

The optimized leach conditions defined during the tests are as follows:

- Slurry density: 50% w/w
- pH: 10.5 to 11.0
- Dissolved oxygen: >15 mg/L
- Cyanide concentration: 0.5 g/L NaCN (maintained)
- Retention time: 36 hours.

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 favourable equilibrium loading. The simulation results are presented in Table 13-17.

Table 13-17: Leach and CIP Modelling Results

Parameter	Value
Number of Leach Tanks	6 at 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 and Gold in Feed	1,427
Carbon Advance Rate to Elution and 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 d
Soluble Gold Losses	0.007 mg/L

Cyanide Destruction

The bulk leach product of the global and variability composites was 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. However, since GGM is adhering to the International Cyanide Management Institute (ICMI) code, which has a limit of 50 mg/L CN_{WAD} for end-of-pipe discharge to a tailings facility, the process will be adjusted in such a way that it respects this limit instead of the 1 mg/L target that was used in the testwork.

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 sulphate, and more than 7.0 g of sulphur 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 sulphate and 5.7 to 6.1 g of sulphur dioxide per gram of CN_{WAD}. Finally, variability composites C and H required only 60 to 90 minutes of retention and 5.5 g of sulphur 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 sulphate addition can be increased to reduce total cyanide levels further.

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 sulphate and 7.32 g of sulphur dioxide per gram of CN_{WAD}. A shorter retention time of 60 minutes during test CND12-4 led to an increased concentration of CN_T in the cyanide-destruction discharge solution (Table 13-18). This indicates that the ICMI limit of 50 mg/L could be achieved within a 60-minute retention period, probably with lower reagent consumption; however, no testwork has been done to quantify the retention reduction.

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.

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 flow rates) 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 ID	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 ($m^3/m^2/d$)	Solids Loading ($t/m^2/d$)	Net Hydraulic Loading ($m^3/m^2/d$)	Residence 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 SG –1.000.
[1] Ultimate underflow 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 is practical for pumping.

All the underflow samples displayed Bingham plastic behaviour, and the CSD for all four samples varied between 65% and 69% solids.

Table 13-20: Underflow Rheology Test Results

Sample ID	CSD (wt % solids)	Yield Stress (Pa)		Flow Behaviour & Range (wt % solids) Thixotropy
		Unsheared	Sheared	
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 testwork 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 to 75 ppm solids concentration in the overflow.

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 15 to 25 g of flocculant per tonne of dry solids.

The settling flux tests determined that a feedwell percent solids of between 8% and 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	Unit	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		
Estimated Bed Solids at 0.5 h Retention Time/Estimated Yield Stress	wt%/Pa	57/<25
Estimated Bed Solids at 1 h Retention Time/Estimated Yield Stress	wt%/Pa	58/<25
Estimated Bed Solids at 2 h Retention Time/Estimated Yield Stress	wt%/Pa	60/<25
Estimated Bed Solids at 4 h Retention Time/Estimated Yield Stress	wt%/Pa	61/<25
Estimated Bed Solids at 6 h Retention Time/Estimated Yield Stress	wt%/Pa	73/120
High-Rate Thickeners Sizing Basis:		
Design U/F Solids	wt%	50–60
Design U/F Retention Time	h	2 or less
Design Yield Stress	Pa	25

Note: U/F = underflow.

13.2.4 HPGR Testwork

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 size the equipment adequately. 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). The abrasion tests (ATWAL) and the pilot-plant test were performed at ThyssenKrupp's Resource Technologies Research Center.

Samples from each major lithology (greywacke, iron formation, and porphyry) were prepared and sent to the ThyssenKrupp laboratory for the Labwal tests. A representative composite sample was made from the samples. The pilot-plant composite sample was prepared at the same time to ensure the samples used for the laboratory-scale tests and future pilot-scale tests would have the same characteristics. Table 13-22 shows the sample preparation details.

Table 13-22: HPGR Test Sample Preparation Details

Samples	Material Weight Distribution (kg)						Compositing Ratio (%)
	Received	Stored[1]	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	100.0
Total	**2,471**	**-**	**1,673**	**630**	**-**	**168**	**-**

Notes: [1] Material set aside for the composite.
HPGR = high-pressure grinding roll.

Labwal Tests Results

The results of the Labwal tests are summarized in Table 13-24. 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 test results were used in SGS's comminution circuit simulations to size the HPGR.

Table 13-23: Labwal Test 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 (%)	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

Notes: CL = circulating load; HPGR = high-pressure grinding roll; m_f = specific throughput.

ATWAL Tests Results

The results of the ATWAL tests are summarized in Table 13-24. The greywacke sample was the most abrasive, followed by the porphyry and the iron formation, which showed similar wear rates. According to these results, all the samples were classified as low to medium abrasiveness when dry (1% moisture) or wet (3% moisture).

Table 13-24: ATWAL Tests Results

Ore	Test No.	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

Bond Ball Mill Grindability Tests Results

Bond grindability tests were performed at 106 µm on the four HPGR feed samples and 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 product 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 varied from hard to moderately hard. Results are summarized in Table 13-25.

Table 13-25: Bond Ball Mill Grindability Tests Results

Sample Name	Grind Mesh	F_{80} (µm)	P_{80} (µm)	Grams per Revolution	BWI (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 Product[1]	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 Product[1]	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 Product[1]	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: [1] Represents a different sample preparation approach explained above.

SGS Canada Inc. developed a method that accounts for the effect of the increased quantity of fines in the HPGR product to estimate better the power reduction needed to grind from P_{100} 6 mesh to P_{100} 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 P_{100} 6 mesh to P_{100} 150 mesh.

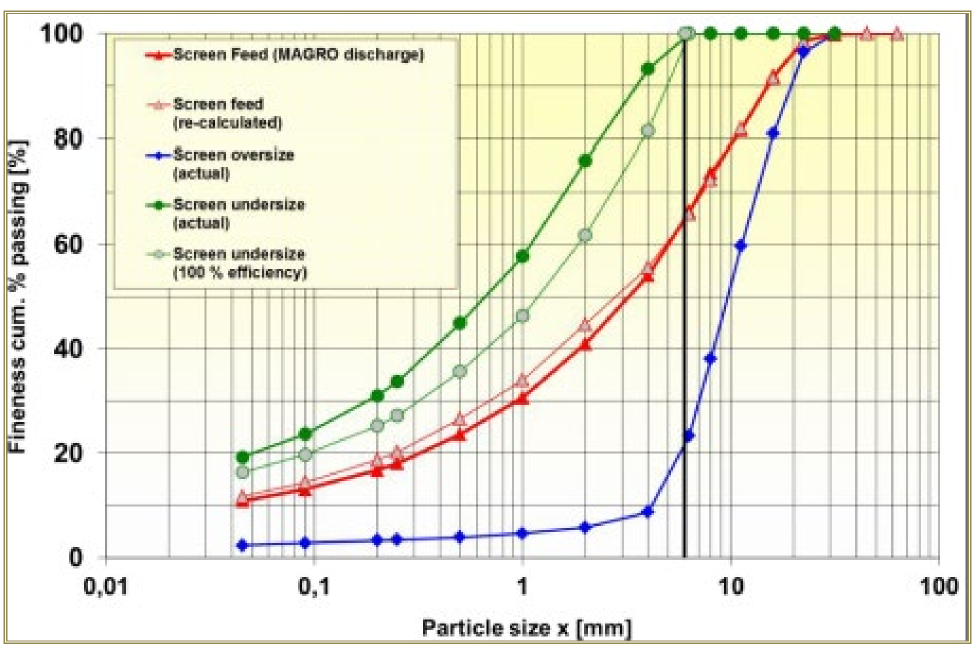
Pilot-Plant Tests Results

Pilot-plant tests were carried out on about 950 kg of gold ore sampled from the Hardrock deposit. The sample material was 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 HPGR with 0.35 m-wide, 0.95 m-diameter rolls. Process data obtained from testwork 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. Therefore, the key parameters are 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 BWI 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.



Source: ThyssenKrupp (2005, May 15).

Figure 13-6: Pilot-Plant Test Third-Cycle Size Distribution

Detailed Engineering Testwork

This section summarizes the testwork program performed during detailed engineering, from 2019 to 2020, to obtain recovery data while varying feed and test conditions and further refine the block model gold-recovery equation (SGS, 2019, March 5).

13.2.5 Characterization and Recovery Testwork

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 testwork. The material used for the composites was 85 individual intervals of crushed rejects.

Characterization and Recovery Tests Results

Head-Grade Determination

Head grade 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.

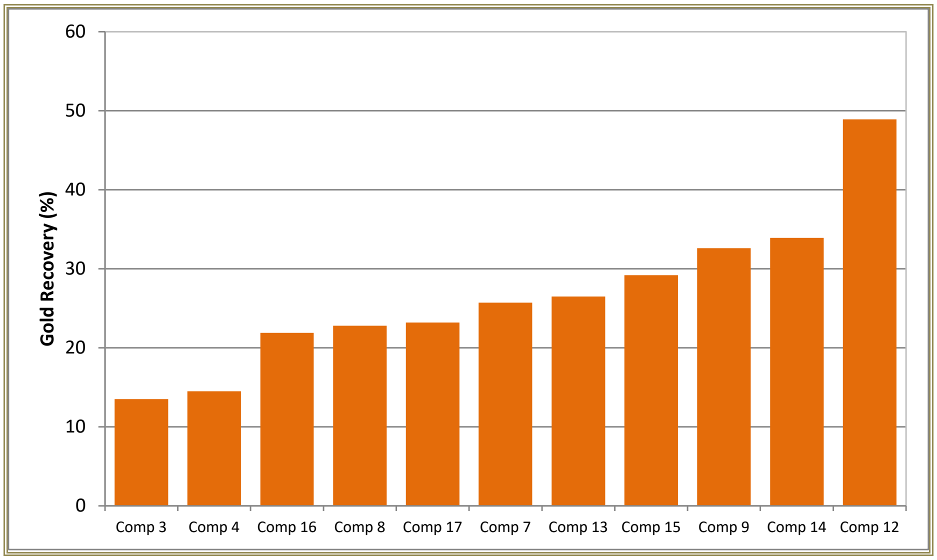
The gold head grades of the composites were determined by metallic sieve, and a weighted average was calculated from the testwork. Calculated head grades were obtained from the cyanidation leach test results. The direct and calculated gold head grades correlate well. The gold head-grade results are presented in Table 13-26, as well as the sulphur-speciation assay results and arsenic results.

Table 13-26: Composite Samples Direct and Calculated Gold Head Grade

Sample Name	Direct (g/t Au)	Calculated from Testwork (g/t Au)	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

Gravity Recovery

Eleven composites were subjected to gravity separation testing using a Knelson concentrator and a Mozley table. Ten composites were ground to P_{80} 90 µm and one sample (Composite 9) was ground to P_{80} 110 µm. The results are shown in Figure 13-7.



Source: SGS, Soutex (2005, March 19).

Figure 13-7: Composite Gravity Recovery Results

Cyanidation Testing

The composites were subjected to confirmatory cyanidation testing. The program included whole-ore versus gravity-tailings leaching, the 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 to 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:

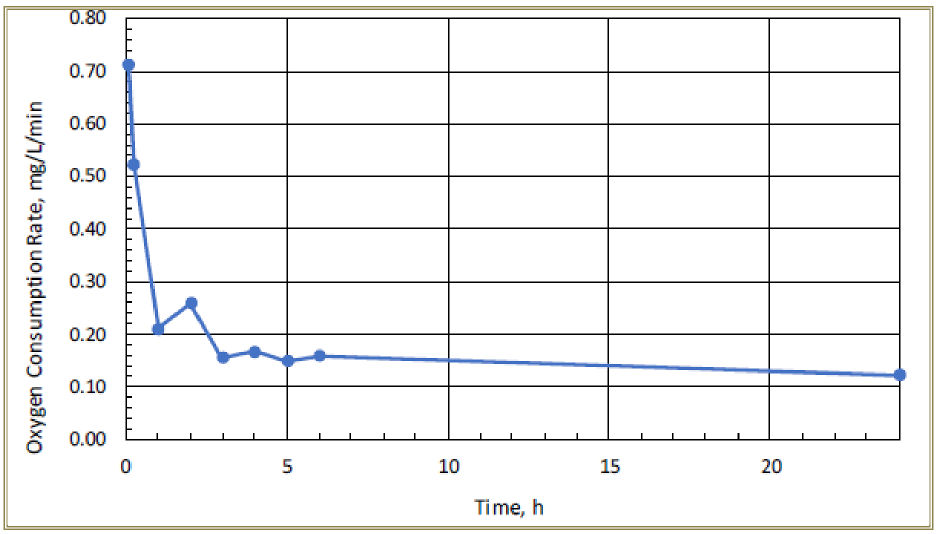
- Cyanide concentration was varied from 0.10 to 0.5 g/L.
- pH was varied from 9.8 to 10.3 and 10.5 to 11.
- The final dissolved oxygen varied from 7 to 33 mg/L.
- The gravity tailings varied from P_{80} 70 to 110 µm.

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.

Oxygen Uptake Test

An oxygen uptake test was performed to provide baseline oxygen consumption information for one of the gravity-tailings samples. Composite 4 was used for the grind-size sample at P_{80} 93 µm, obtained after the gravity separation. Oxygen was sparged into the pulp sample at a rate to reach 15 mg/L, then it was regulated to maintain the dissolved oxygen target concentration in the range of 15 to 20 mg/L. The oxygen consumption rate is presented in Figure 13-8.



Source: SGS (2005, March 19).

Figure 13-8: Oxygen Uptake Test—Composite 4 Gravity Tailings Oxygen Consumption

Solid–Liquid Separation and Rheology

As part of the test program, two ground composite samples were subjected to solid–liquid separation and rheology testing. Both composite samples were tested at particle size K_{80} targets at 90 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 pHs 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 ID	Dosage[1] (g/t dry)	Unit Area[2] ($m^2/t/d$)	Solids Loading ($t/m^2/d$)	Net Rise Rate ($m^3/m^2/d$)	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

Notes: [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 TSS range, expressed in mg/L.
TSS = total suspended solids.

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 to 25 g/t.
- Rheology characterization was performed 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 interparticle 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 behaviour and were generally thixotropic.

13.3 Conclusions and Recommendations

13.3.1 Grinding

Grindability tests have been performed on a sufficient number of samples to assess the comminution characteristics of the Hardrock deposit properly. 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 ROM composition (weighted averages). These results were used as a basis for plant design.

13.3.2 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 an HPGR product.

13.3.3 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.

13.3.4 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.

13.3.5 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.

13.3.6 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. However, pressure oxidation is a costly method for increasing gold extraction.

13.3.7 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 to 11.0
- Dissolved oxygen: >15 mg/L
- Cyanide concentration: 0.35 g/L NaCN (maintained)
- Retention time: 30 hours.

13.3.8 Cyanide Destruction

GGM is adhering to the ICMI code, which has a limit of 50 mg/L CN_{WAD} for end-of-pipe discharge to a tailings facility, the process will be adjusted in such a way that it respects this limit. The SO_2/Air process is effective at reducing cyanide levels to below 1 mg/L in the final tailings as shown in laboratory testwork

A 90-minute retention time is required, with the addition of 45 mg/L of copper sulphate and 7.32 g of sulphur dioxide per gram of CN_{WAD}.

13.3.9 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.

13.4 Process Plant Operation Performances

The process plant for the Greenstone Mine has been in operation since the second quarter of 2024. Since the plant is still in the start-up phase, it is too early to use the operational data as an input for determining future metallurgical performances since current operational data may not be indicative of long-term performance. During ramp-up the metallurgical team is focused on optimizing process plant performance with the objective of achieving optimal efficiencies and operating costs. The cyanide destruction process will be adjusted in such way that the ICMI code is respected while optimising the destruction process cost.

14 MINERAL RESOURCE ESTIMATE

14.1 Hardrock Mineral Resource Estimate

This MRE is an update to the one GMS prepared for the Project in 2019, the results of which were disclosed in the 2021 Report issued January 26, 2021.

Completion of the current MRE update involved assessing an updated drill-hole database, which included data for an additional 67 RCGC and 56 DDHs completed since the last MRE, and an updated block model for mine planning purposes.

In 2022, GGM contracted GMS to update the gold domain wireframes, estimation processes, and the 2019 block models using the 2021 DDH and RC drilling. The 2022 MRE update was produced under the supervision of GMS's then-Vice President Geology and Resources Mr. James Purchase, P.Geo., an independent QP as defined in NI 43-101. The block models Mr. Purchase produced were for GGM internal use only.

In 2023, and more recently in 2024, GGM engaged GMS's External Auditor—Geology and Resources Réjean Sirois, P.Eng., an independent QP as defined in NI 43-101, to validate and endorse Mr. Purchase's work and sign-off as the 2024 MRE's QP. Mr. Sirois did that work; he was the Mineral Resource QP for the 2019 MRE. Mr. Sirois tested the database, interpretations, estimation parameters, and resulting block models. He agreed with the models and parameters Mr. Purchase used, and no fatal flaws were encountered. The only potentially significant change between 2019 and 2022 was an increase from 0.1 g/t Au to 0.15 g/t Au for constraining the lower grades of the external grade-shells. An assessment of the difference has shown that the change was not material to the global MRE. As a result of the assessment, some external grade-shell Indicated resources were retagged as Inferred resources for the underground MRE. The resulting adjustments led to the current 2024 block models, and when optimized, to the 2024 MRE.

This MRE update was produced by Mr. Sirois, who visited the Project on multiple occasions in 2018 and 2019 to review drilling and sampling protocols. He also visited the Greenstone Mine from July 22 to 25, 2024, to collect project information since 2019 for this update. Drill core was reviewed for a few new holes drilled in 2021. No drill collars surrounding the current pit were visited, since most of these collars are already consumed by mining, and diamond drilling was not ongoing at the time. The effective date of the updated MRE is June 30, 2024.

For this work GMS supplied an updated drilling database export. GMS excluded from the MRE unassayed geotechnical holes, blastholes, abandoned holes with redrills, and channels, consistent with the 2019 approach.

In 2024, GMS asked for a copy of the database used in 2022 and was able to compare both data sets. The database is identical.

This MRE was prepared from 22 subvertical mineralization domains (the "principal" domains) and remaining mineralization was captured by Leapfrog RBF grade shells. (the "external grade shells"). Leapfrog GEO was used for wireframing and geological interpretation, and Geovia GEMS was used for block modelling and estimation.

ID^3 was used to interpolate gold grades (g/t) into a block model using the modelled mineralization domains. Measured, Indicated, and Inferred Mineral Resources are reported in the summary tables in

Section 14.3.12. 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 the Canadian Institute of Mining, Metallurgy and Petroleum *CIM Definition Standards for Mineral Resources & Reserves* (adopted May 19, 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 Definition Standards for Mineral Resources & Reserves*. GMS believes that the MRE reported herein is a reasonable representation of the global Mineral Resources found at the Greenstone Mine at the current level and spacing of sampling.

The MRE includes Inferred Mineral Resources that are 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.

Mineral Resource estimation methodologies, results, and validations are presented in this section of the Technical Report.

14.1.1 Drill Hole Database

To complete an updated MRE for the Project, GGM provided to GMS a database comprising a series of comma-delimited files containing information for 67 RCGC drill holes and 56 DDHs completed since the 2021 MRE. The database included drill-hole collar information (UTM Zone 16; NAD 83), surveys, assays, and lithological, alteration, structural, and geotechnical data. The data for the additional drill holes were subsequently imported and merged with the database used in the 2019 MRE using Geovia GEMS (GEMS) (Version 6.8.2) for statistical analysis, block modelling, and MRE.

The current MRE is derived exclusively from the database described in Section 12. GMS has reviewed the database and is satisfied with the integrity of the drilling database. GMS believes it is suitable for the purposes of mineral resource estimation. A summary of the updated drill-hole database is presented in Table 14-1.

Table 14-1: Summary of the Drilling Database for the Hardrock Gold Deposit

Drilling Type	No. of Drill Holes	Total Length (m)	Assayed
Borehole (vertical)	90	965	684
Channel	31	1,513	1,498
Diamond	1,846	738,232	462,540
Reverse Circulation-Grade Control	549	30,183	27,389
All	**2,516**	**770,893**	**492,110**

The drill-hole database for the 2024 MRE contains 549 RCGC and 1,846 DDH drill holes. All 2,395 drill holes were used in the 2024 MRE update, representing those completed and validated at the data close-out date of March 23, 2022. Boreholes and channel samples were not considered in the current MRE. Historical underground drilling was not considered in the 2024 MRE (consistent with the 2019 MRE).

The drill holes cover the 5.7 km strike-length of the deposit at an irregular 50 m (X) by 25 m (Y) drill spacing within the extents of the proposed open pit, with some infill drilling 25 m (X) apart 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, and 2021 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.

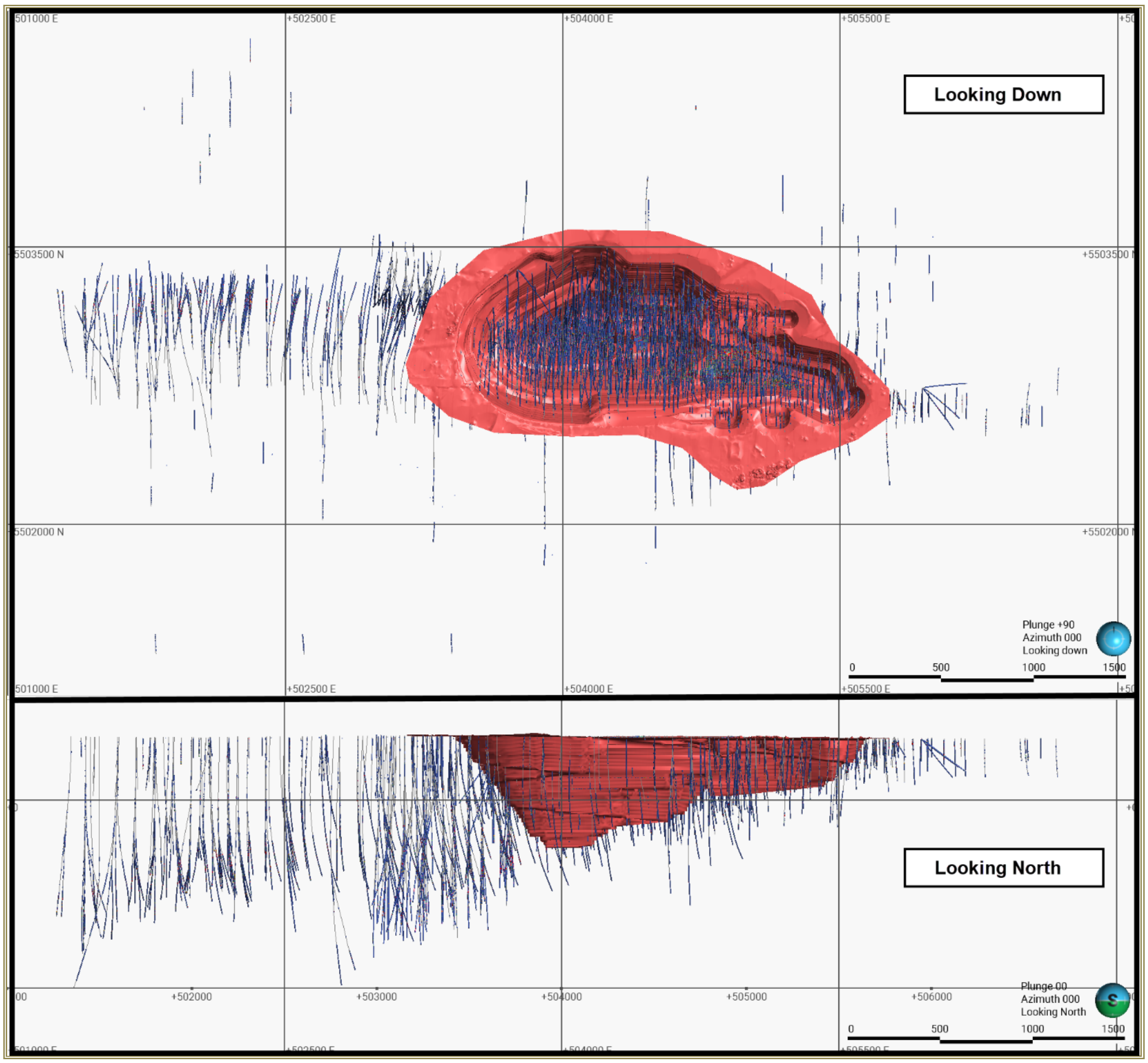


Figure 14-1: View of All Drilling Used for the 2024 MRE, Showing the 2024 Pit Design (Red) for Reference

A surface channel-sample database with a total of 1,219 assays from 26 channel samples collected in 2014 was already integrated into the 2016 GEMS project. Channel samples were not used in the 2019 and 2022 MRE, as they are superseded by RCGC drilling.

14.1.2 Geological Modelling Approach

A litho-structural model was developed that divided mineralization into structurally controlled, subvertical mineralization wireframes, and remaining mineralization was captured using domains based on a sectional interpretation of lithology (iron formation, greywacke, porphyry, gabbro, and ultramafic). The structural wireframes demonstrated continuity between drilling sections though required refining as they contained significant dilution. 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 excessive grade smearing.

The previous mineralization wireframes were interpreted using 17 principal domains representing the various styles of mineralization. In addition, grade shells were used to model remaining mineralization outside of the principal domains, representing roughly 20% of the ounces in the MRE. A similar approach was used for the 2024 block model update. GMS applied the following methodologies to update the mineralization model:

- Update the A Zone, SP Zone, Lower Zone, and Tenacity Zone wireframes with the new diamond drilling.
- Split the A Zone into two discrete wireframes representing the two subparallel zones intercepted in drilling.
- Model additional zones that were previously in the external grade shells if they demonstrated continuity over three drill sections.
- Review external grade shell thresholds (previously 0.1 and 0.6 g/t).

New zones modelled as part of the 2024 block model update are shown in Figure 14-2. GMS retained the internal grade shells for six of the principal domains as per the 2019 modelling approach.

After extensive review on section, the lower cut-off grade for the external grade shell was revised upwards from 0.1 to 0.15 g/t Au. This is due to the presence of excessive extrapolation of grade at a 0.1 g/t Au cut-off, and a tighter interpretation around mineralized intercepts at 0.15 g/t Au (Figure 14-3). In 2024, GMS validated and confirmed that the increase from 0.1 to 0.15 g/t Au has no material effect on the global open pit MRE, with only 1% fewer ounces of gold than predicted by the 2019 block models.

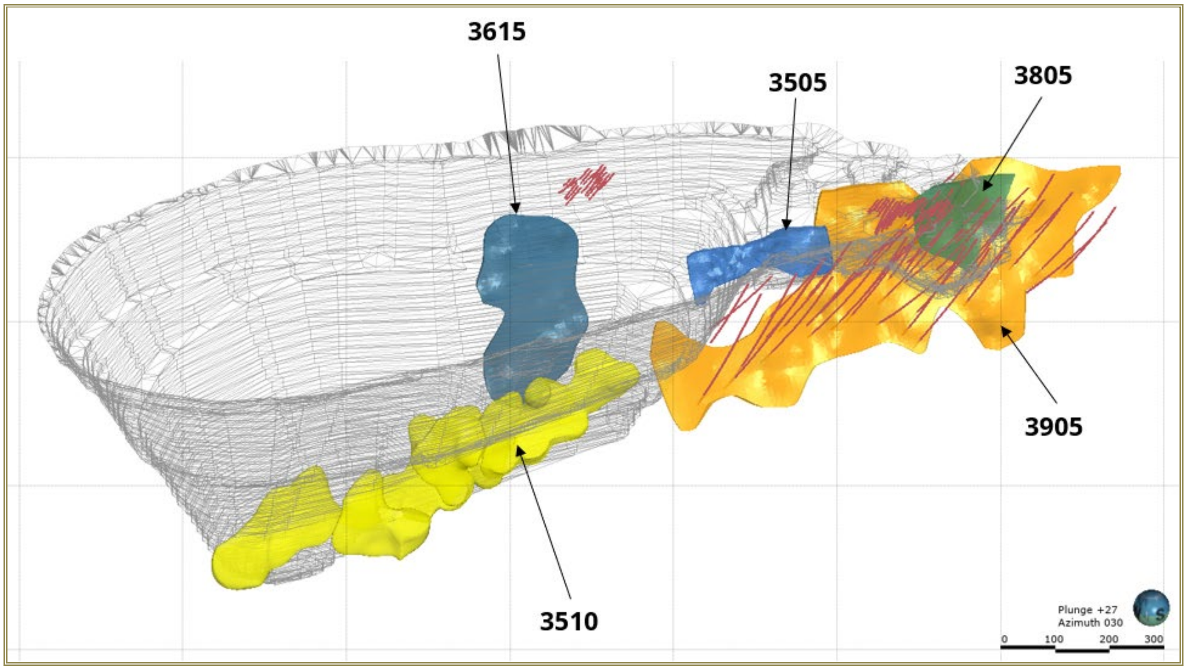


Figure 14-2: Isometric View (Looking NNE) of New or Remodelled Zones Added Since the 2019 MRE

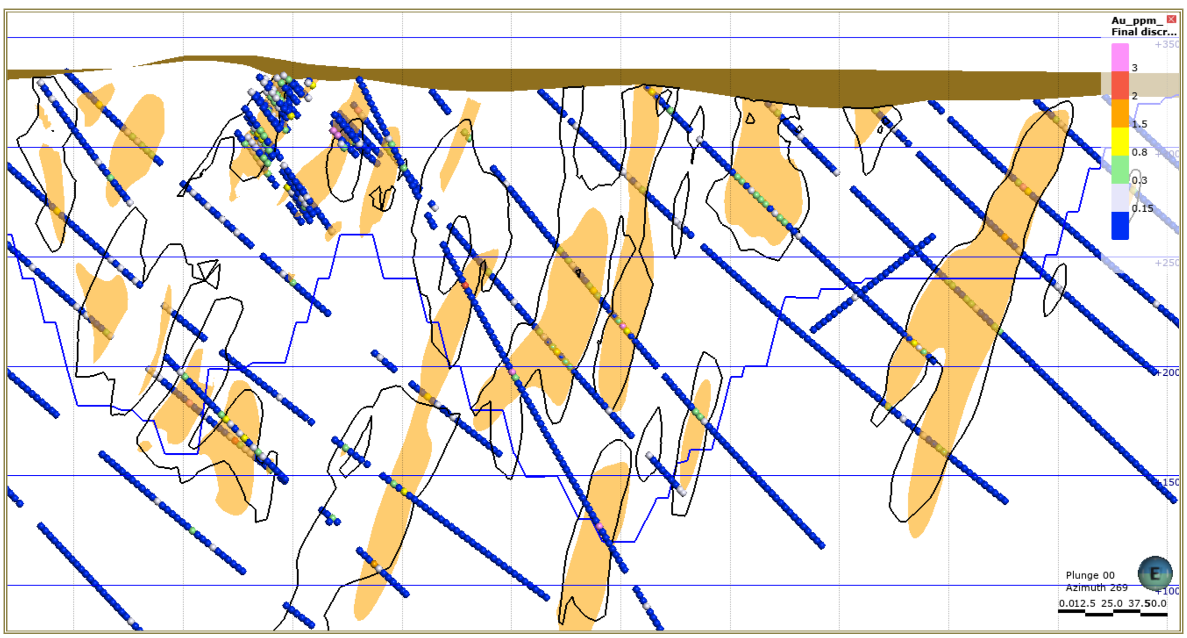


Figure 14-3: Comparison Between 0.1 g/t Au (Black Line) and 0.15 g/t Au (Orange Fill) Grade Shells

In all, 22 mineralized domains were modelled in Leapfrog Geo software using as reference the mineralized domains defined in the 2019 MRE. The new interpretation was completed on cross-section using an average grade of 0.3 g/t Au as the lower limit and three-dimensional (3-D) 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 Au 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 adjusted the SP Zone at depth, where data-spacing and lack of grade continuity did not permit a continuous geological interpretation.

Figure 14-4 and Figure 14-5 show 3-D views of the 22 principal mineralized domains that GMS modelled for the current MRE. The nomenclature is as follows:

- F Zone (3105)
- North 1 Zone (3205)
- 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 Zones (3500, 3505, and 3510)
- North 2 Zones (3600 and 3615)
- 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 and 3805)
- A Zones (3900 and 3905)
- Tenacity Zone (4000)
- SP2 Zone (4100).

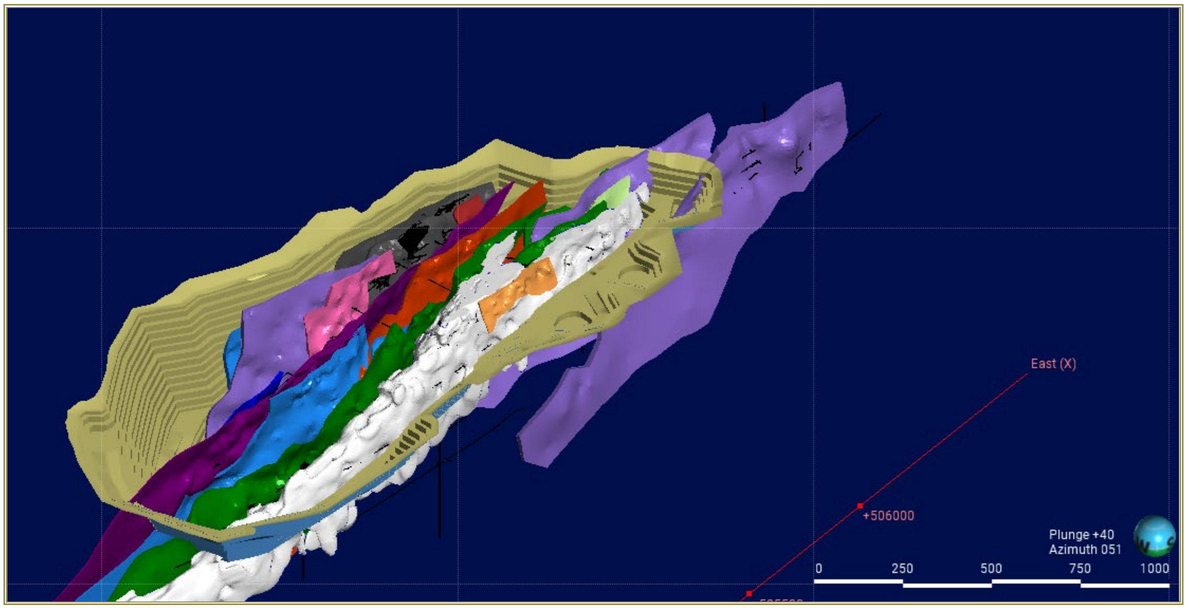


Figure 14-4: Isometric View of the 22 Principal Domains, Looking NE, within the 2024 Pit Design

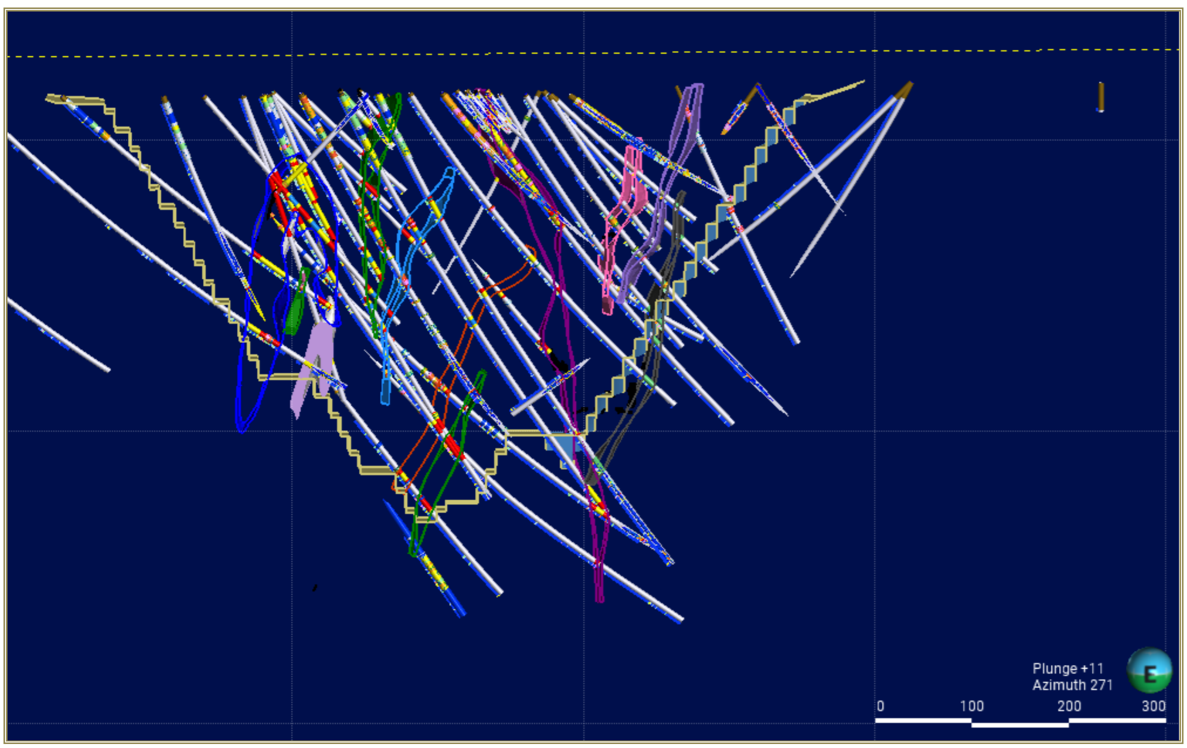


Figure 14-5: Section 504,600 m E, Looking West: Principal Domains and Drilling, 2024 Pit Design shown for Scale

14.2 Wireframing—Lithology

GGM updated the lithology model using the new drilling, which supersedes the previous model created in 2019. No major reinterpretations were made, and the lithology model was built in Leapfrog Geo. An example of a cross-section is presented in Figure 14-6.

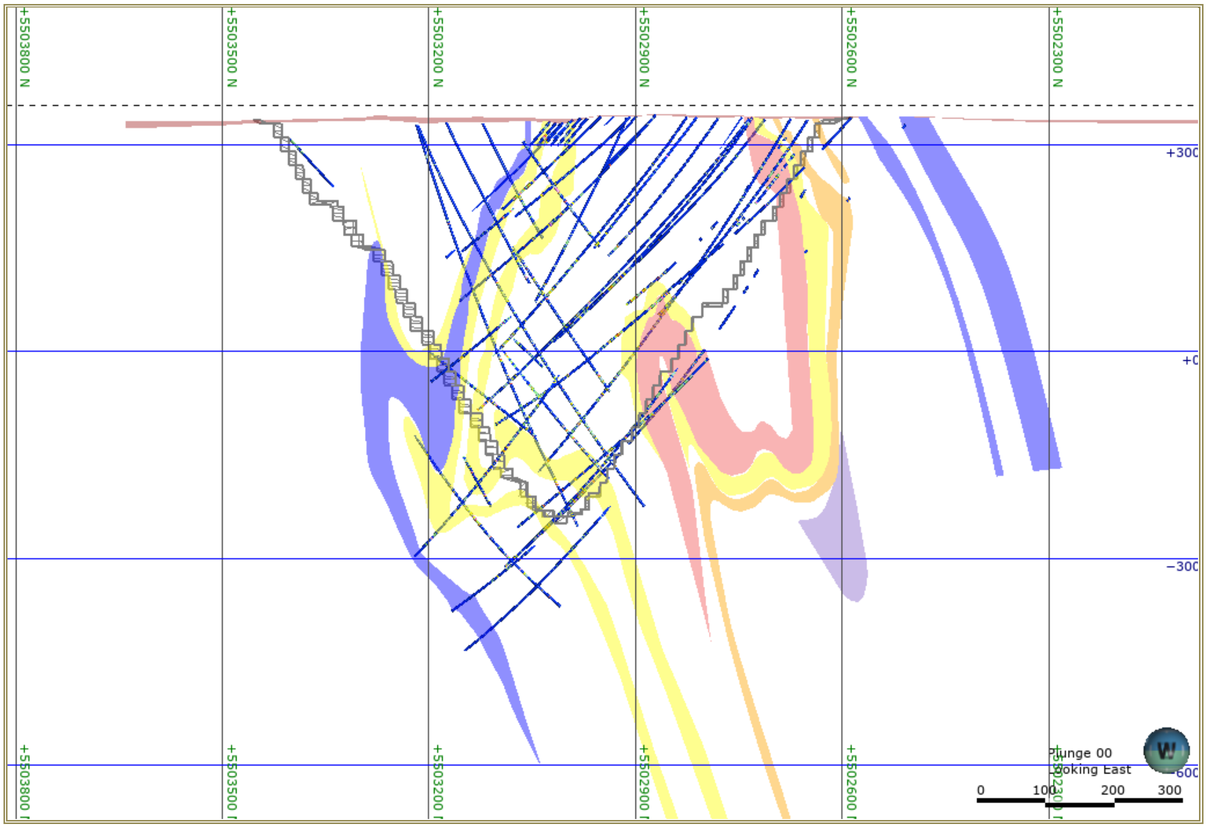


Figure 14-6: Example Section of the Updated Lithology Model

An overview of the various rock codes and corresponding zones are presented in Table 14-2.

Table 14-2: Summary of the Mineralized Zones and Lithology Codes for the Greenstone Mine

Mineralized Zones	Rock Code		Lithology	Rock Code
F Zone	3105[1]		Porphyry	8100
North 1 Zone	3205[1] and 3210		Conglomerate 1	9100
Central Zone	3300[1]		Conglomerate South (1, 2, 3, and 4)	10100
	3305[1]			10200
F2 Zone	3405[1]			10300
SP Zone	3500[1]			10400
	3505		Iron Formation North 1	11100
	3510		Iron Formation North 2	11200

Mineralized Zones	Rock Code		Lithology	Rock Code
North 2 Zone	3600		Iron Formation North 3	11300
	3605		Lower Iron Formation	12000
	3610		Middle Iron Formation	13100
	3615		Upper Iron Formation	14100
	3620		Ultramafic	15000
North 3 Zone	3710		Gabbro North	16000
	3720		Gabbro South 1	17100
Lower Zone	3800		Gabbro South 2	17200
	3805		Gabbro South 3	17300
A Zone	3900		Gabbro South 4	17400
	3905			
Tenacity Zone	4000			
SP 2 Zone	4100			
External Grade Shells	500			
	501			
	506			

Notes: [1] These domains were further subdivided internally using grade shells (<0.15 g/t, 0.15–0.6 g/t, and >0.6 g/t). A suffix was added to the rock code depending on the grade shell status (i.e., 3105 high-grade (>0.6 g/t) was named 31056, and 3405 low-grade (<0.15 g/t) was named 34050.

14.2.1 Internal Subdomain Grade Shells

Six of these 22 principal domains were further split into low-grade (<0.15 g/t Au), medium-grade (0.15–0.6 g/t Au), and higher-grade (>0.6 g/t Au) subdomains 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 Au (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 0.15 g/t Au and 0.6 g/t Au were chosen based on statistical analysis. The 0.15 g/t Au threshold appears to be a natural limit between mineralized and non-mineralized material, and 0.6 g/t Au was chosen to ensure that continuity between medium- and higher-grade populations was preserved during the modelling in sections (Figure 14-7).

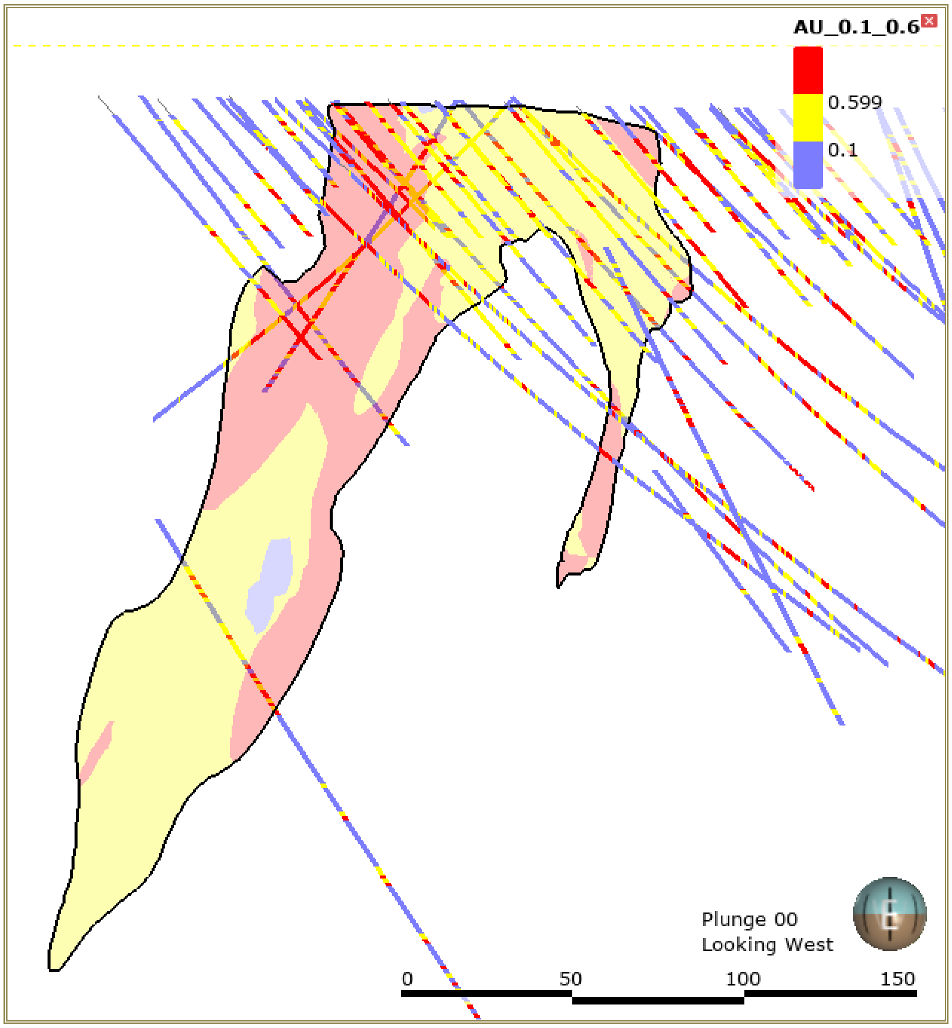


Figure 14-7: Section 504,815 m E (Looking West)—Example of Internal Grade Shell Sub-Domains for the SP Zone. Blue: <0.15 g/t Au, Yellow: 0.15–0.6 g/t Au, Red: >0.6 g/t Au

14.2.2 External Grade Shells

All mineralization outside of the 22 principal domain wireframes were captured by Leapfrog RBF grade shells. Three grade-controlled domains were chosen: low (<0.15 g/t Au), medium (0.15—0.6 g/t Au), and high (>0.6 g/t Au) shells. The construction of the grade shells was guided by a trend based on lithological wireframes and the 22 principal domains.

Gold values were temporarily capped at 10 g/t Au during construction of the grade shells to reduce the "leapfrog bubble effect" and to prevent the overestimation of volume in data-sparse areas.

14.2.3 Topographic and Bedrock Surfaces

A topographic surface was generated from drone-collected LiDAR data. That surface was 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 historical 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.

14.3 Wireframing—Void Model

The underground void model was updated to account for new voids intercepted during the winter 2021/2022 drilling program. Densities were coded according to the type of the void fill (sand = 2.02 g/cm^3, waste = 2.08 g/cm^3), as shown in Figure 14-8.

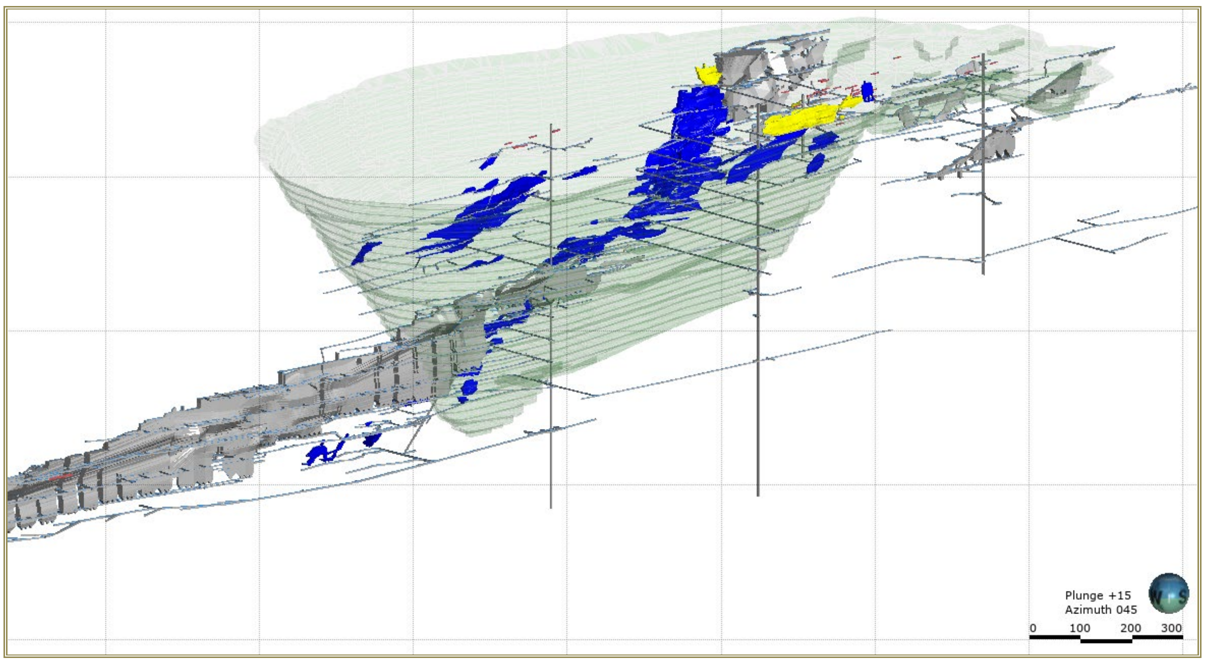


Figure 14-8: Underground Void Model Coloured by Void Fill (Blue = Waste Fill, Yellow = Sand Fill, Grey = Open)

14.3.1 Assay Capping

High-Grade Capping

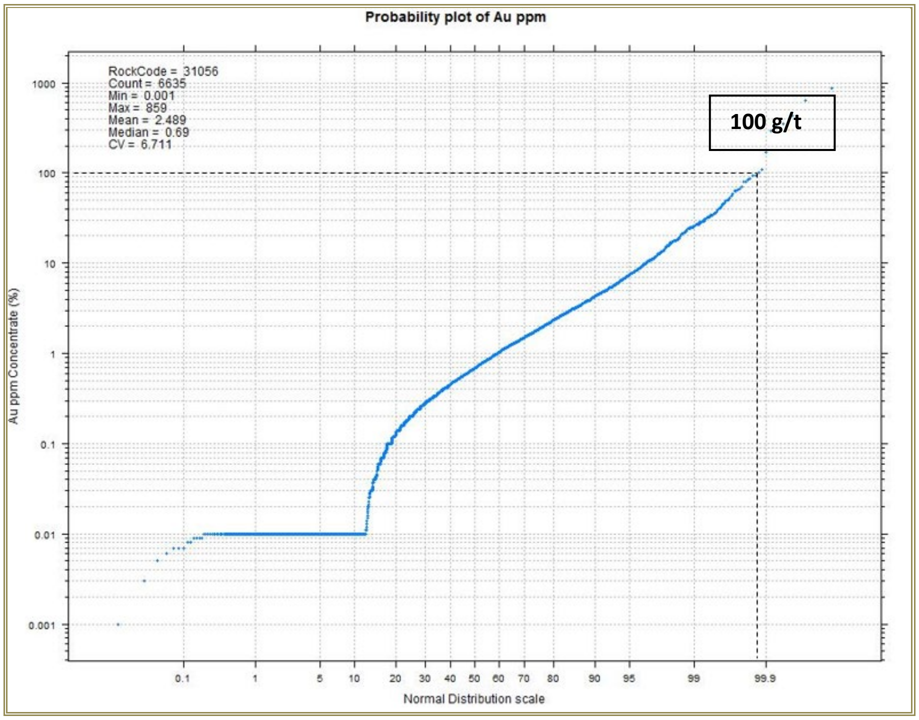
Basic univariate statistics were performed on raw assay data sets grouped by rock code. Assay capping was undertaken before compositing for each mineralized domain and subdomain, and capping levels were chosen primarily using probability plots.

In all, 174 assay results were capped using the determined capping limits, which represents a metal loss-factor of 9.3%. The differences are mainly due to some high-grade samples being moved out of the external grade-shell domain into a new principal domain and capped more conservatively.

Table 14-3 presents a summary of the statistical analysis for each domain for the raw assays. Figure 14-9 shows an example of probability plots for the high-grade subdomains of the F Zone.

Table 14-3: Summary Statistics of Raw Assays by Domain and Sub-Domain

Rock Code	No. of Assays	Total Assay Length (m)	Max. Au Uncapped (g/t)	Capping Levels (g/t) Au	No. Capped	Coefficient of Variation	Metal Loss (%)
500	222,471	313,015.21	19.90	4.00	1	2.81	0.1
501	46,578	63,688.3	19.90	14.00	3	2.17	0.0
506	7,509	9,609.88	3,880.00	140.00	10	4.38	9.3
3210	547	811.5	53.40	20.00	11	2.48	13.8
3505	538	891.4	4.98	4.98	-	1.47	0.0
3510	749	1,051.8	101.00	6.00	1	1.61	17.6
3600	2,259	2,914.1	2,000.00	50.00	3	2.84	26.4
3605	1,602	2,250.65	41.00	20.00	10	3.07	4.8
3610	290	398.5	234.00	20.00	3	2.83	51.6
3615	306	414.7	2,580.00	30.00	2	3.69	86.1
3620	56	82.2	3.76	3.76	-	1.05	0.0
3710	1,213	1,587.3	511.00	20.00	5	2.32	32.5
3720	345	465.18	36.10	15.00	4	1.91	8.5
3800	1,021	14,24.51	114.69	25.00	4	2.70	6.2
3805	100	114.11	10.10	10.00	1	2.62	0.1
3900	1,846	2,151.48	559.00	50.00	4	4.13	14.9
3905	978	1,097.4	237.00	25.00	2	2.58	6.8
4000	1,948	2,480.55	1,560.00	15.00	14	2.08	29.7
4100	483	612.5	156.00	30.00	5	2.66	23.1
31050	245	337.38	30.30	2.00	5	2.95	57.4
31051	2,221	3,292.39	18.70	3.00	11	1.68	5.0
31056	6,187	8,845.74	859.00	100.00	10	3.27	10.8
32050	804	1,168.35	3.20	2.00	1	3.28	2.6
32051	5,306	7,566.24	21.00	6.00	3	1.94	1.5
32056	6,547	8,557.76	402.00	110.00	5	3.25	3.9
33000	99	143.1	1.14	1.14	-	2.65	0.0
33001	1,571	2,171.26	3.92	3.00	2	1.44	0.2
33006	1,949	2,503.35	436.00	60.00	13	3.60	19.6
33050	4	6	0.01	0.01	-	0.00	0.0
33051	186	247.9	3.42	3.42	-	3.31	0.0
33056	542	703.88	77.20	15.00	10	2.01	11.6
34050	266	400.2	1.43	1.43	-	2.84	0.0
34051	1,707	2,417.65	6.27	3.00	5	1.65	0.8
34056	2,466	3,219.44	251.00	90.00	5	3.61	5.3
35000	236	355.72	44.10	3.00	3	3.20	53.3
35001	7,821	11,624.69	51.12	5.00	10	1.30	1.3
35006	9,677	13,581.35	2,366.06	125.00	8	3.89	2.5
Total Metal Removed (%)							**9.3**



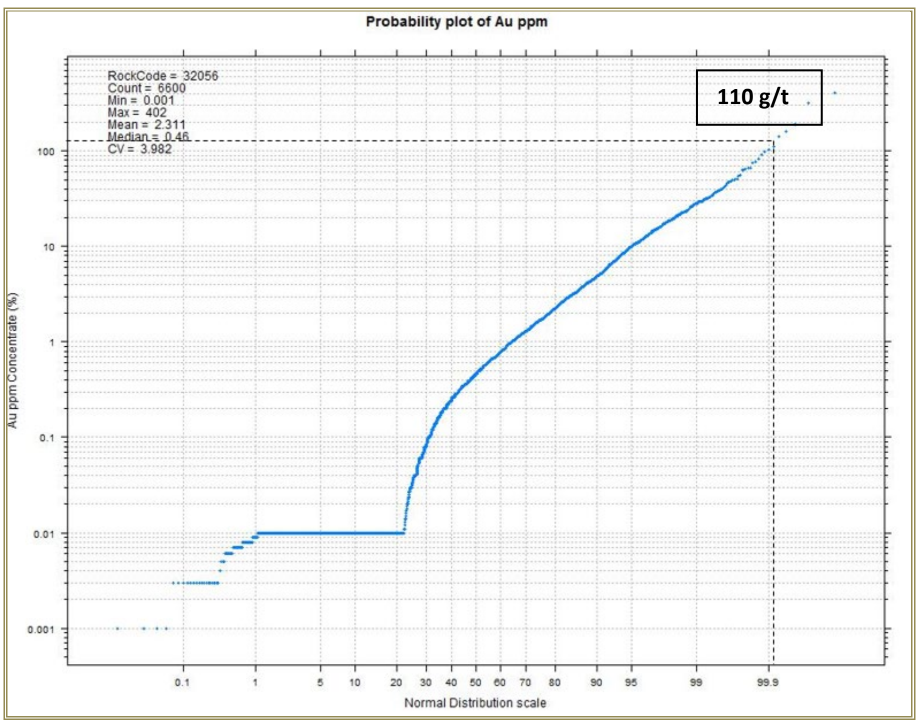


Figure 14-9: Probability Plot Au (g/t)—Top, High-Grade Sub-Domain (>0.6 g/t Au) of the F Zone; Bottom, North 1 Zone

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 RCGC drilling in 2018, 2019, and 2021.

Composites of 2.0 m (downhole) 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).

The total number of composites used in the 2022 and 2024 MRE is 359,517. A grade of 0.00 (g/t Au) was assigned to missing sample intervals during compositing; however, unsampled intervals within voids were removed from the estimation. Table 14-4 summarizes the basic statistics of the gold composites used for the 2024 MRE.

Table 14-4: Summary Statistics for the 2.0 m Composites

Zone	Principal Domain	Subdomain	No. of Composites	Max. (g/t Au)	Mean (g/t Au)	Standard Deviation	Coefficient of Variation
F Zone	3105	31050	210	1.15	0.07	0.16	2.29
		31051	1,923	2.93	0.27	0.34	1.26
		31056	4,656	100.00	2.00	4.58	2.29
North 1 Zone	3205	32050	663	1.32	0.03	0.08	2.17
		32051	4,177	7.93	0.24	0.37	1.58
		32056	4,586	77.26	1.98	4.39	2.22
	3210	3210	423	20.00	1.51	3.02	2.00
Central Zone	3300	33000	100	0.78	0.07	0.13	2.04
		33001	1,183	2.24	0.28	0.30	1.08
		33006	1,394	53.83	1.76	4.52	2.57
New Zone	3305	33050	3	0.01	0.01	0.00	0.00
		33051	137	1.73	0.13	0.28	2.17
		33056	381	15.00	1.71	2.45	1.43
F2 Zone	3405	34050	194	1.01	0.04	0.10	2.47
		34051	1,336	2.24	0.24	0.28	1.16
		34056	1,812	59.00	1.83	4.41	2.41
SP Zone	3500	35000	233	2.10	0.13	0.30	2.23
		35001	6,369	5.00	0.32	0.30	0.95
		35006	7,393	125.00	1.55	4.14	2.66
SP New Zone	3505	3505	476	4.86	0.42	0.56	1.33
SP New Zone	3510	3510	563	3.47	0.41	0.47	1.14
North 2 Zones	3600	3600	1,523	40.70	1.95	3.92	2.01
	3605	3605	1,153	20.00	0.99	2.05	2.06
New Zone	3610	3610	207	15.36	1.05	2.39	2.29
New Zone	3615	3615	224	30.00	1.07	2.93	2.75
New Zone	3620	3620	43	2.82	0.68	0.58	0.86

Zone	Principal Domain	Subdomain	No. of Composites	Max. (g/t Au)	Mean (g/t Au)	Standard Deviation	Coefficient of Variation
North 3 Zones	3710	3710	834	15.30	1.04	1.72	1.66
	3720	3720	254	13.70	1.45	2.15	1.49
Lower Zone	3800	3800	758	25.00	1.01	2.01	1.99
Lower New Zone	3805	3805	67	6.24	0.67	1.14	1.71
A Zone	3900	3900	1,159	22.61	1.06	2.00	1.89
A New Zone	3905	3905	602	25.00	0.98	1.92	1.96
Tenacity Zone	4000	4000	1,286	11.56	1.06	1.49	1.41
SP 2 Zone	4100	4100	332	23.70	1.63	3.27	2.01
External Grade Shells	500	500	272,242	4.00	0.02	0.06	2.73
	501	501	35,196	6.16	0.26	0.37	1.40
	506	506	5,425	105.11	1.81	4.68	2.59
Total			**359,517**				

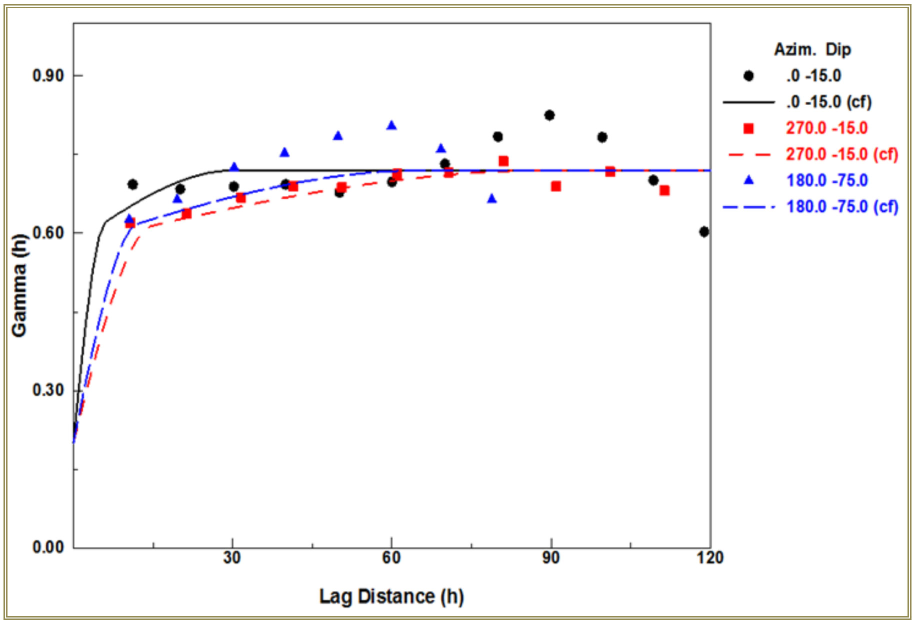
14.3.2 Variography

Variograms were not updated for the 2024 MRE since ID^3 estimators were used for the grade interpolations. The following information was extracted from the 2021 Report as support for the ranges used in the gold grade interpolations.

3-D directional variography was used in the 2019 estimates 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 modelling 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 downhole variogram.
- Pairwise-relative variograms were chosen for modelling the major, semi-major, and minor axes, as they showed the clearest structure.

Figure 14-10 illustrates 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.



Notes: * S1 Range–Major (Red) = 15 m, Semi (Blue) = 12 m, Minor (Black) = 6 m, S2 Range–Major = 90 m, Semi = 65 m, Minor = 30 m

Figure 14-10: Pairwise-Relative Model for the SP Zone (Domain 3500)

The selected variogram model parameters are tabulated in Table 14-5.

Table 14-5: Variogram Model Parameters for Domain

Zone	Domain	Axis	Nugget	Sill 1	Range 1 (m)	Sill 2	Range 2 (m)
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

14.3.3 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., the major range was longer than the 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 subdomains, and the North 2 Zone was divided into two search-ellipse subdomains, 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 subdomains. 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 visualization of search ellipses in three dimensions.

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 subdomains (SP Zone, North 1 Zone, and external grade shells).

14.3.4 Treatment of High Grades

To control the influence of isolated high-grade composites during grade estimation, GMS used high-grade restraining (also known 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 identifying 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 subhorizontal 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 half the first pass dimensions.

In the 22 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 14-6 summarizes the parameters of the final ellipsoids and threshold dimensions used for grade interpolations.

Table 14-6: Final Search Ellipsoid Parameters and Threshold Dimensions

Domain	Rotation GEMS			Pass 1			Pass 1 High-Grade (HG) Threshold				Pass 2			Pass 2 HG Threshold				Pass 3			Pass 3 HG Threshold			
	Z	X	Z	X (m)	Y (m)	Z (m)	X (m)	Y (m)	Z (m)	HG Threshold (g/t Au)	X (m)	Y (m)	Z (m)	X (m)	Y (m)	Z (m)	HG Threshold (g/t Au)	X (m)	Y (m)	Z (m)	X (m)	Y (m)	Z (m)	HG Threshold (g/t Au)
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
3505	−5	65	0	50	50	15	None				75	75	25	None				120	120	40	None			
3510	−10	75	15	40	30	15	None				70	50	25	None				100	75	35	None			
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			
3615	−5	75	0	50	40	15	None				75	60	25	None				100	80	35	None			
3620	−15	65	0	50	40	12	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			
3905	−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
	Sediment-Related			50	50	15	20	20	5	21	75	75	25	20	20	5	21	100	100	40	20	20	5	21

14.3.5 Bulk-Density Data

For the 22 principal domains, GMS assigned the bulk density by estimation domain. For the external grade shells, bulk density was assigned based on the updated 2022 lithology model. Median values were used to reduce the influence of outliers.

A density of 2.00 g/cm^3 was assigned to the overburden, and 2.05 g/cm^3 to the tailings. For the voids, densities of 0.00 g/cm^3 and up to 2.08 g/cm^3 (backfilled) were used for drifts and stopes, which remains similar to the 2016 MRE.

GMS believes that the bulk-density database is of sufficient quality for mineral resource estimation at the Hardrock deposit.

Table 14-7 presents the bulk density values assigned to the block model.

Table 14-7: Bulk Density Assigned to Block Model by Domain

Zone	Principal Domain	Block Code	Dominant Lithology	No. Samples	Bulk Density (g/cm^3)		
					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
F 2 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
	3505, 3510	3505, 3510	Mixed (IF, Porphyry, and Greywacke)	-	2.75	-	-
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, 3805	3800, 3805	Mixed (IF, Porphyry, and Greywacke)	8	2.77	2.68	3.27
A Zone	3900, 3905	3900, 3905	Porphyry and Iron Formation	10	2.74	2.59	2.94
Tenacity Zone	4000	4000	Greywacke	12	2.73	2.57	3.57
SP 2 Zone	4100	4100	Greywacke	7	2.76	2.58	2.78
Porphyry	-	8100	Porphyry	539	2.73	2.31	3.62

Zone	Principal Domain	Block Code	Dominant Lithology	No. Samples	Bulk Density (g/cm³)		
					Median	Min.	Max.
Conglomerate	-	9100	Conglomerate	144	2.75	2.47	3.41
Conglomerate	-	10100–10400	Conglomerate	45	2.74	2.53	3.77
IF North 1	-	11100	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	Iron Formation	10	3.28	2.72	3.94
Upper IF	-	14100	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	-	500	Greywacke	1,311	2.74	2.15	3.63

14.3.6 Block Model

The block model dimensions remained the same as the 2019 block model and are described in Table 14-8. A regular block size of 10 m (X) by 5 m (Y) by 10 m (Z) was used, and volume cross-checks were undertaken to ensure block volumes are with 1% of wireframe volumes.

Table 14-8: Block Model Properties

Description		No. 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

Note: The block model origin is the upper southwest corner of the block model.

The 22 mineralized domains (and their internal grade-shell subdomains) 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 various tailings volumes from the MRE.

14.3.7 Grade Estimation

No major changes were made to the grade estimation approach compared to the 2019 MRE. ID^3 was used as the interpolator, and the three-pass search-ellipse strategy was retained as described in the 2019 technical report. A summary of search-ellipse parameters is presented in Table 14-6, and the estimation parameters for each pass are summarized below:

- Pass 1
  - Minimum of 7 and maximum of 15 composites in the search ellipse for interpolation
  - Maximum of three composites from any one drill hole
  - Minimum of three drill holes required for interpolation of the given block.
- Pass 2
  - Minimum of 4 and maximum of 15 composites in the search ellipse for interpolation
  - Maximum of three composites from any one drill hole
  - Minimum of two drill holes required for interpolation of the given block.
- Pass 3
  - Minimum of 3 and maximum of 15 composites in the search ellipse for interpolation
  - Maximum of three composites from any one drill hole
  - Minimum of one drill hole required for interpolation of the given block.

The estimation of block grades is illustrated on a plan view and a cross-section (Figure 14-11 and Figure 14-12, respectively).

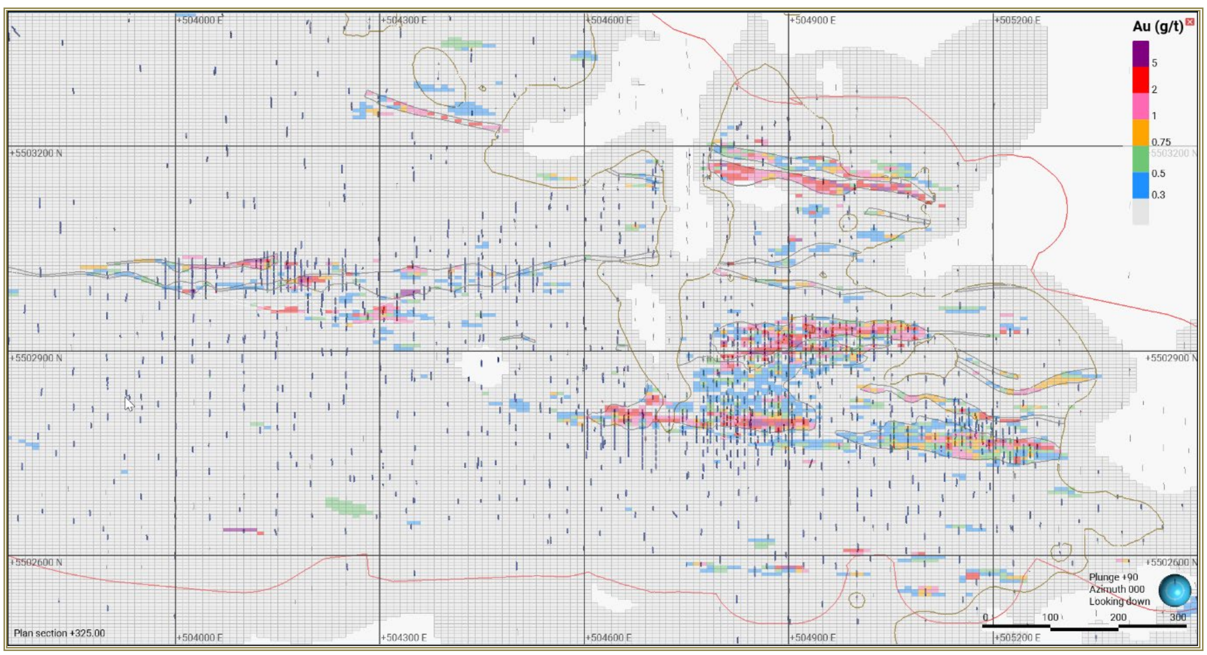


Figure 14-11: Plan View at 325 RL Showing Estimated Block Grades, Drill Holes, Mineralized Domains, and 2024 Pit Design (in Red)

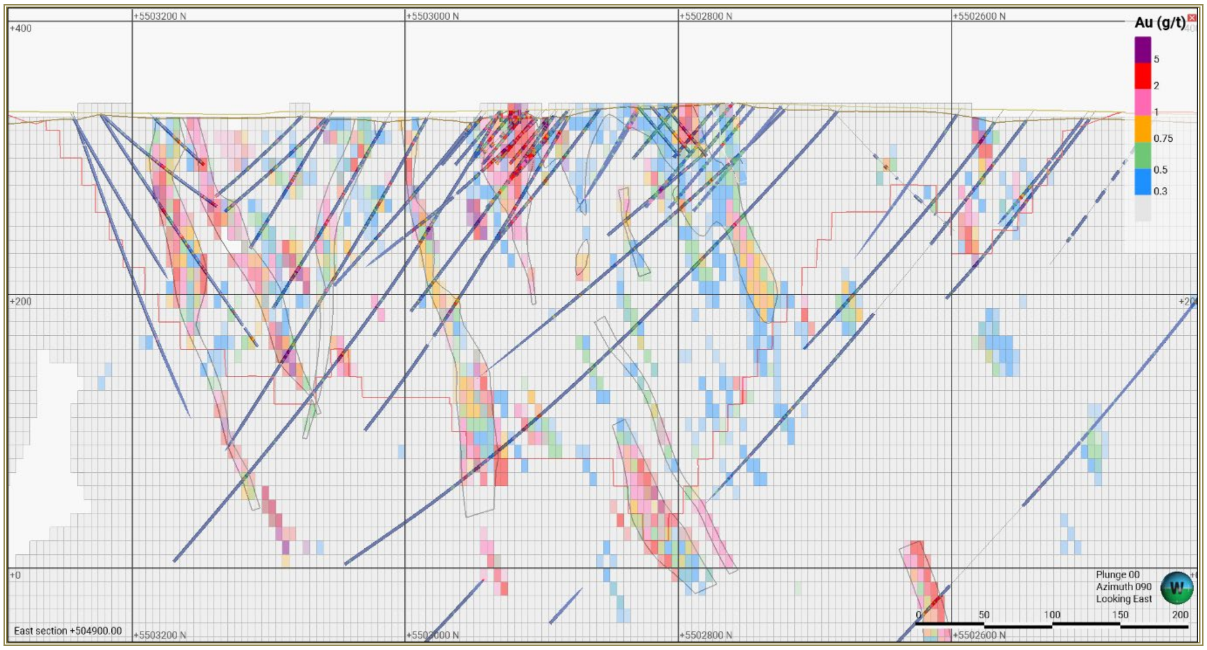


Figure 14-12: Section 504,00 m E Showing Estimated Block Grades, Drill Holes, Mineralized Domains, and 2024 Pit Design (in Red)

14.3.8 Block Model Validation

Various block model validation steps were taken to ensure that the block model is a robust representation of the composites and mineralization trends. 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.
- Visual check of high-grade blocks in relation to high-grade samples to ensure limited influence.
- 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.

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-hole intersections. No excessive over-extrapolation of grade was observed, and the block grades were found to be a good representation of the composite grades.

Global Statistical Validation

Table 14-9 shows a comparison of the mean block and declustered composite grades for the mineralized domains considering Pass 1, 2, and 3 within the 2024 MII pit optimization of the deposit. For the 22 principal domains, the declustered, capped composite-mean gold grade and block-mean gold grade are very similar (1.16 g/t vs. 1.18 g/t).

Table 14-9: Comparison of the Block- and Composite-Mean Gold Grades within In-Pit Area for the Mineralized and External Grade-Shell Domains

Block Code	2 m Composites						Blocks				
	No. Obs	Min. (g/t Au)	Max. (g/t Au)	Mean (g/t Au)	Mean Declustered	CV	No. Blocks	Min. (g/t Au)	Max (g/t Au)	Mean (g/t Au)	CV
3210	423	0.00	20.00	1.51		2.00	605	0.01	10.61	1.48	1.06
3505	476	0.00	4.86	0.42		1.33	406	0.02	4.17	0.46	0.77
3510	419	0.01	3.47	0.43		1.16	2,422	0.01	2.22	0.42	0.61
3600	1,409	0.00	40.70	2.04		1.98	5,143	0.01	27.21	1.90	1.05
3605	1,111	0.01	20.00	1.00		2.05	2,515	0.01	10.51	1.01	1.04
3610	196	0.00	15.36	1.04		2.35	726	0.01	6.37	0.87	1.02
3615	100	0.01	8.09	0.63		1.94	327	0.02	5.61	0.95	0.84
3620	43	0.08	2.82	0.68		0.86	137	0.23	1.58	0.73	0.34
3710	831	0.00	15.30	1.03		1.66	3,715	0.01	9.98	1.06	0.84
3720	254	0.00	13.70	1.45		1.49	742	0.00	8.76	1.48	0.84
3800	758	0.00	25.00	1.01		1.99	1,621	0.02	14.50	1.06	0.90
3805	67	0.00	6.24	0.67		1.71	276	0.01	2.49	0.64	0.72
3900	470	0.00	22.61	1.01		2.30	2,601	0.01	10.92	0.88	1.15
3905	299	0.00	7.66	0.78		1.51	1,501	0.00	4.16	0.70	0.82
4000	59	0.01	4.64	1.12		1.09	134	0.03	5.71	1.39	0.78
31051	1,625	0.00	2.93	0.27		1.18	3,026	0.01	2.15	0.28	0.70
31056	3,583	0.00	100.00	1.87		2.26	8,364	0.03	38.01	1.69	1.09
32051	3,047	0.00	6.00	0.24		1.53	7,316	0.01	3.34	0.24	0.78
32056	2,793	0.00	77.26	1.93		2.32	11,952	0.01	40.44	1.93	1.07
33001	1,115	0.01	2.24	0.28		1.09	2,965	0.01	1.19	0.30	0.52
33006	1,256	0.01	53.83	1.76		2.61	5,534	0.01	37.35	1.75	1.27
33051	125	0.01	1.73	0.13		2.25	228	0.01	1.14	0.19	1.11
33056	369	0.00	15.00	1.73		1.43	2,156	0.01	7.91	1.73	0.67
34051	1,185	0.00	2.05	0.25		1.08	3,936	0.00	0.82	0.25	0.55
34056	1,264	0.00	59.00	1.65		2.32	5,585	0.01	32.29	1.66	0.99
35001	5,432	0.00	3.58	0.32		0.88	8,233	0.00	4.73	0.35	0.68
35006	6,179	0.00	125.00	1.53		2.66	13,239	0.00	46.64	1.50	1.22
ALL WF	**34,889**	**0.00**	**125.00**	**1.09**	**1.16**	**1.85**	**95,405**	**0.00**	**46.64**	**1.18**	**1.35**
501	18,505	0.000	5.36	0.34		1.32	60,143	0.00	2.29	0.25	0.69
506	2,456	0.002	75.39	4.30		2.36	6,617	0.00	33.02	2.09	1.21

Notes: CV = coefficient of variation; WF = wireframe.

Local Statistical Validation—Swath Plots

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 2024 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 well-informed areas can occasionally show some discrepancies between the grades.

Figure 14-13 illustrates a swath plot of gold grades for the SP Zone by easting (subdomains were grouped to produce swath plots). Peaks and lows in gold content generally match peaks and lows in composite frequency; no bias was found in the MRE in this regard.

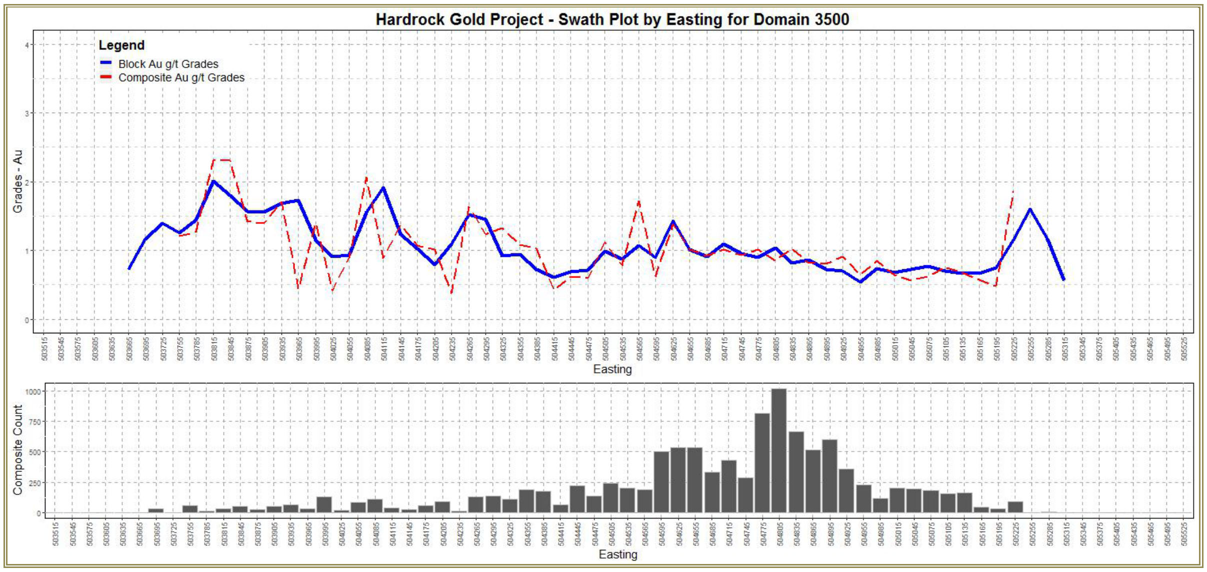


Figure 14-13: Swath Plot of Gold g/t for the SP Zone by Easting (Pass 1 to 3) within the In-Pit Area

Grade Smoothing and 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), ID^3, and ordinary kriging (OK) interpolators. 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 14-14 shows an example Q:Q plot for the F Zone (3105 rock code; subdomains were grouped to produce Q:Q plots).

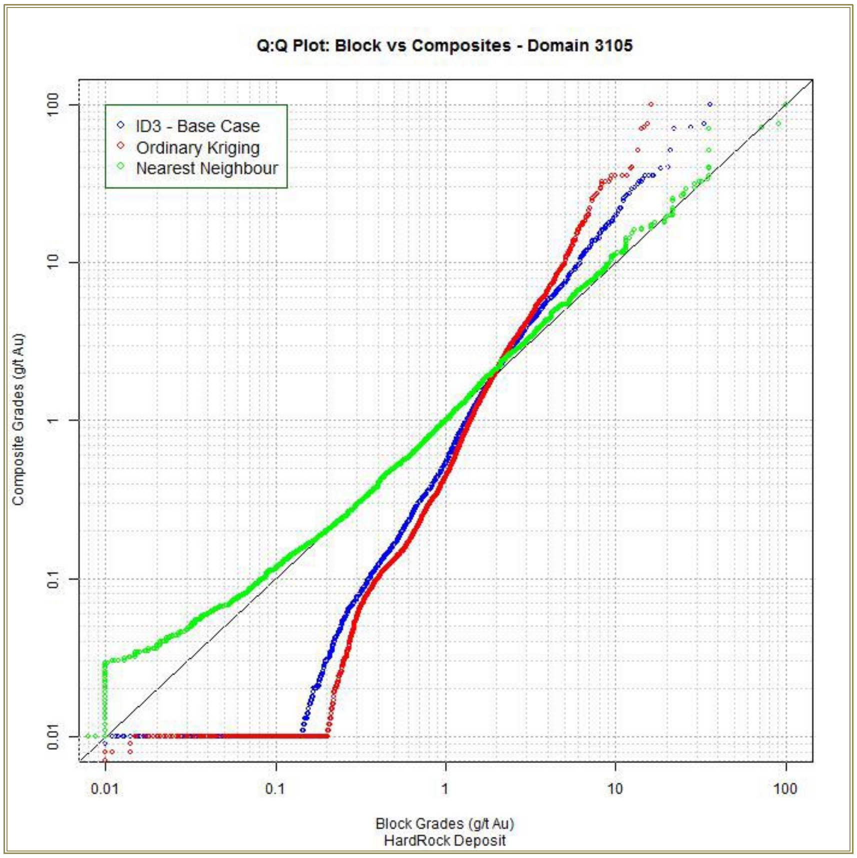


Figure 14-14: Q:Q Plot Comparing the Three Grade Interpolators for the F Zone (3105) within the MII Pit Optimization

Reconciliation Studies

Per CIM Guidelines, for Mineral Resource block models of deposits that have had mine production or are currently being mined, the validation should include a reconciliation of production against the Mineral Resource model, to the extent that reconciliation data are available and are in a format suitable for comparison purposes. These reconciliation studies are useful in evaluating both the long-term and short-term accuracy of the data collection, sample collection, preparation, and analysis procedures, and modelling procedures and parameters used to prepare the Mineral Resource block models.

GGM's mill started in April 2024, and the MRE QP believes that presenting the preliminary reconciliation work based on such a short time will be misleading. Reconciliations should cover a period of six months or more to be named "representative."

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 overestimation or underestimation of gold grades. Local statistical validations illustrate good local correlation between the interpolated blocks and the composite for gold grades. No obvious overestimation of gold grades was observed during the validation of estimated grades for the Hardrock gold deposit.

14.3.9 Mineral Resource Classification

Mineral Resource Classification Definition

The Mineral Resource classification definitions used for this report are those published by the CIM. The *CIM Definition Standards for Mineral Resources and Mineral Reserves*. provides standards for the classification of Mineral Resources and Mineral Reserve estimates into various categories. The category to which a Mineral 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 those data and information.

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all of an Inferred Mineral Resource could be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Merely having confidence in the estimate is insufficient to allow the meaningful application of technical economic parameters 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.

Resource Classification for the Greenstone Mine

Mineral Resource classification of Measured, Indicated, and Inferred was undertaken for any blocks meeting all the conditions below:

- Measured Mineral Resources are defined as blocks within ~15 m of the RCGC drilling; stockpile materials at the end of June 2024 are also included in this category.
- 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 believes that this step was necessary to homogenize (smooth out) the resource volumes in each category. Nevertheless, the condition where the distance to the closest composite is less than 35 m prevailed when finalizing the 2024 classification. In addition, any blocks located within the external grade shell domains (500, 501, or 506) in the underground Mineral Resources (below the 2024 MII pit optimization) were recategorized as Inferred.

Figure 14-15 and Figure 14-16 show the Mineral Resource classification, as well as the 2024 Whittle-optimized pit shell delimiting the in-pit and underground Mineral Resources.

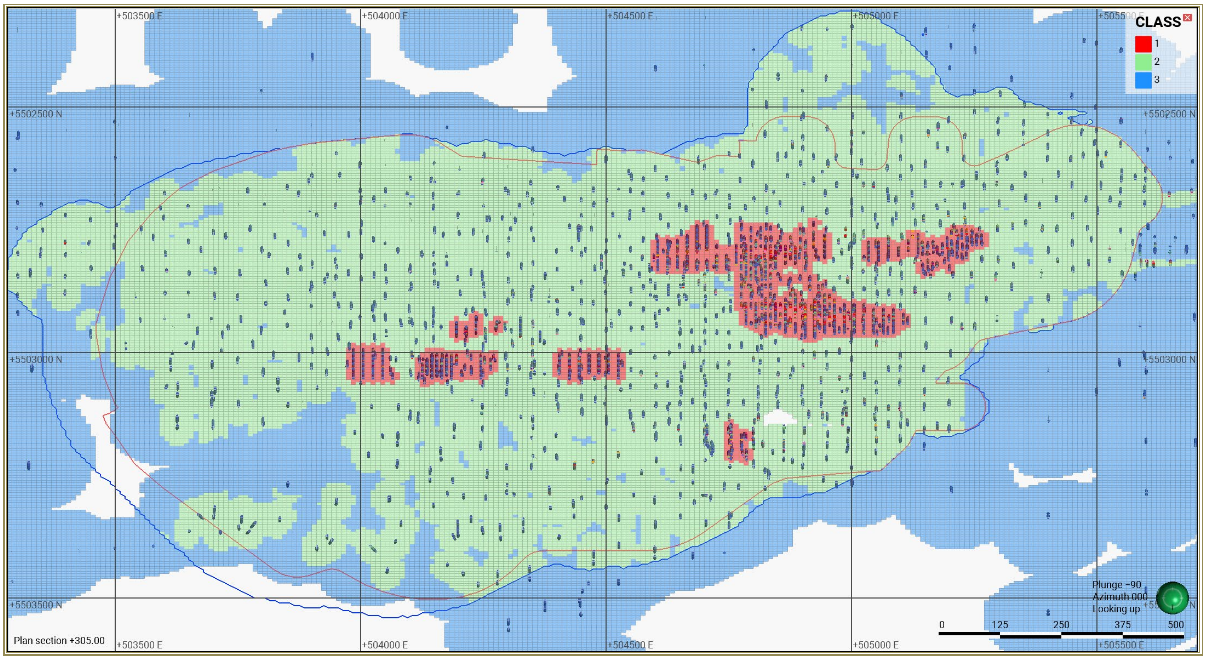


Figure 14-15: Plan View (305 m Elevation) Showing the Categorized Mineral Resources (1 = Measured, 2 = Indicated, 3 = Inferred), Drill Holes, the Resource Shell (Blue), and Design Pit (Red)

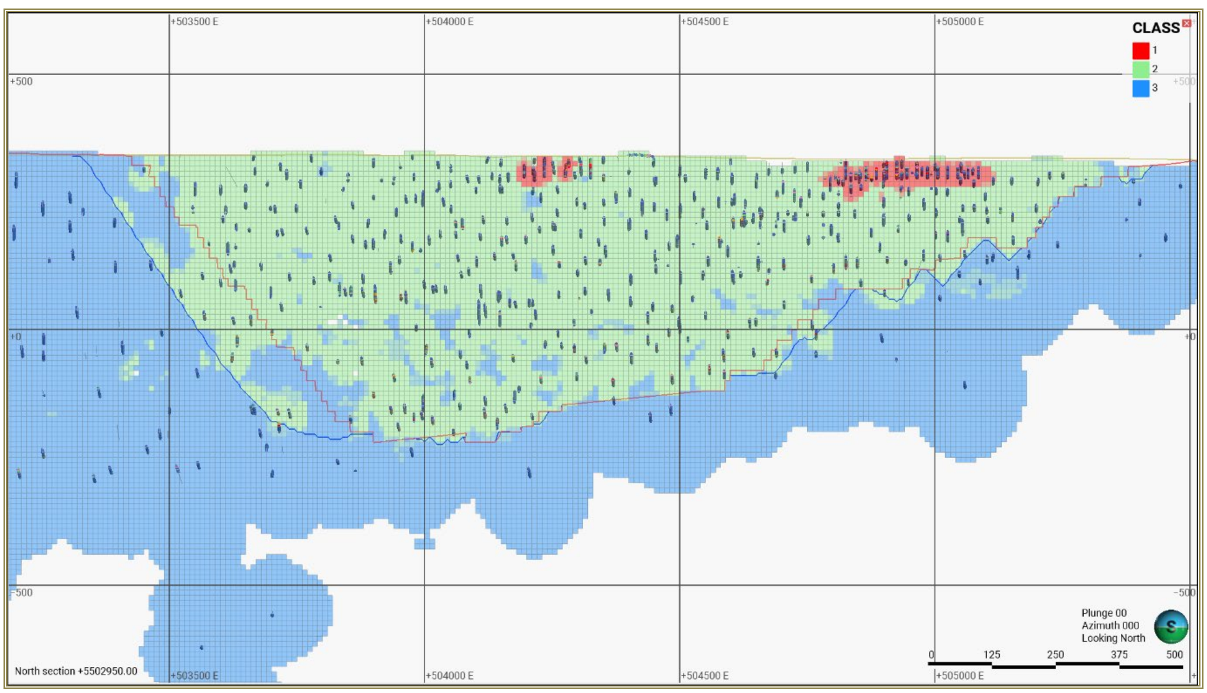


Figure 14-16: Longitudinal View Looking North (5,502,950 mN) Showing the Categorized Mineral Resources (1 = Measured, 2 = Indicated, 3 = Inferred), Drill Holes, the Resource Shell (Blue), and Design Pit (Red)

Stockpiles

The stockpiles as of June 30, 2024, are estimated at around 2.0 Mt, averaging 0.98 g/t Au, for 62,270 oz of contained gold. The breakdown is presented in Table 14-10.

Table 14-10: 2024 Stockpile Materials—Measured Category

Material	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)
Bin 1 (>1.10 g/t)	399	2.16	28
Bin 2 (0.50–1.10 g/t)	1,034	0.69	23
Bin 3 (0.35–0.50 g/t)	473	0.41	6
Magnetic Material	69	2.23	5
Material with Tramp Metal	9	1.44	0
Total Stockpile	**1,985**	**0.98**	**62**

14.3.10 In-Pit Constrained Mineral Resources (Inclusive of Mineral Reserves)

To determine in-pit Mineral Resources, pit optimizations were conducted at various gold prices using the 2024 MRE block model, including Measured, Indicated, and Inferred blocks. The final, selected Whittle input parameters and cut-off grade parameters used to define the 2024 in-pit Mineral Resource are defined in Table 14-11.

GGM chose a pit optimization based on a gold price of $1,700. A low revenue factor was chosen due to constraints such as the surrounding lakes, Mine 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 2024 MRE pit optimization.

Table 14-11: 2024 Hardrock—Pit Optimization Parameters

Parameter	Unit	Value
Mining Cost (LOM average)	$/t	2.34
Mining Recovery	%	98.8
Mining Dilution	%	17.2
Slope Angle	°	55° rock, 25° overburden
Processing Cost	$/t	6.98
G&A	$/t	3.31
Others	$/t	0.92
Total Ore-Based Cost	$/t	11.20
Cut-Off Grade	g/t Au	0.30
Gold Price	$	1,700
Exchange Rate	USD/CAD	1.00:1.28
Maximum Mining Rate	Mt/a	72.0
Processing Rate	Mt/a	9.86
Discount Rate	%	5

Table 14-12 displays the results of the 2024 MRE for Measured and Indicated categories for the in-pit portion of the Hardrock deposit at various cut-off grades to illustrate sensitivity. The Inferred Mineral Resources are similarly presented in Table 14-13.

Table 14-12: 2024 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/t Au)**	**Tonnes (kt)**	**Grade (g/t Au)**	**Gold (koz)**	**Zone**	**Cut-Off (g/t Au)**	**Tonnes (kt)**	**Grade (g/t Au)**	**Gold (koz)**
All Zones	>1.00	1,743	2.55	143	All Zones	>1.00	60,647	2.38	4,650
	>0.90	3,919	1.68	211		>0.90	67,672	2.24	4,865
	>0.80	4,150	1.63	218		>0.80	75,561	2.09	5,080
	>0.70	4,373	1.59	223		>0.70	84,107	1.95	5,286
	>0.60	4,705	1.52	230		>0.60	94,108	1.82	5,494
	>0.50	5,034	1.46	236		>0.50	106,451	1.67	5,712
	>0.40	5,660	1.34	245		>0.40	124,433	1.49	5,970
	>0.30	**6,733**	**1.18**	**256**		**>0.30**	**154,012**	**1.27**	**6,298**
	>0.20	8,211	1.02	268		>0.20	196,476	1.05	6,636

Table 14-13: 2024 In-Pit Inferred Mineral Resources (Inclusive of Mineral Reserves) at Various Cut-Off Grades for the Hardrock Deposit—Inferred Category

Inferred Resource				
Zone	**Cut-Off (g/t Au)**	**Tonnes (kt)**	**Grade (g/t Au)**	**Gold (koz)**
All Zones	>1.00	7,091	2.32	528
	>0.90	7,757	2.20	549
	>0.80	8,493	2.08	569
	>0.70	9,222	1.98	586
	>0.60	9,938	1.88	601
	>0.50	10,793	1.78	616
	>0.40	11,919	1.65	633
	>0.30	**13,387**	**1.51**	**649**
	>0.20	15,811	1.31	668

14.3.11 Underground Mineral Resources

The cut-off grade for 2024 underground Mineral Resources remains unchanged from the 2019 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/t Au allowed the mineral potential of the deposit to be outlined for the underground mining option, beneath the 2024 MRE pit optimization shell. The estimation of the underground cut-off grade was based on the parameters presented in Table 14-14.

Table 14-14: Input Parameters used for the Underground Cut-Off Grade (U CoG) Estimation—Hardrock Deposit

Input Parameter	Unit	Value
Exchange Rate	USD/CAD	1.00:1.28
Gold Price	$/oz	1,700
Gold Selling Costs	$/oz	2.34
Royalty	%	4.70
Net Gold Price	**$/oz**	**1,616.97**
Mining Costs	$/t	70.31
Milling Costs	$/t	6.98
G&A Costs	$/t	1.56
Total Costs	$	**100.93**
Processing Recovery	%	91.1
Mining Dilution	%	20
Marginal Cut-Off Grade	**g/t Au**	**2.00**

The 2024 underground MRE presented herein uses a rounded value of 2.00 (g/t Au) for the lower cut-off grade. The underground MRE is defined by blocks that are located beneath and adjacent to the 2024 MII pit optimization.

Table 14-15 displays the results of the MRE for the in situ underground portion of the Hardrock deposit at various cut-off grades to illustrate sensitivity.

Table 14-15: 2024 Underground Mineral Resources at Various Cut-Off Grades for the Hardrock Deposit—Indicated and Inferred Category

Indicated Resource					Inferred Resource				
Zone	Cut-Off (g/t Au)	Tonnes (kt)	Grade (g/t Au)	Gold (koz)	Zone	Cut-Off (g/t Au)	Tonnes (kt)	Grade (g/t Au)	Gold (koz)
All Zones	>5.00	2,430	8.34	651	All Zones	>5.00	3,260	8.22	861
	>4.50	3,030	7.62	743		>4.50	4,214	7.43	1,007
	>4.00	3,719	7.00	837		>4.00	5,736	6.58	1,214
	>3.50	4,704	6.31	955		>3.50	7,647	5.87	1,443
	>3.00	6,135	5.59	1,103		>3.00	10,199	5.21	1,709
	>2.50	7,990	4.93	1,267		>2.50	13,980	4.53	2,038
	>2.00	**10,959**	**4.20**	**1,480**		**>2.00**	**19,479**	**3.88**	**2,432**
	>1.50	16,078	3.41	1,764		>1.50	28,118	3.22	2,914

14.3.12 Summary of the 2024 Hardrock Mineral Resource

The 2024 MRE update presented in this section (Table 14-16) includes a compilation of:

- An in-pit MRE, within the 2024 Whittle-optimized shell (Table 14-12 and Table 14-13)
- An underground MRE, outside the 2024 Whittle-optimized pit shell (Table 14-15).

Table 14-16 presents the combined resources by resource category for the Hardrock Deposit.

Table 14-16: Summary of 2024 MRE (Inclusive of Mineral Reserves) for the Greenstone Mine

Category	In-Pit >0.3 g/t Au			Underground >2.0 g/t Au		
	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)
Measured	6,733	1.17	256	-	-	-
Indicated	154,012	1.27	6,298	10,959	4.20	1,480
M+I	160,745	1.27	6,554	10,959	4.20	1,480
Inferred	13,387	1.51	649	19,479	3.88	2,432

Notes:

- The Independent and Qualified Person for the MRE, as defined by NI 43-101, is Réjean Sirois, B.Sc., P.Eng., of GMS, and the effective date of the estimate is June 30, 2024.
- These Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability.
- Mineral Resources are reported inclusive of Mineral Reserves.
- In-pit results are presented undiluted within a merged surface of the pit optimization shell at $1,700/oz Au and the 2024 pit design at a cut-off grade of 0.30 (g/t Au).
- Underground Mineral Resources are presented undiluted and are defined as blocks below and adjacent to the 2024 pit optimization at a cut-off grade of 2.00 (g/t Au).
- Any discrepancies in the totals are due to rounding effects.
- 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: Reference mining cost: $1.97/t, Incremental bench cost ($/10 m bench): $0.03, Milling cost: $6.98/t, Royalty: 3.0%, G&A: $3.31/t, Sustaining capital: $0.92/t, Gold price: $1,700/oz, Milling recovery: 91.1% and Exchange rate 1.28 CAD/USD.

14.3.13 Comparison with the Previous Estimate

A comparison of the 2024 MRE (in-pit and underground) to the 2019 MRE is presented in Table 14-17.

The overall 2024 combined in-pit and underground Measured and Indicated Mineral Resources of 8,034,000 ounces of gold for the Greenstone Mine represents a 13% increase in total ounces versus the 2019 estimate (at their respective cut-off grades).

The 2024 in-pit Measured and Indicated Mineral Resources represent a 5% decrease in grade, a 12% increase in ounces, and a 17% increase in tonnes versus in-pit Measured and Indicated Resources reported in the 2019 feasibility study.

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

- The 2024 MRE is constrained by a deeper pit optimization, which incorporates significantly more Mineral Resources compared to the 2019 MRE.

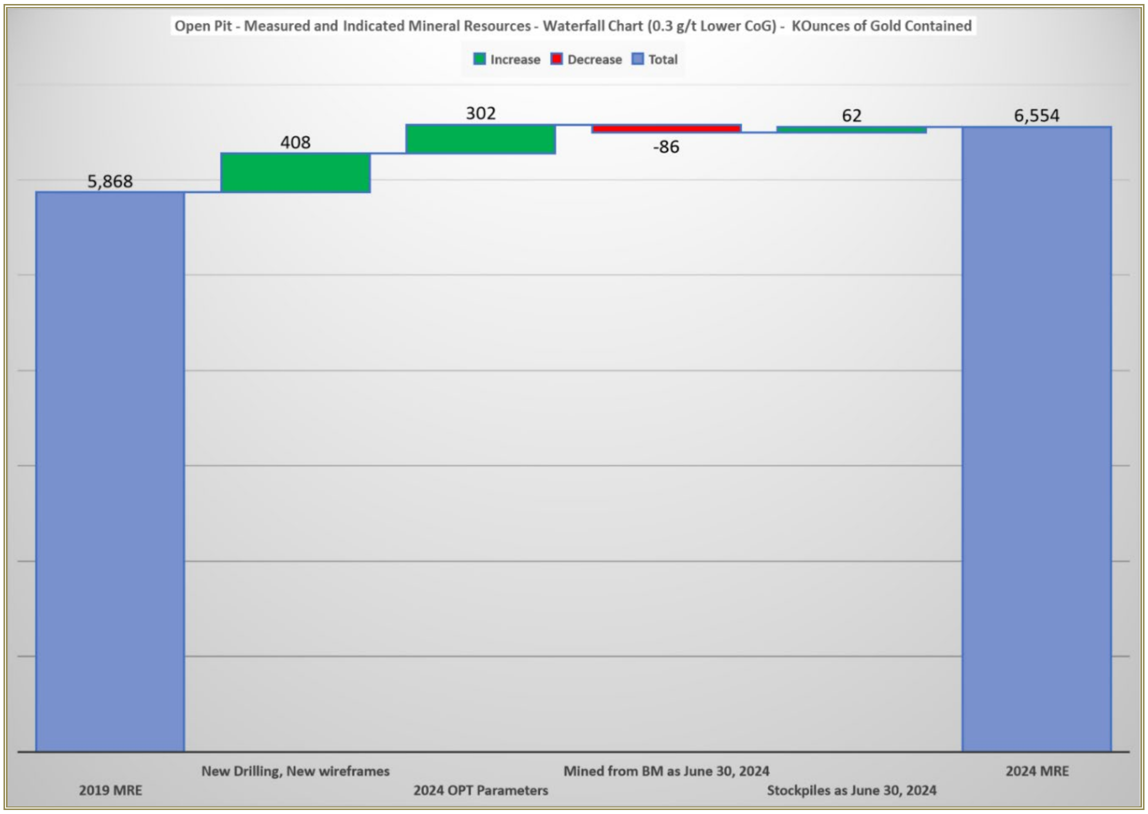
- The Southeast extension drilled in 2022 shows usually lower gold grades than the previous 2019 average of the deposit. This has resulted in a slightly lower grade for the 2024 Mineral Resource.
- Grade capping was revisited in 2024 (due to the refined wireframes), and new capping thresholds were chosen. They are similar to the grade capping chosen in 2019, and they have only a small effect on the results.
- RC grade-control drilling and validation diamond drilling conducted in 2018, 2019, and 2021 confirmed grade continuity, and generally intersected similar gold grades as those expected in the 2019 block model. Despite the derisking drilling programs undertaken in during those years, these RC grade-control programs cover only a small portion of the entire Mineral Resource.

A gold price of $1,700 was used during the pit optimization process in 2024, versus $1,500 used for the 2019 estimate; therefore, this increase has contributed to the increased value of the new Mineral Resource. Table 14-17 summarizes the changes in the 2024 MRE update compared to the 2019 MRE. The majority of the increase in the global Measured and Indicated ounces between the 2019 and 2024 MRE is a result of the deepening of the pit optimization shell and the increase in gold price. The global increase in the Inferred Mineral Resources in the open pit of 613 koz was counterbalanced by the loss of 627 koz in the underground Mineral Resources, mainly due to lowering the pit shell.

Table 14-17: Summary of Changes in 2024 MRE vs. 2019 Estimates

		In Pit >0.3 g/t Au			Underground >2.0 g/t Au		
		Tonnage (kt)	**Gold Grade (g/t)**	**Contained Gold (koz)**	**Tonnage (kt)**	**Gold Grade (g/t)**	**Contained Gold (koz)**
Measured & Indicated Resources (M+I)	2019 MRE	137,700	1.33	5,868	9,800	3.93	1,237
	2024 MRE	160,800	1.27	6,554	11,000	4.2	1,480
	2024 vs. 2019	+23,100	−0.06	+686	+1,200	+0.27	+243
Inferred Resources	2019 MRE	900	1.19	36	24,600	3.87	3,059
	2024 MRE	13,400	1.51	649	19,500	3.88	2432
	2024 vs. 2019	+12,500	+0.32	+613	−5,100	+0.01	−627

Figure 14-17 represents the variation of the Measured and Indicated MRE for the open pit portion since 2019 (cut-off grade [COG] = 0.3 g/t Au). The 686 koz increase comes mainly from the drilling and the new wireframes used in the internal 2022 MRE, and the expanded resource shell. The only decrease is attributed to mining to June 30, 2024.



Block Model Version	Pit Shell	Domain	Tonnage (Mt)	Gold Grade (g/t)	Metal Content (koz)
2019	2019	Principal WF	98.9	1.48	4,710
		Grade Shells	38.8	0.93	1,158
		Total	**137.7**	**1.33**	**5,868**
2022 (Internal)	2022	Principal WF	112.5	1.42	5,138
		Grade Shells	38.2	0.93	1,138
		Total	**150.7**	**1.30**	**6,276**
NEW 2024 optimization parameters	2024	Principal WF	117.0	1.40	5,269
		Grade Shells	43.9	0.93	1,309
		Total	**160.9**	**1.27**	**6,578**
Mined from the BM as June 30, 2024	2024	Principal WF	1.6	1.37	73
		Grade Shells	0.5	0.79	13
		Total	**2.1**	**1.23**	**86**
Stockpiles as June 30, 2024	2024	Principal WF	-	-	-
		Grade Shells	-	-	-
		Total	**2.0**	**0.98**	**62**
2024	2024	Principal WF	115.4	1.40	5,196
		Grade Shells	43.4	0.93	1,296
		Stockpiles	2.0	0.98	62
		Total	**160.8**	**1.27**	**6,554**

Figure 14-17: In-Pit Constrained Measured and Indicated Mineral Resources Waterfall Chart (0.3 g/t Cut-Off Grade) of Contained Gold (koz)

Figure 14-18 represents the variation of the Inferred MRE for the open pit portion (COG = 0.3 g/t Au) since 2019. The Inferred Mineral Resource increased by 613 koz since the last technical report. The increases are coming mainly from drilling, new wireframes, and the changes in the Mineral Resource categorization used in 2024.

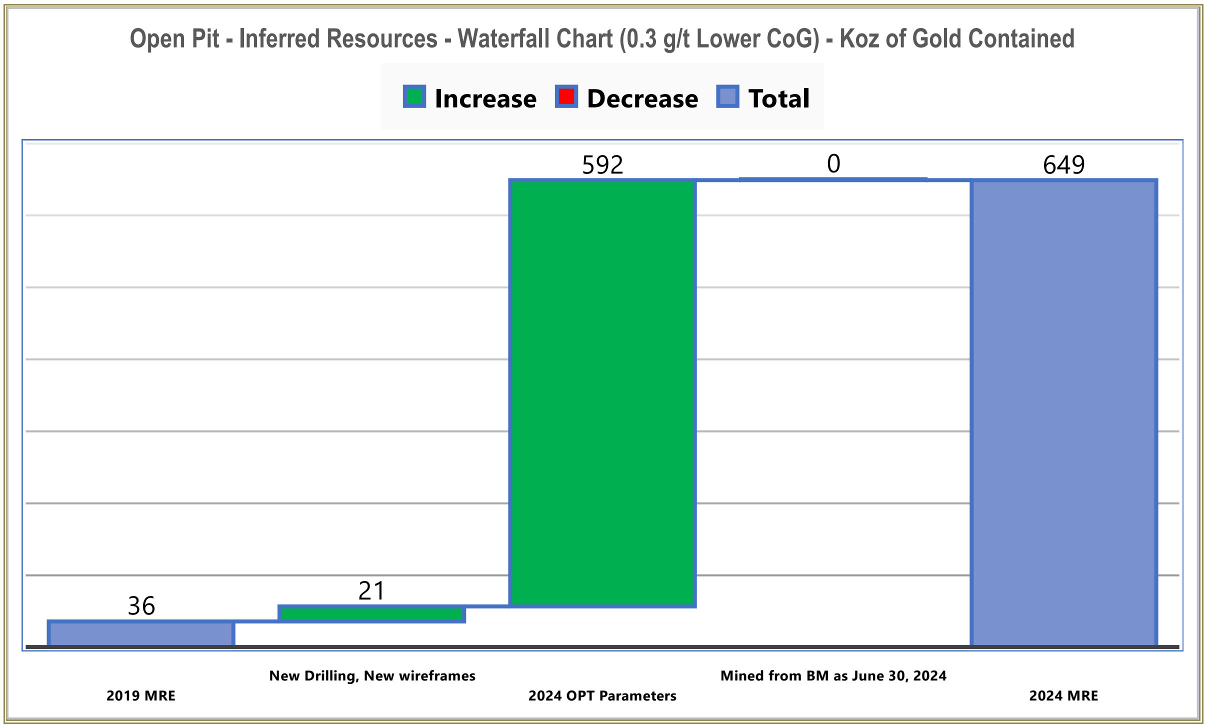


Figure 14-18: In-Pit Constrained Inferred Mineral Resources Waterfall Chart (0.3 g/t Cut-Off Grade) of Contained Gold (koz)

The total underground Mineral Resource remains relatively unchanged, apart from some reclassifications, as described below:

- 1.2 Mt (243 koz) increase in Indicated Mineral Resources:
  - These blocks were previously estimated within a poorly constrained lithological domain and have now been reclassified into the Indicated category.
- 5.1 Mt (627 koz) decrease 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, and the pit shell goes deeper than in 2019.

Figure 14-19 and Figure 14-20 represent the variation of the Indicated MRE for the underground portion since 2019 (COG = 2.0 g/t Au). The underground Indicated Mineral Resource base increased by 243 koz since the last Technical Report. The increase is due to improved classification based on new wireframes.

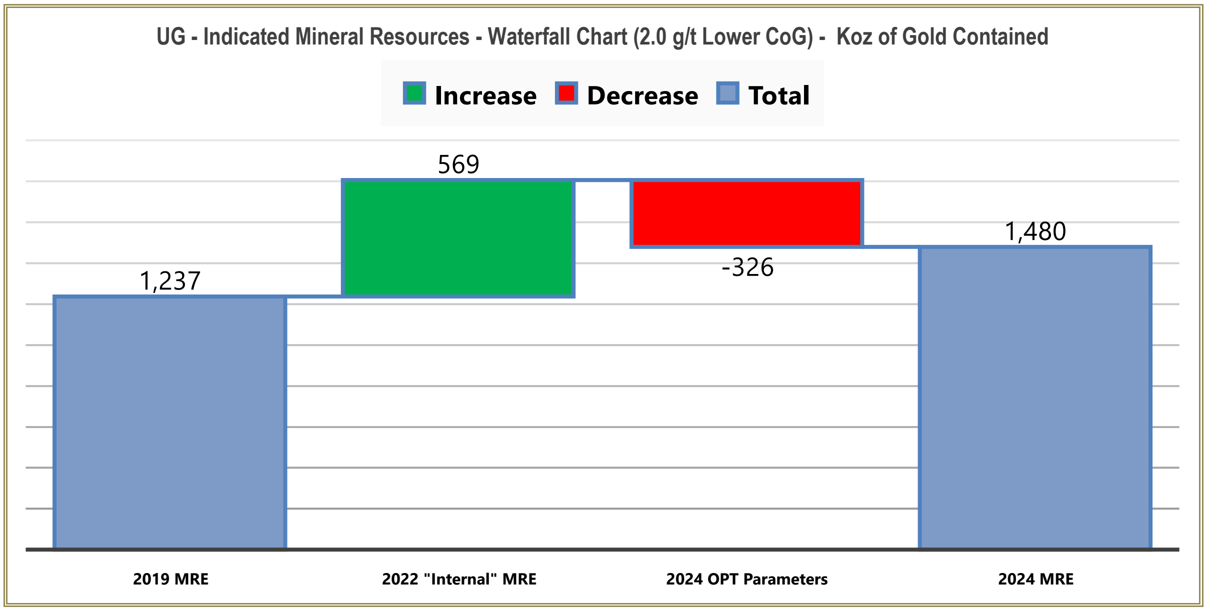


Figure 14-19: Underground Indicated Mineral Resources Waterfall Chart (2.0 g/t Cut-Off Grade) of Contained Gold (koz)

Figure 14-20 represents the variation of the Inferred MRE for the underground portion since 2019 (COG = 2.0 g/t Au). The underground Inferred Mineral Resource base decreased by 627 koz since the 2021 Report. The decrease comes mainly from the bigger 2024 Whittle shell compared to 2019 and the Mineral Resource reclassification done in 2024.

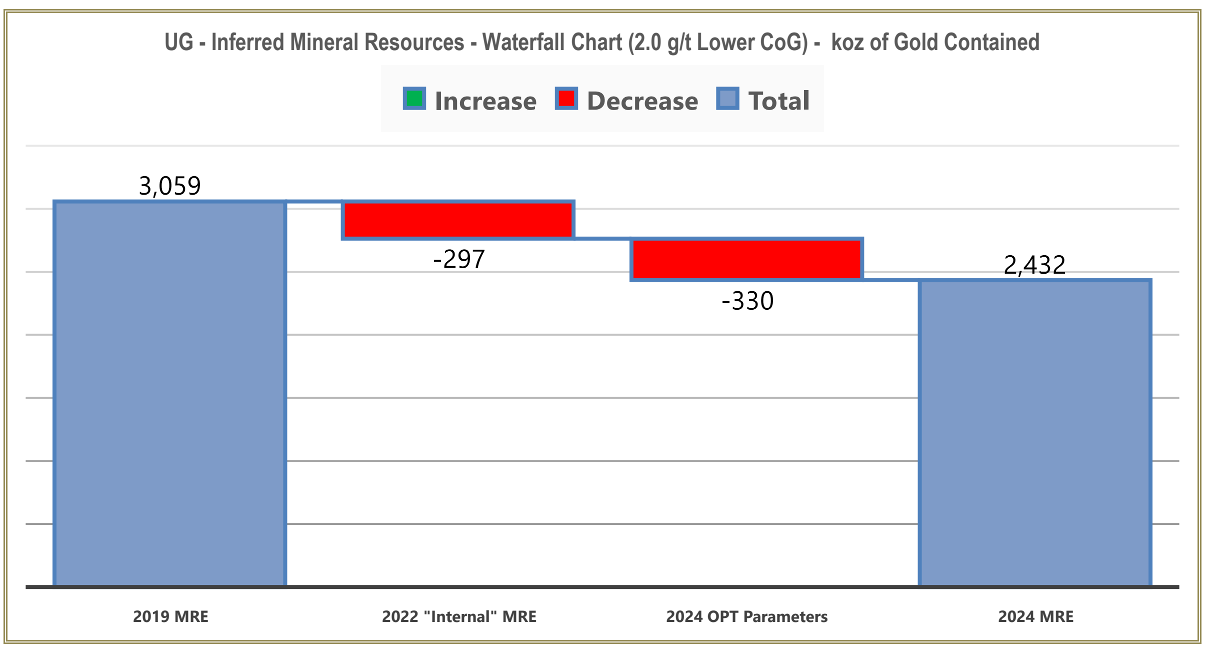


Figure 14-20: Underground Inferred Mineral Resources Waterfall Chart (2.0 g/t Cut-Off Grade) of Contained Gold (koz)

Figure 14-21 represents the 2024 MRE for the open pit portion (COG = 0.3 g/t Au) and for the underground extension (COG = 2.0 g/t Au). The overall extension of the modelled mineralized ore bodies reaches close to 5 km longitudinally.

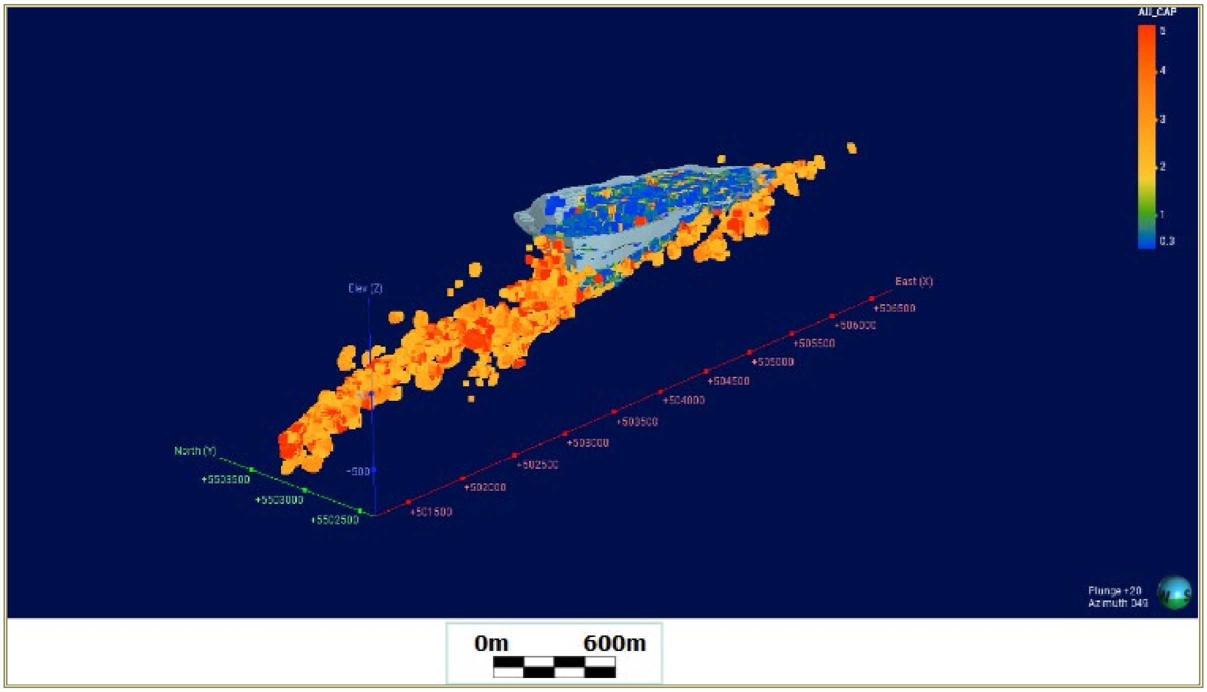


Figure 14-21: In-Pit and Underground Mineral Resources—Greenstone Mine

14.3.14 2024 In-Pit Constrained Mineral Resources (Exclusive of Mineral Reserves)

Equinox Gold publishes its Mineral Resources exclusive of Mineral Reserves (i.e., Mineral Resources are in addition to Mineral Reserves). Table 14-18 presents the 2024 in-pit Mineral Resources exclusive of Mineral Reserves for Indicated and Inferred categories. When reporting Mineral Resources exclusive of Mineral Reserves, no remaining Measured Mineral Resources exist at the Greenstone Mine.

The Mineral Reserve estimate is discussed in Section 15.

Table 14-18: 2024 In-Pit Mineral Resources (Exclusive of Mineral Reserves) for the Greenstone Mine

Category	In-Pit >0.3 g/t Au			Underground >2.0 g/t Au		
	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)
Measured	0	-	0	-	-	-
Indicated	19,008	1.21	738	10,959	4.20	1,480
M+I	19,008	1.21	738	10,959	4.20	1,480
Inferred	6,892	1.49	331	19,479	3.88	2,432

Notes:

- The Independent and Qualified Person for the MRE, as defined by NI 43-101, is Réjean Sirois, B.Sc., P.Eng., of GMS., and the effective date of the estimate is June 30, 2024.
- These Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability.
- Mineral Resources are presented **exclusive** of Mineral Reserves.
- In-pit results are presented undiluted within a merged surface of the pit optimization shell at $1,700/oz Au and the 2024 pit design.
- In-pit Mineral Resources are stated at a cut-off grade of 0.30 g/t Au.
- Underground Mineral Resources are presented undiluted and are defined as blocks below and adjacent to the 2024 pit optimization at a cut-off grade of 2.00 (g/t Au).
- Any discrepancies in the totals are due to rounding effects.
- 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: Reference mining cost: $1.97/t, Incremental bench cost ($/10 m bench): $0.03, Milling cost: $6.98/t, Royalty: 3.0%, G&A: $3.31/t, Sustaining capital: $0.92/t, Gold price: $1,700/oz, Milling recovery: 91.1% and Exchange rate 1.28 CAD/USD.

14.3.15 Underground Mineral Resources (Exclusive of Mineral Reserves)

Since no Mineral Reserves are estimated for the underground portion of the Hardrock deposit, the 2024 underground Mineral Resources exclusive of Mineral Reserves are identical to the underground Mineral Resource inclusive of Mineral Reserves.

14.4 Brookbank, Kailey, and Key Lake Deposits Mineral Resource Estimates

The previous MRE for the Brookbank, Kailey, and Key Lake deposits were prepared by Mr. James Purchase, P.Geo., Director of Geology and Resources at GMS at that time using Leapfrog EDGE (Version 5.1). The MRE is based on a drilling database for the three deposits provided to GMS on June 26, 2020. The effective date of the MRE for the Brookbank, Key Lake, and Kailey deposits was September 3, 2020.

In 2024, the new QP validated the work done by Mr. Purchase and agreed with the modelling, chosen parameters, and estimation processes that subsequently led to the MRE stated in that document.

Sections 14.4.1, 14.4.2, 14.4.3 were extracted from the 2021 Report with minor edits and updates.

All location data relating to these MRE are in UTM Zone 16N and the NAD 83 datum.

No Mineral Reserves have been calculated at the Brookbank, Kailey, and Key Lake deposits, Mineral Resources reported in this section are considered to be exclusive of Mineral Reserves.

14.4.1 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 targets named Foxear and Cherbourg, which form part of the same mineralized system along strike.

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, as well as the Foxear and Cherbourg targets.

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 Section 12.2.

Topography

GGM supplied topography 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.

Geological Modelling Approach

The Brookbank gold deposit and the Cherbourg and Foxear targets (Figure 14-22) occur at three different localities within the 6.5 km-long Brookbank shear zone. Gold mineralization occurs at or near the contacts between mafic volcanics and metasediments. The deposits have both structural and lithological controls; however, a second-order, subvertical plunge and continuity of mineralization is also apparent within the shear zone. Gold mineralization occurs within multiple quartz–carbonate stringers, veinlets, 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 20 to 50 m wide at or close to surface.

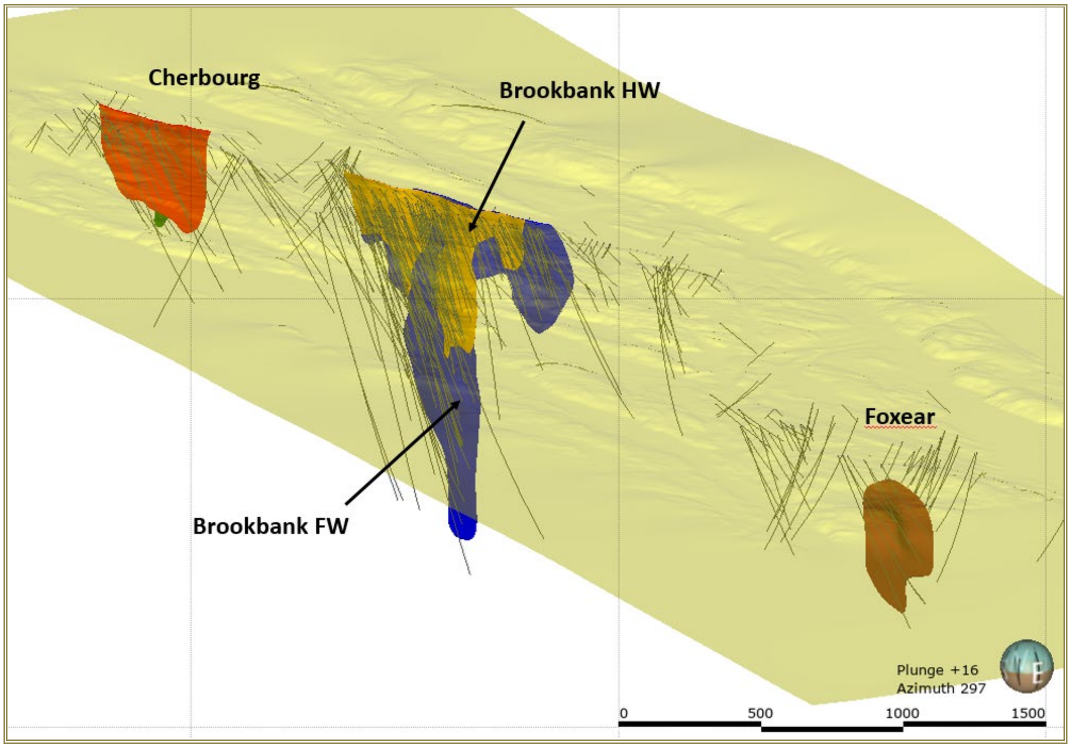


Figure 14-22: Modelled Solids of the Brookbank Deposits, Looking NW

The Brookbank Main deposit has been modelled as two discrete, continuous zones named the footwall (FW) and hanging wall (HW) veins (Figure 14-23). A minimum true width of 1.5 m and a 2 g/t Au 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 thick).

The Noranda-era drilling (1940s, 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 current resource estimation.

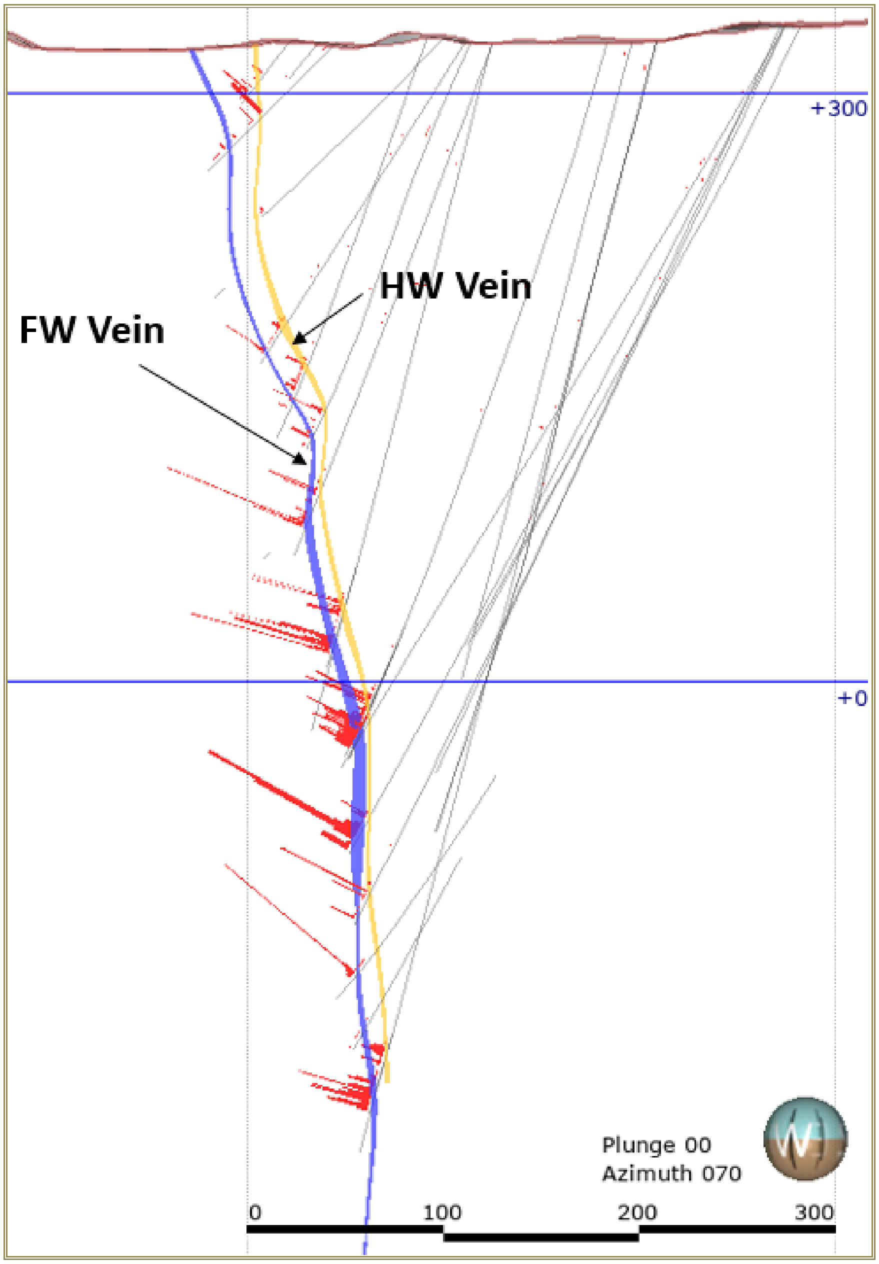


Figure 14-23: Typical Section (Looking East) Showing the Footwall (Blue) and Hanging Wall (Orange) Veins (Bar Charts on Drilling Traces show Gold Grades)

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 14-19. An example of the probability plot for the FW vein domain is shown in Figure 14-24.

Table 14-19: Length-Weighted Assays Statistics Showing Grade Capping Levels and Metal Loss Factors

Domain	No. of Assays	Max. (g/t Au)	Uncut Mean (g/t Au)	High-Grade Capping	Cut Mean (g/t Au)	No. 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: % metal loss factors calculated from length multiplied by grade and does not consider the spatial location of the outliers.
FW = footwall; HW = hanging wall.

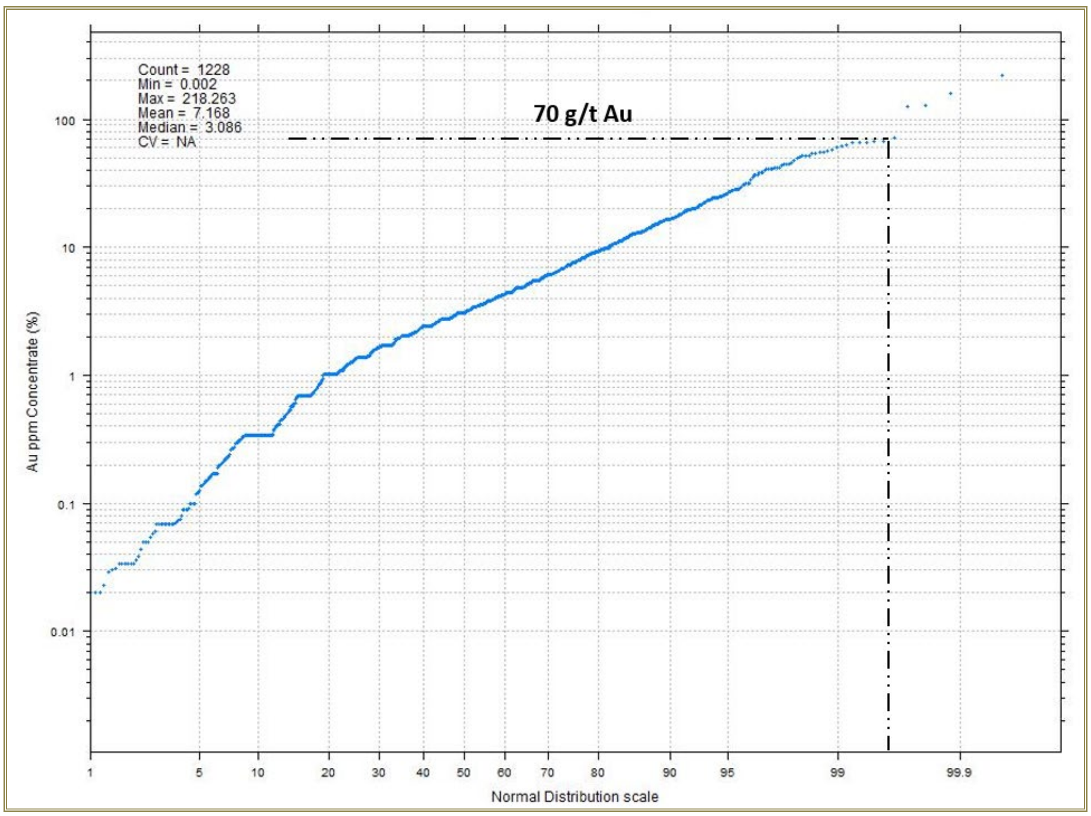


Figure 14-24: Example of a Probability Plot for the FW Vein

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/t Au).

Length-weighted composite statistics of drilling used in the estimation are shown in Table 14-20.

Table 14-20: Length-Weighted Composite Statistics of Capped Gold Grades by Domain

Domain	No. of 1 m Comps.	Min. (g/t Au)	Max. (g/t Au)	Mean (g/t Au)	Median (g/t Au)	SD	Variance	CV
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

Notes: FW = footwall; HW = hanging wall; SD = standard deviation; CV = coefficient of variation.

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. Pairwise 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 were insufficient data to model variograms for the Cherbourg and Foxear deposits. Variogram parameters are shown in Table 14-21.

Table 14-21: 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			

Notes: FW = footwall; HW = hanging wall.

Block Modelling

The block model definition is presented in Table 14-22. The upper limit was defined by the surface topography. The parent block size was based primarily on drill-hole spacing, envisaged selective mining unit 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 14-25 shows a plan view of the Brookbank block model location.

Table 14-22: Brookbank Main Deposit Block Model Attributes

Item	X	Y	Z
Origin Coordinates (m)	439,160.00	5,506,730.00	450
Block Extents (m)	1,480	678	1,600
Number of Parent Blocks	148	226	160
Parent Block Size (m)	10	3	10
Sub-Block Size (m)	2.5	1	2.5
Rotation		20° counter-clockwise	

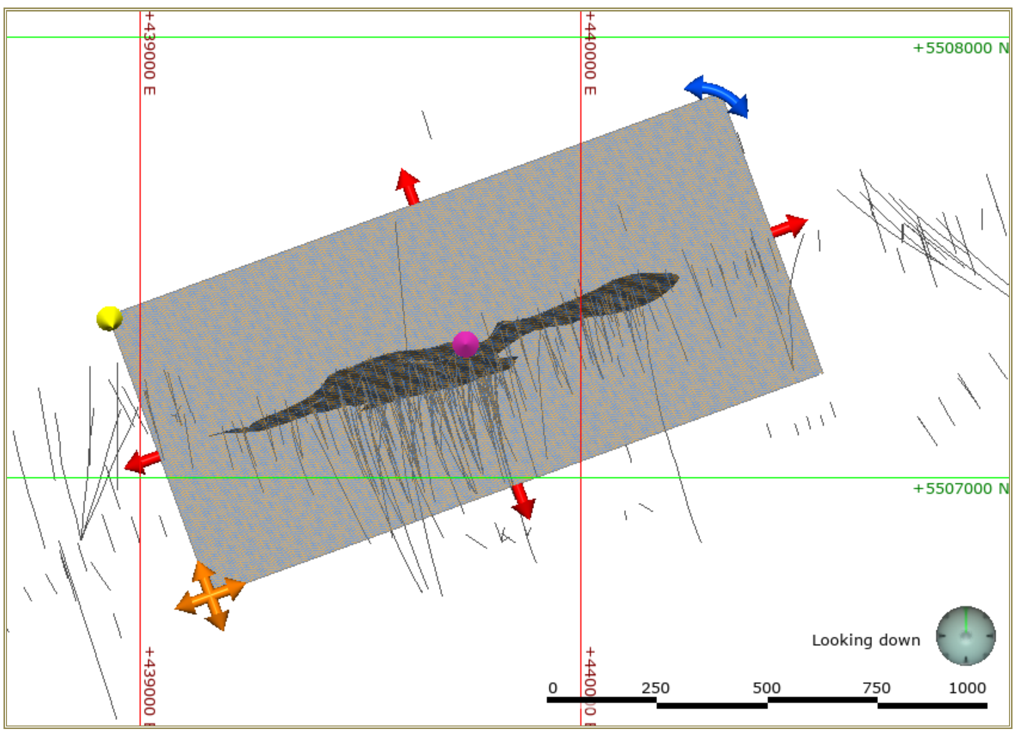


Figure 14-25: Plan View of Brookbank Block Model, Wireframes, and Drill Traces

Bulk-Density Data

Bulk-density data were supplied by GGM in the form of a Microsoft 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 14-23 presents the bulk-density data available for the Brookbank deposit.

Table 14-23: Statistical Summary of Bulk-Density Data for the Brookbank Deposit

Lithology	No. of Meas.	Mean Density (g/cc)	SD Density (g/cc)
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

Note: SD = standard deviation.

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 none is incorporated into the block model at this time.

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 midpoints 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, between 7 and 16 samples, and a maximum of 3 samples per drill hole was applied. This ensured that a minimum of 3 drill holes was required to estimate blocks in the first pass.

For Pass 2, between 4 and 16 samples, and a maximum of 3 samples per drill hole was applied. This ensured that a minimum of 2 drill holes was required to estimate blocks in the second pass.

For Pass 3, between 2 and 16 samples, and a maximum of 3 samples per drill hole was applied. This ensured that a minimum of 1 drill hole was required to estimate blocks in the third pass.

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 (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 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 14-24.

Table 14-24: Summary of Search Parameters—Brookbank Deposit

Domain	Pass	X (m)	Y (m)	Z (m)	Min. Samples	Max. Samples	Max. Samples/DH	High-Grade Restraining X (m)	High-Grade Restraining Y (m)	High-Grade Restraining Z (m)	High-Grade Restraining Threshold (g/t Au)
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: DH = drill hole; FW = footwall; HW = hanging wall.

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. Figure 14-26 presents 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.

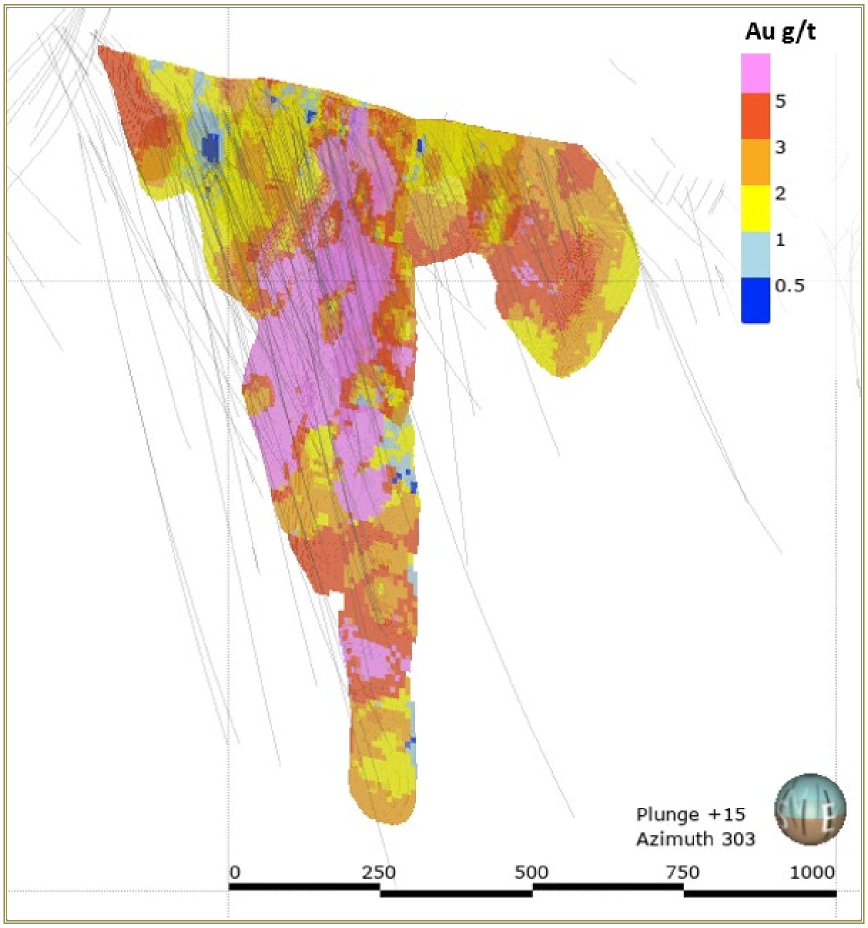


Figure 14-26: Brookbank Gold-Grade Distribution in the FW Domain, Looking NW

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 14-25), and local validations such as swath plots were used to ensure grade smearing was reduced to a minimum (Figure 14-27).

Table 14-25: Global Statistical Comparison between Blocks and Declustered Composites for all Estimation Passes at Brookbank

Domain	Composites			Blocks		Difference Mean (%)
	No. Comps.	Mean (g/t)	Mean Declustered (g/t)	No. Blocks	Mean (g/t)	
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

Notes: FW = footwall; HW = hanging wall.

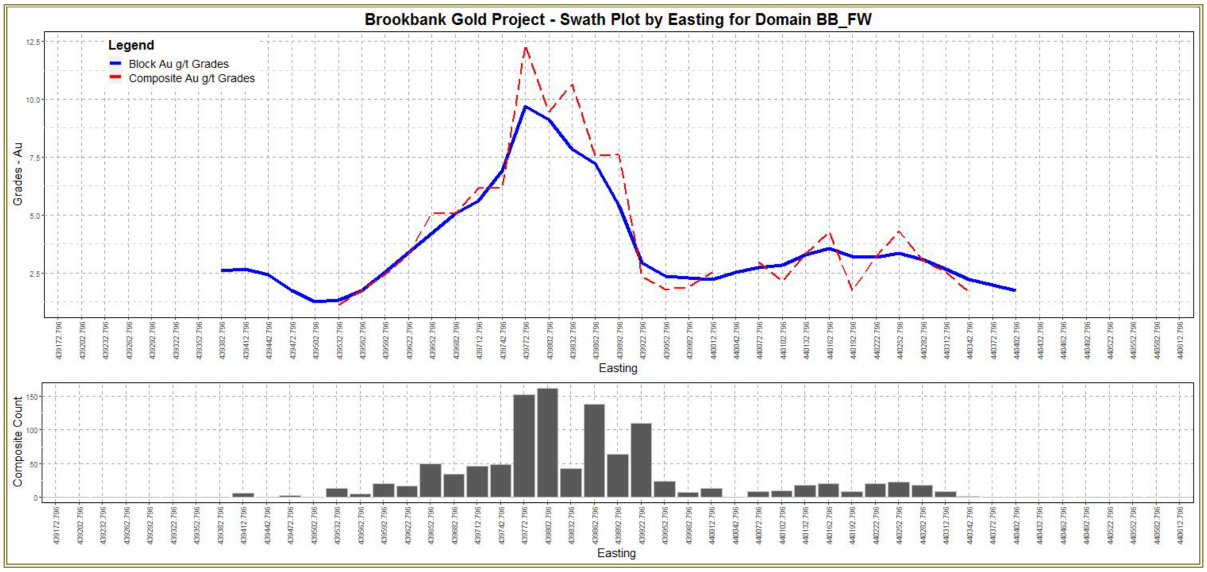


Figure 14-27: Swath Plot Comparing Block Gold Grades (Blue) with Capped Composite Gold Grades (Red Dotted) for the FW Domain, by Easting

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.

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 was derived from the RPA 2009 technical report, where testwork 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 were 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 14-26 and Table 14-27 show the various parameters and assumptions used in the open pit and underground analysis as well as the gold cut-off grades used for reporting the MRE.

Table 14-26: Economic Parameters used in the Open Pit Analysis

Item	Unit	Brookbank	Kailey	Key Lake
Open Pit Mining Cost (All Material)	$/t	3.08	3.08	3.08
Open Pit Dilution + Mining Loss	%	19.0	12.0	19.0
Ore Processing Cost	$/t	5.80	5.80	5.80
Ore G&A Cost	$/t	1.22	1.22	1.22
Gold Price	$/oz	1,500	1,500	1,500
Incremental Ore Haulage Cost	$/t	13.78	1.31	3.47
Mill Recovery	%	92	90	90
Exchange Rate	CAD/USD	1.30	1.30	1.30
Open Pit Calculated Gold Cut-Off Grade	**g/t Au**	**0.60**	**0.40**	**0.40**

Note: G&A = general and administrative.

Table 14-27: Economic Parameters used in the Underground Analysis

Item	Unit	Brookbank
Exchange Rate	CAD/USD	1.30
Discount Rate	%	6%
Gold Price	$/oz	1,500
Transport and Refining Cost	$/oz	2.31
Royalty Rate	% NSR	3
Net Gold Value	$/oz	1,452.69
Metallurgical Recovery	%	92
Mining Dilution	%	20
Mining Recovery	%	90
Processing Cost	$/t milled	6.15
Surface Ore Haulage	$/t milled	13.77
Diamond Drilling	$/t milled	1.15
Stope Preparation	$/t milled	15.38
Mining	$/t milled	19.23
Services—Surface	$/t milled	2.31
Service—Mine	$/t milled	6.15
Service—Mechanical	$/t milled	2.31
Electrical	$/t milled	1.54
Technical Services	$/t milled	1.92
G&A	$/t milled	7.69
Underground Calculated Gold Cut-Off Grade	g/t Au	2.40

After completing the Whittle pit optimization, the results were reimported 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 14-28.

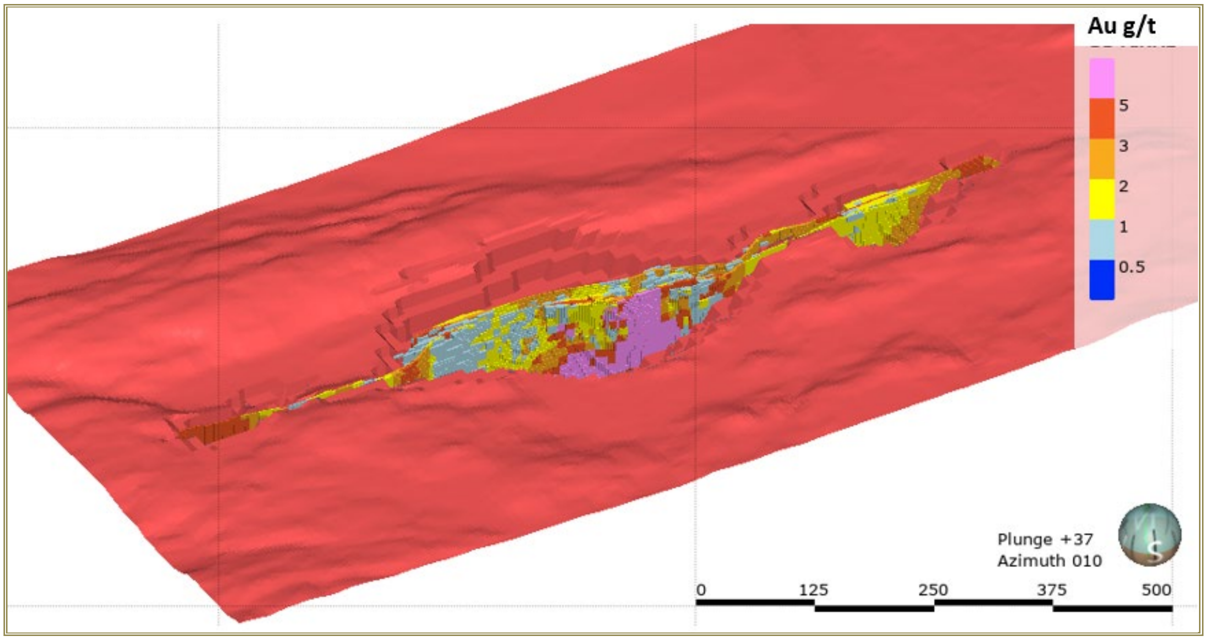


Figure 14-28: Brookbank Property Pit Optimization—$1,500 Pit Shell, MII Blocks

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 Mineral Resource categories are shown on Figure 14-29. At this stage, there are no Measured Mineral Resources for the Brookbank property.

The Indicated Mineral Resource category was assigned to coherent portions of the deposit covered by 40 x 40 m drill spacing, and estimated predominantly in Pass 1, including islands of Pass 2 encompassed within. Good visual evidence of adequate sample and drill-hole coverage was also considered.

The Inferred Mineral Resource category was assigned to areas outside of the 40 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 domains were not classified as Mineral Resources as they contained insignificant tonnage.

The Foxear and Cherbourg targets were deemed too small to meet the requirements for Reasonable Prospects for Eventual Economic Extraction; however, they do represent good near-term targets for expansion drilling.

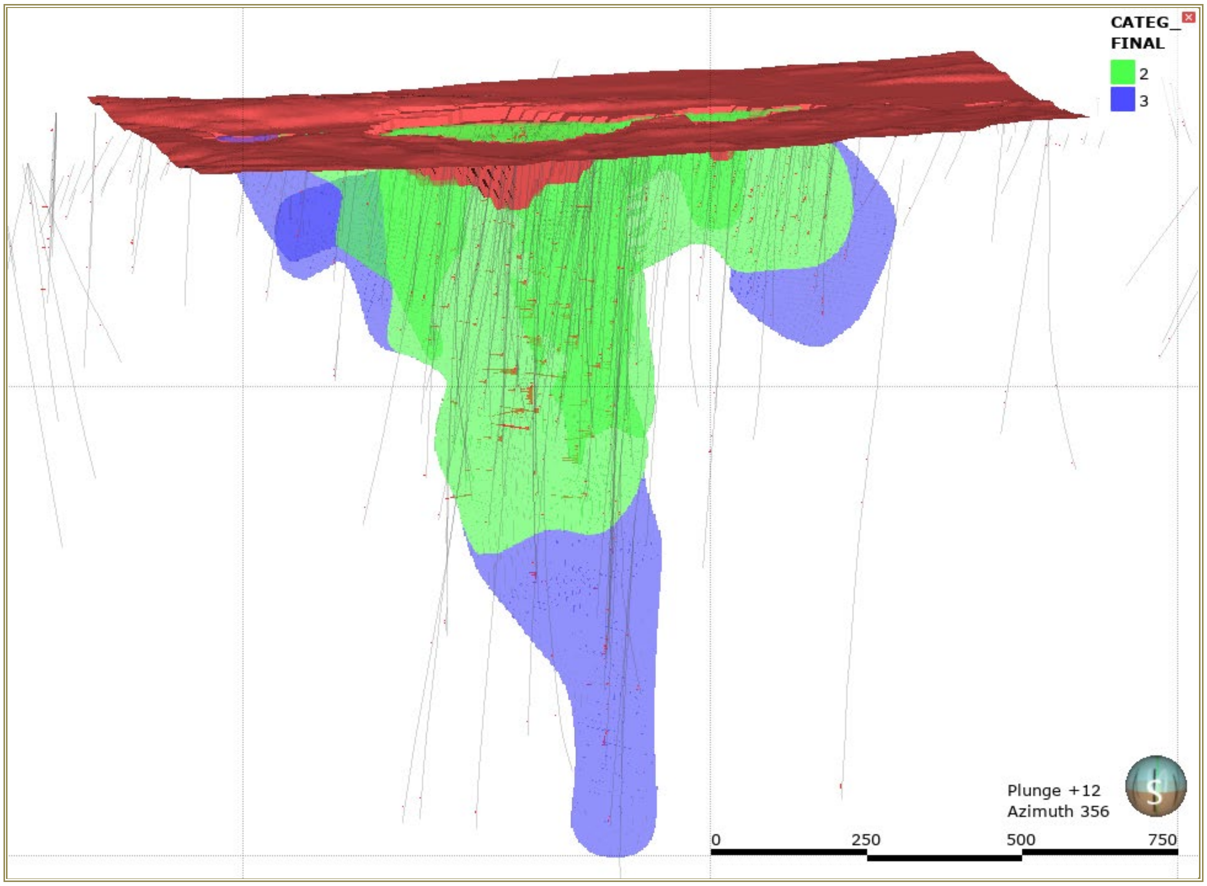


Figure 14-29: Brookbank Mineral Resource Classification

Mineral Resource Statement

The Mineral Resources are summarized in Table 14-28 at cut-off grades of 0.6 g/t Au and 2.4 g/t Au 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 14-26.

The estimated Mineral Resources conform to the *2014 CIM Definition Standards for Mineral Resources & Mineral Reserves*, as required by *NI 43-101—Standards of Disclosure from Mineral Projects*.

Table 14-28: Summary of the Brookbank Mineral Resource

Deposit	Cut-Off Category	Mineral Resource Category	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)
Brookbank Property	Open Pit (0.6 g/t Au)	Measured (M)	-	-	-
		Indicated (I)	1,147	2.24	83
		Subtotal M & I	**1,147**	**2.24**	**83**
		Inferred	45	2.07	3
	Underground (2.0 g/t Au)	Measured (M)	-	-	-
		Indicated (I)	2,281	7.06	517
		Subtotal M & I	**2,281**	**7.06**	**517**
		Inferred	706	3.38	77

Notes:

- Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability.
- There are no Mineral Reserves at Brookbank.
- The independent and qualified person for the Brookbank MRE is Mr. Réjean Sirois, P.Eng. of GMS.
- The effective date of the Mineral Resource was adjusted to June 30, 2024.
- Open pit Mineral Resources are constrained within an optimized pit shell using a gold price of $1,500, a USD/CAD exchange rate of 1.3 and a metallurgical recovery of 92%. An incremental ore haulage cost of $13.77/t milled is also assumed for Brookbank.
- 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 $50/t.
- 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.

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 14-29 and Table 14-30.

Table 14-29: Brookbank Open-Pit Mineral Resource Sensitivity

Cut-Off Grade (g/t Au)	Indicated			Inferred		
	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)
2.0	481	3.81	59	20	3.67	2
1.5	611	3.37	66	20	3.63	2
1.0	814	2.83	74	27	3.01	3
0.8	936	2.59	78	32	2.64	3
0.6	**1,147**	**2.24**	**83**	**45**	**2.07**	**3**
0.4	1,606	1.74	90	76	1.43	3

Table 14-30: Brookbank Underground Mineral Resource Sensitivity

Cut-Off Grade (g/t Au)	Indicated			Inferred		
	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)
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

Qualified Person's Commentary

When compared to the 2020 MRE by Mr. James Purchase of GMS at that time, the 2024 MRE reported above by Mr. Réjean Sirois of GMS for the Brookbank deposit is identical. Gold price used for the Mineral Resource evaluation remains the same as 2020, and since no new drilling, nor interpretations nor estimation were produced, MRE remains unchanged. Only the QP was changed and agreed to stand behind the current MREs.

14.4.2 Key Lake

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 m, of which 23,112 m were 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 Section 12.2.

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.

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 a 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 subvertical, 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 14-30). The zones are generally wide; therefore, no minimum mining thickness was considered during modelling (Figure 14-31). An overburden surface was also modelled from the lithology logging and varies between 10 and 20 m thick above the deposit.

The Dome-era drilling was used in the interpretation, but excluded from the resource 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.

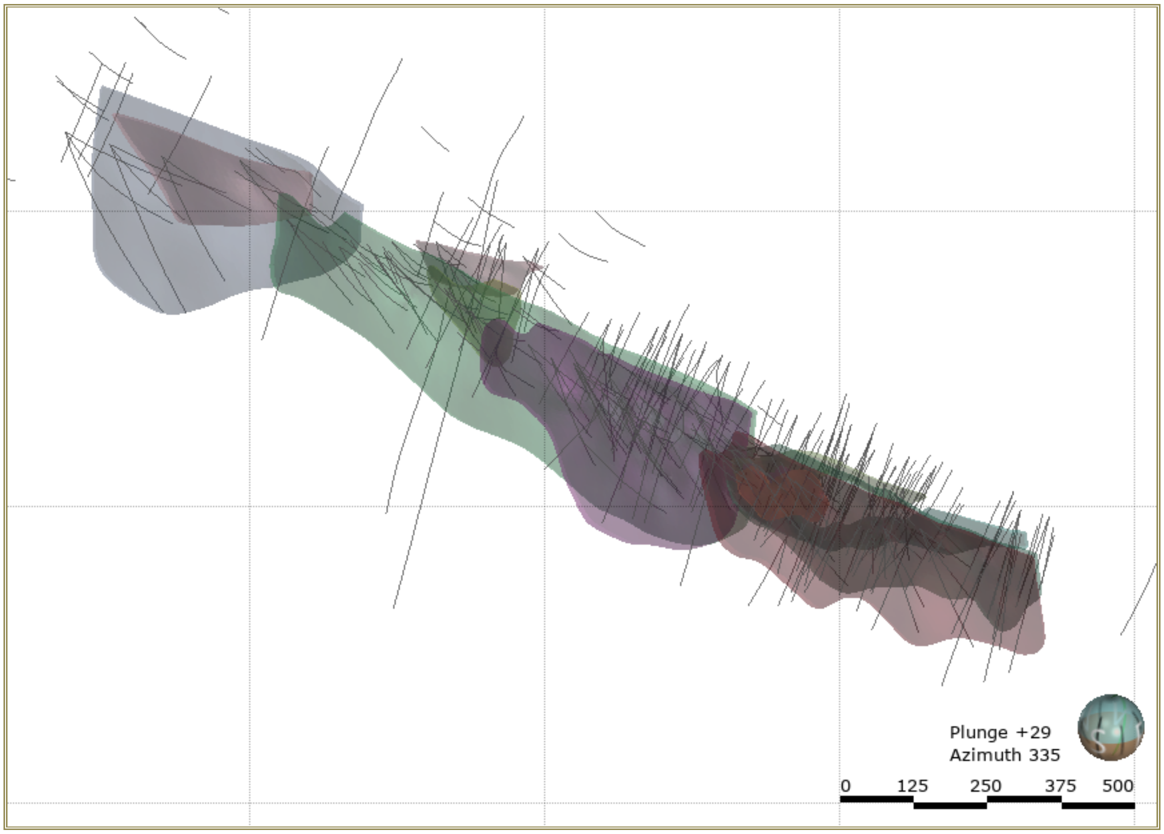


Figure 14-30: Modelled Solids of the Key Lake Deposit, Looking NW

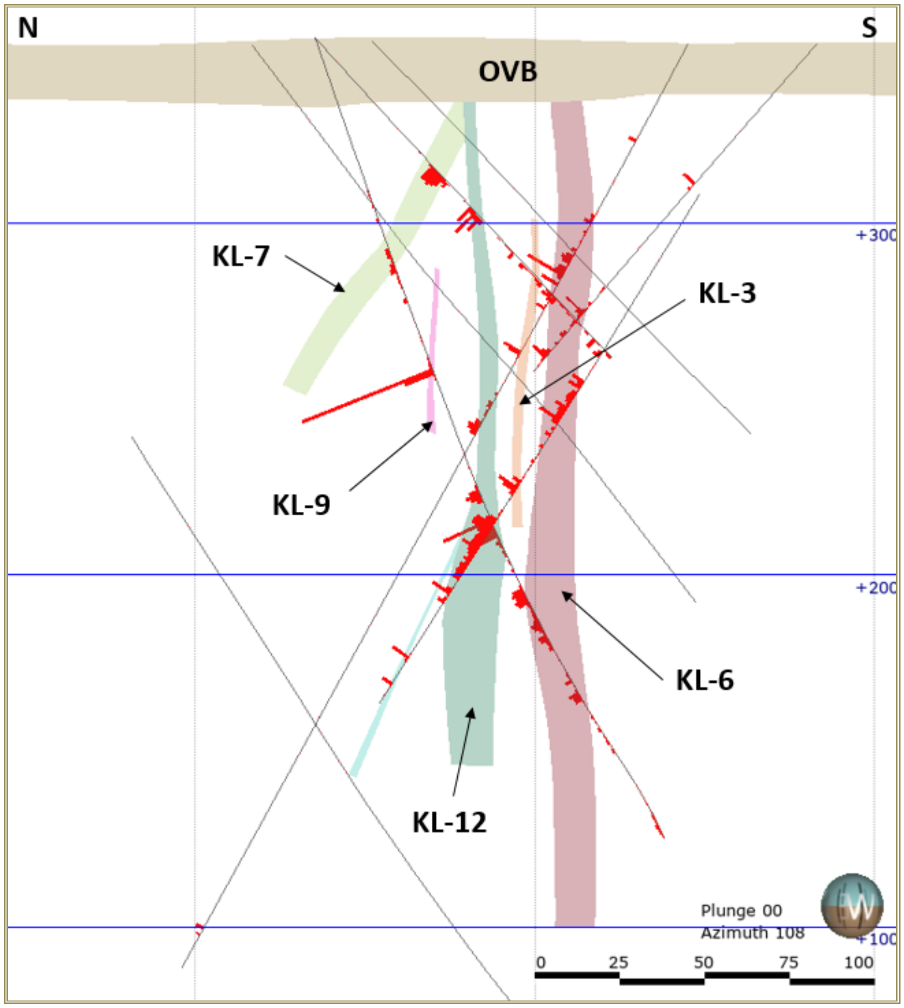


Figure 14-31: Typical Section (Looking Southeast) Showing the Mineralization Wireframes and Overburden (Bar Charts on Drilling Traces show Gold Grades)

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 14-31. An example of the probability plot for the KL-5 domain is shown in Figure 14-32.

Table 14-31: Length-Weighted Assays Statistics Showing Grade Capping Levels and Metal Loss Factors

Domain	No. of Assays	Max. (g/t Au)	Uncut Mean (g/t Au)	High-Grade Capping (g/t Au)	Cut Mean (g/t Au)	No. 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: % metal loss factors calculated from length multiplied by grade and does not consider the spatial location of the outliers.

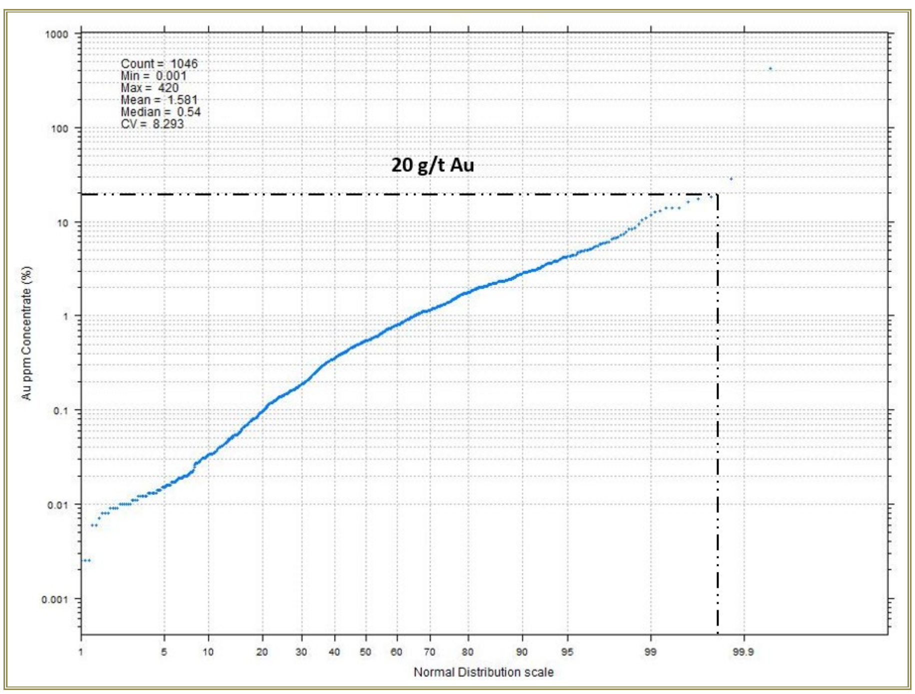


Figure 14-32: Example of a Probability Plot for the KL-5 Domain

Core sampling was undertaken at 1.0 and 1.5 m intervals, and samples 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/t Au).

Length-weighted composite statistics of drilling used in the estimation is shown in Table 14-32. The coefficient of variations are generally low for this style of deposit (less than 2.0)

Table 14-32: Length-Weighted 2 m Composite Statistics of Capped Gold Grades by Domain

Domain	No. of 2 m Composites	Min. (g/t Au)	Max. (g/t Au)	Mean (g/t Au)	Median (g/t Au)	SD	Variance	CV
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

Notes: CV = coefficient of variation; SD = standard deviation.

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% to 50% of the total sill. The major axis was interpreted to be aligned along strike dipping shallowly to the WNW (dip = 20°, 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 on the order of 60 to 70 m, with semi-major ranges slightly less, on the order of 50 to 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, ID^3 will be used as an interpolator, and the results of the variography will be used to guide the dimensions of the search ellipses.

Block Modelling

The block model definition is presented in Table 14-33. 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 14-33 shows a plan view of the Key Lake block model location.

Table 14-33: Key Lake Deposit Block Model Attributes

Item	X	Y	Z
Origin Coordinates (m)	490,150.00	5,506,500.00	600
Block Extents (m)	2,800	915	800
Number of Parent Blocks	280	183	160
Parent Block Size (m)	10	5	5
Sub-Block Size (m)	2.5	1.25	1.25
Rotation		17° clockwise	

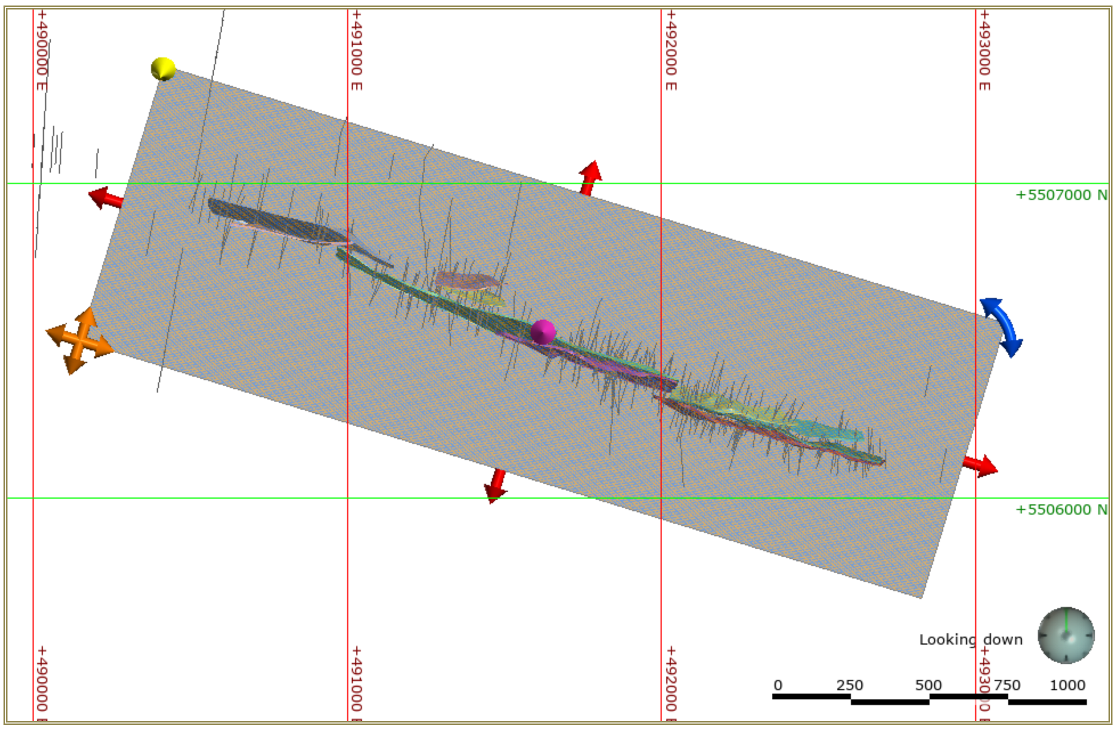


Figure 14-33: Plan View of Key Lake Block Model, Wireframes, and Drill Traces

Bulk Density Data

GGM supplied bulk density data in the form of a Microsoft 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 14-34 presents the bulk-density data available for the Key Lake deposit.

Table 14-34: Statistical Summary of Bulk-Density Data for the Key Lake Deposit

Deposit	Lithology	No. of Measurements	Mean Density (g/cc)	SD 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

Note: SD = standard deviation.

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.

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 midpoints 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, between 7 and 16 samples, and a maximum of 3 samples per drill hole was applied. This ensured that a minimum of three drill holes was required to estimate blocks in the first pass.

For Pass 2, between 4 and 16 samples, and a maximum of 3 samples per drill hole was applied. This ensured that a minimum of two drill holes was required to estimate blocks in the second pass.

For Pass 3, between 1 and 16 samples, and a maximum of 3 samples per drill hole was applied. This ensured that a minimum of one drill hole was 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 14-35.

Table 14-35: Summary of Search Parameters—Key Lake Deposit

Domain	Pass	X (m)	Y (m)	Z (m)	Min. Samples	Max. Samples	Max. Samples/DH	High-Grade Restraining			
								X (m)	Y (m)	Z (m)	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: DH = drill hole.

Grade Interpolation

ID^3 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 14-34.

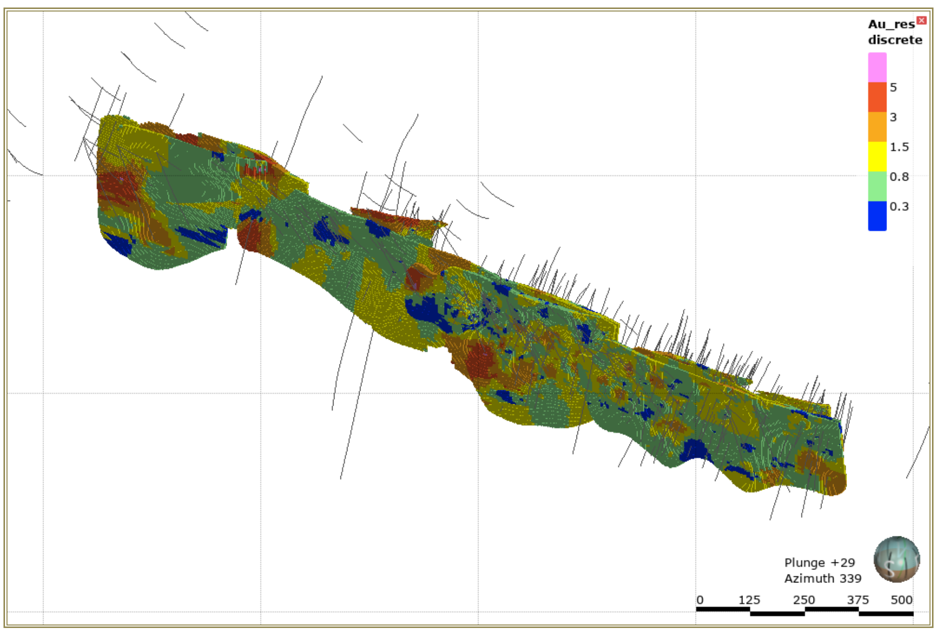


Figure 14-34: Key Lake Gold Grade Distribution in Resource Block Model, Looking NW

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 14-36), and local validations such as swath plots were used to ensure grade smearing was reduced to a minimum (Figure 14-35).

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.

Table 14-36: Global Statistical Comparison between Blocks and Declustered Composites for all Estimation Passes at Key Lake

Domain	Composites			Blocks		Difference Mean (%)
	No. of Comps.	Mean (g/t)	Declustered Mean (g/t)	No. Blocks	Mean (g/t)	
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	2,060	1.08	1.10	1,259,970	1.11	1

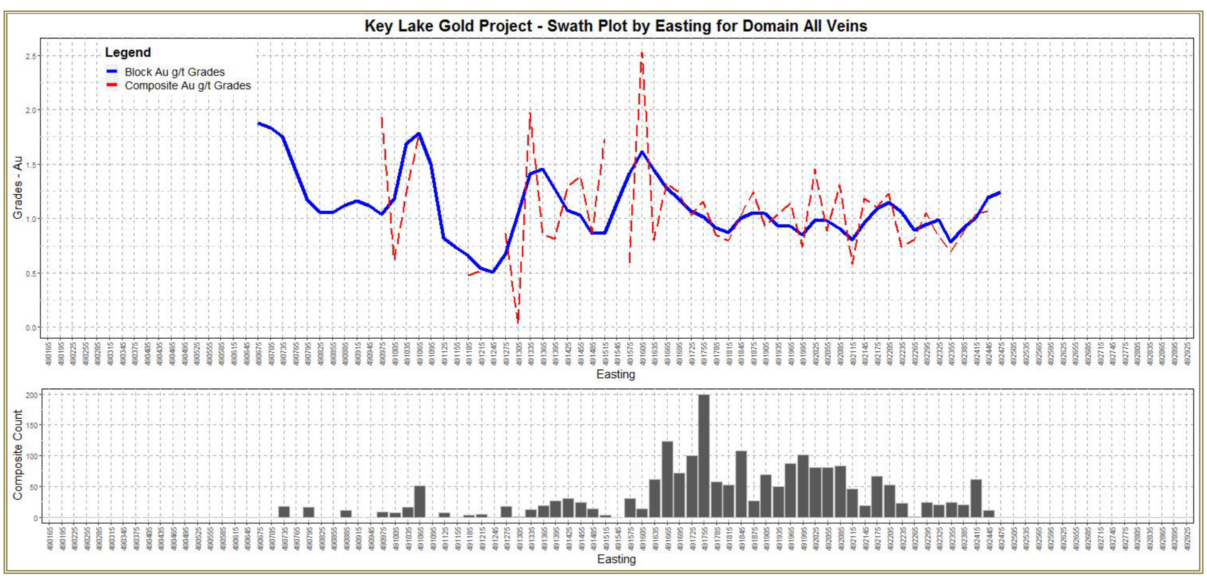


Figure 14-35: Swath Plot Comparing Block Gold Grades (Blue) with Capped Composite Gold Grades (Red Dotted) for the 12 Domains Grouped Together, by Easting

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 14-26, and the chosen pit is shown in Figure 14-36.

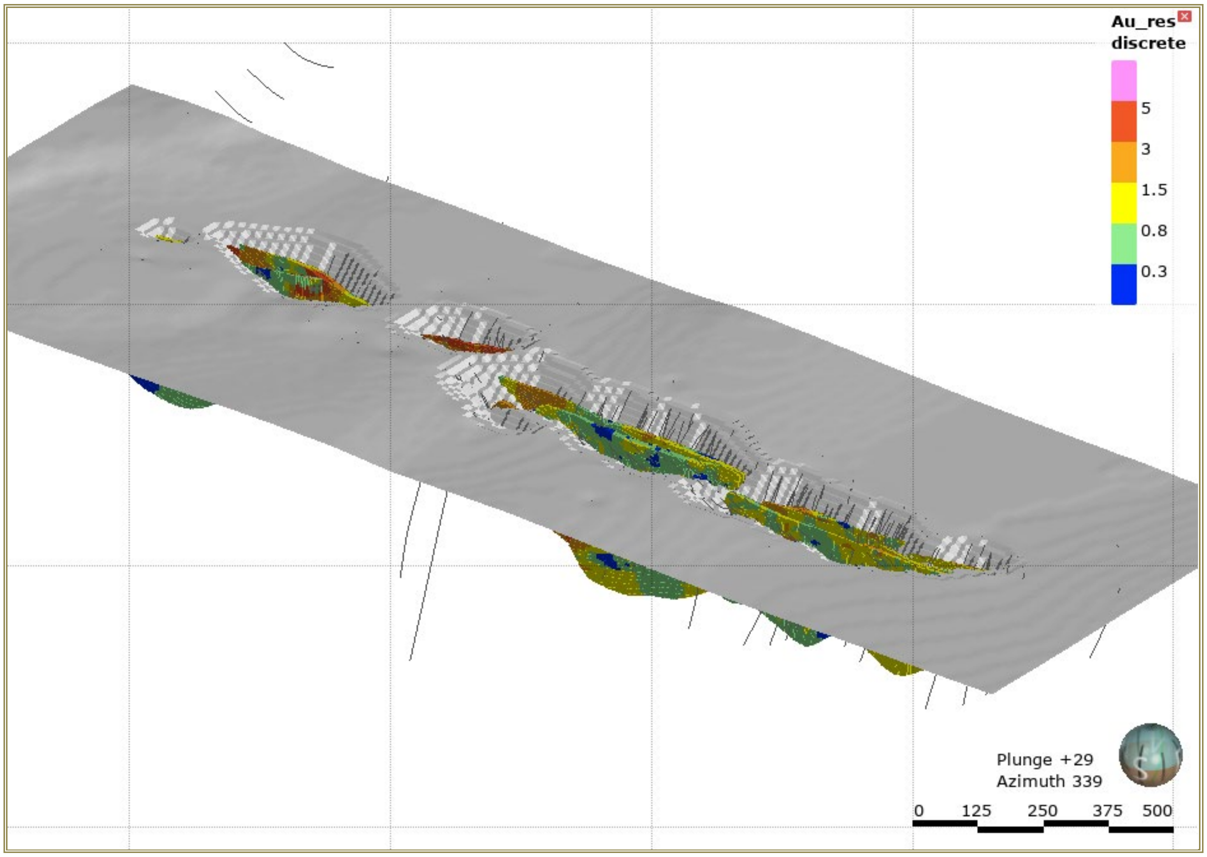


Figure 14-36: Key Lake Deposit Pit Optimization—$1,500 Pit Shell, MII Blocks

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 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.

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 14-37. 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 due to a wider drill spacing, and all blocks outside of the 12 modelled domains and below 0 RL elevation were assigned to the Inferred category.

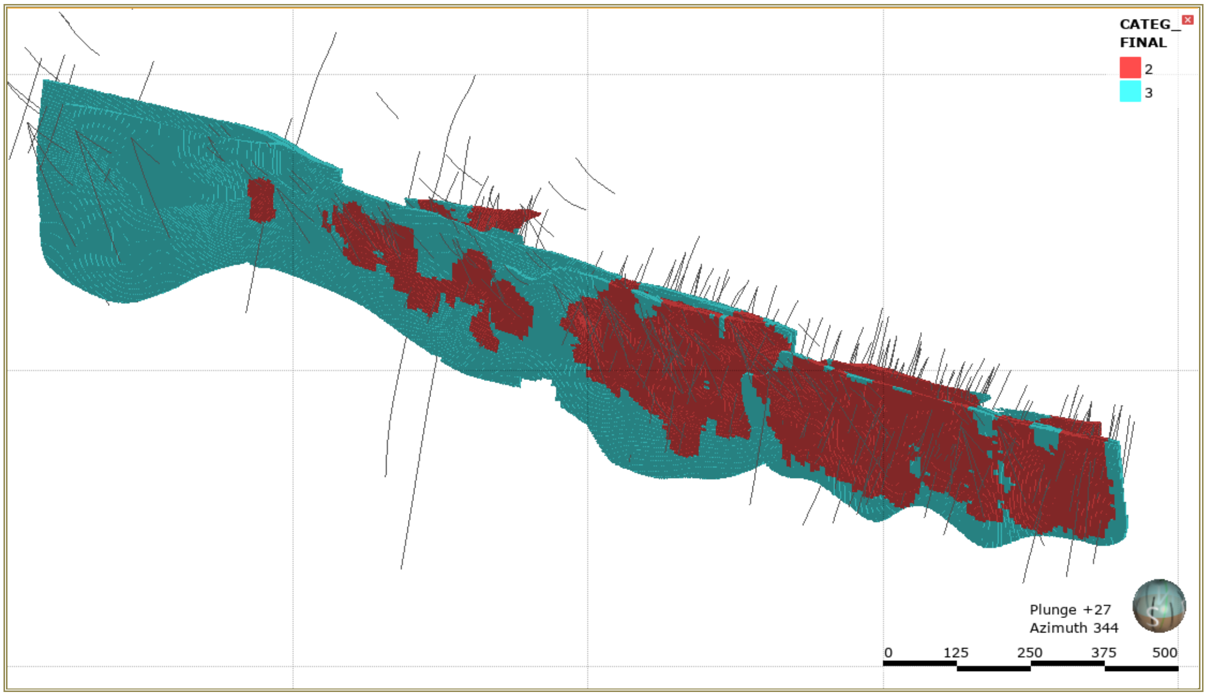


Figure 14-37: Key Lake Deposit Coloured by Resource Category (Red = Indicated, Light Blue = Inferred)

Mineral Resource Statement

The Mineral Resources are summarized in Table 14-37 at a lower cut-off grade of 0.4 g/t Au 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 14-26.

The estimated Mineral Resources conform to the *2019 CIM Definition Standards for Mineral Resources & Mineral Reserves*, as required by NI 43-101—*Standards for Disclosure for Mineral Projects*.

Table 14-37: Summary of the Key Lake Mineral Resource

Deposit	Cut-Off Category	Mineral Resource Category	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)
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:

- Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability.
- No Mineral Reserves are published at Key Lake.
- The independent and qualified person for the Key Lake MRE is Mr. Réjean Sirois, P.Eng. of GMS.
- The effective date of the Mineral Resource was adjusted to June 30, 2024.
- Open pit Mineral Resources are constrained within an optimized pit shell using a gold price of $1,500, a CAD/USD exchange rate of 1.3 and a metallurgical recovery of 90%. An incremental ore haulage cost of $3.47/t milled is also assumed for Key Lake.
- No underground Mineral Resources are quoted.
- 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.

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 14-38.

Table 14-38: Key Lake Open-Pit Mineral Resource Sensitivity

Cut-Off Grade (g/t Au)	Indicated			Inferred		
	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Tonnes (kt)	Gold Grade (g/t)	Contained Gold (koz)
1.5	855	2.18	60	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

Qualified Person's Commentary

When compared to the 2020 MRE by Mr. James Purchase of GMS at that time, the 2024 MRE reported above by Mr. Réjean Sirois of GMS for the Key Lake deposit is identical. Gold price used for the Mineral Resource evaluation remains the same as 2020, and since no new drilling, interpretations, or estimation were produced the MRE remains unchanged. Only the QP was changed and agreed to stand behind the current MRE.

14.4.3 Kailey Deposit

The Kailey deposit is located 3 km northwest of the Greenstone Mine and was originally an underground mine named the Little Long Lac Mine.

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 were assayed.

The drill holes are on an irregular grid spacing of around 50 to 70 m, and at depth the spacing becomes 80 to 100 m. Historical drilling data exists from digitized hard copies; however, no significant assays are available, and all the historical data are unverifiable.

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.

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 are 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 observe in drill core.

The Kailey deposit has been modelled as three, wide (20–30 m), continuous zones named Kailey, Main, and No. 9 (Figure 14-38). The zones are generally wide; therefore, no mining thickness was considered during modelling (Figure 14-39). An overburden surface was modelled from the lithology logging and varies between 5 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.

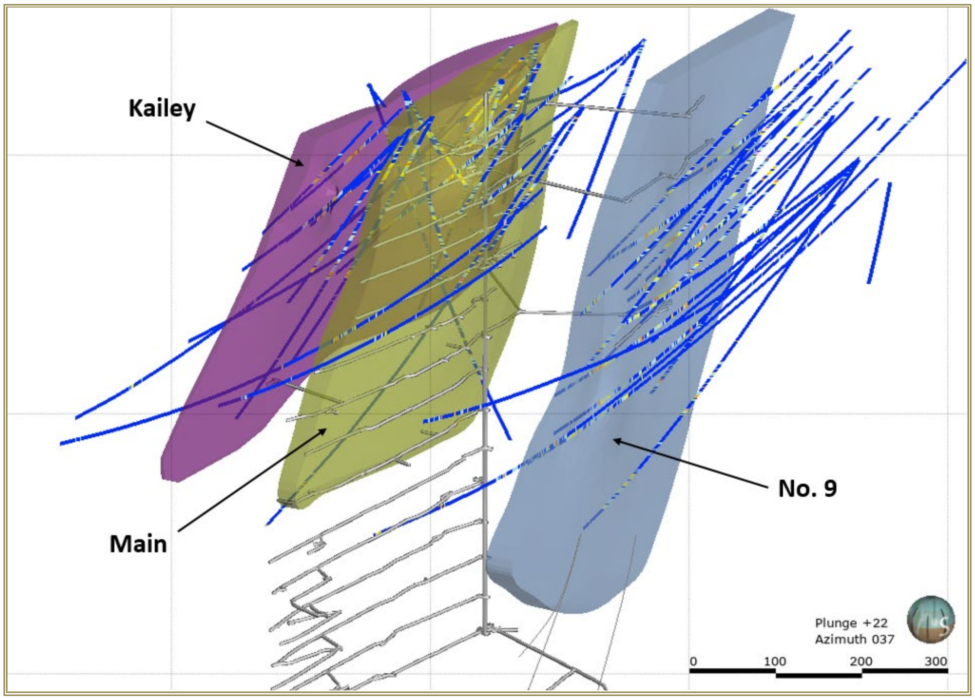


Figure 14-38: Isometric View Looking NW of the Three Domains at the Kailey Deposit with Underground Workings

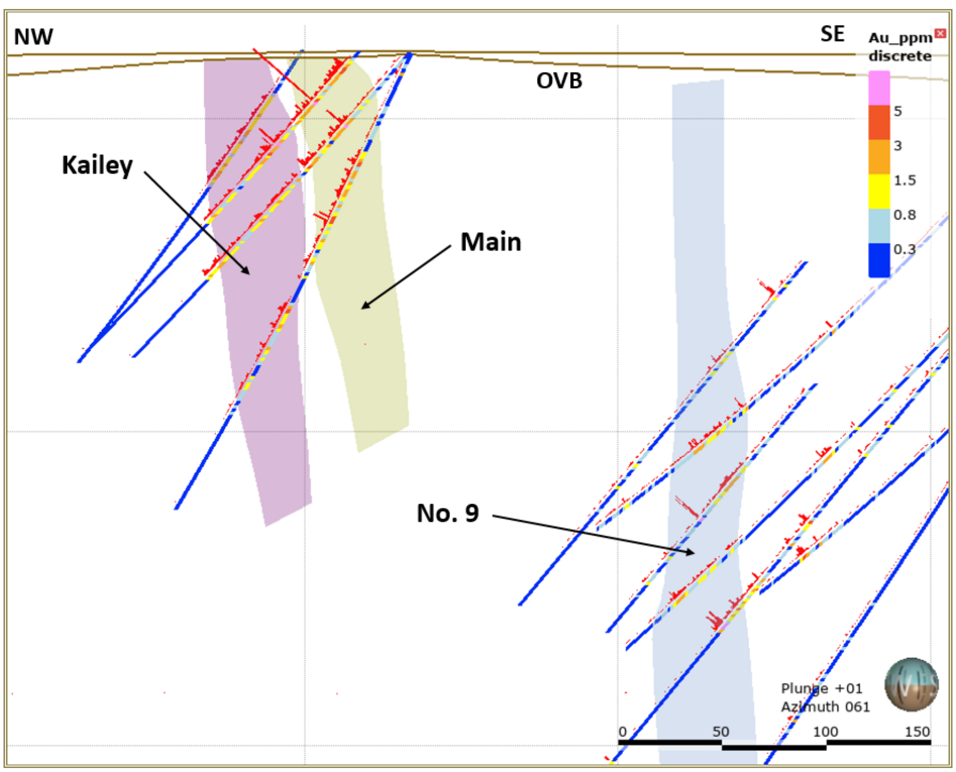


Figure 14-39: Typical Section showing Kailey and Main Domains Near Surface, and No. 9 Domain at Depth (Drill Holes Coloured by Gold)

Assay Capping and Compositing

Grade capping levels were determined using probability plots of the various gold 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 14-39. An example of the probability plot for the KL-5 domain is shown in Figure 14-40.

Table 14-39: Length-Weighted Assays Statistics Showing Grade Capping Levels and Metal Loss Factors

Domain	No. of Assays	Max. (g/t Au)	Uncut Mean (g/t Au)	High-Grade Capping (g/t Au)	Cut Mean (g/t Au)	No. 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**

Note: % metal loss factors calculated from length multiplied by grade and does not consider the spatial location of the outliers.

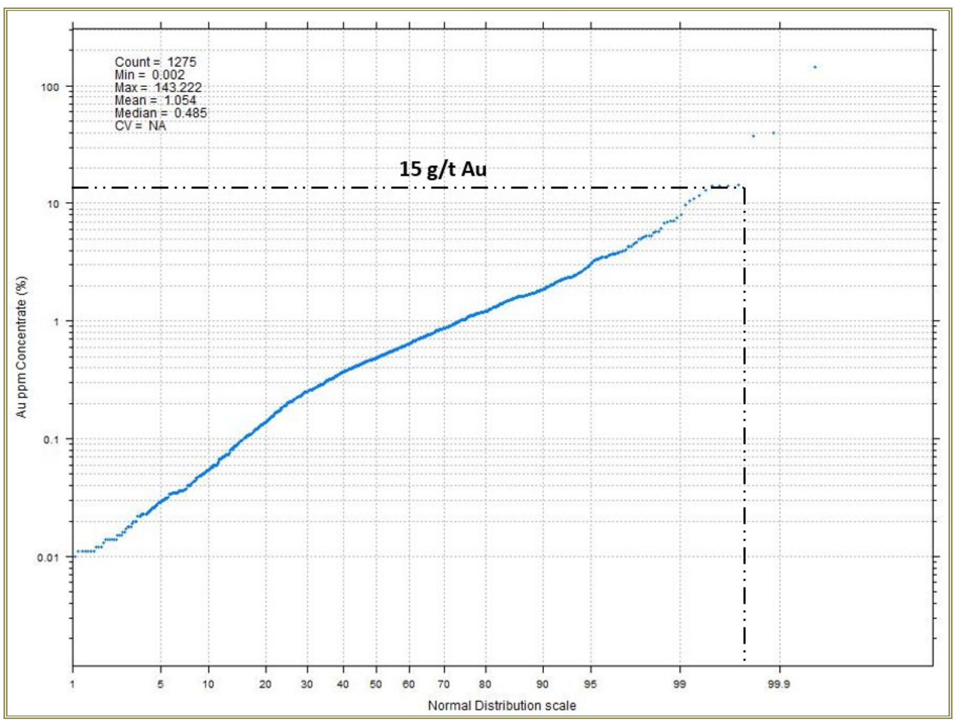


Figure 14-40: Example of a Probability Plot for the Main Domain

Core sampling was undertaken at 1.0 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/t Au). Breakthroughs into underground voids were omitted from the compositing.

Length-weighted composite statistics of drilling used in the estimation is shown in Table 14-40. The coefficients of variations are generally low for this style of deposit.

Table 14-40: Length-Weighted 2 m Composite Statistics of Capped Gold Grades by Domain

Domain	No. of 2 m Comps.	Min. (g/t Au)	Max. (g/t Au)	Mean (g/t Au)	Median (g/t Au)	SD	Variance	CV
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	9,702	0.001	7.78	0.12	0.02	0.35	0.12	2.82

Notes: CV = coefficient or variation; SD = standard deviation.

Variography

Experimental variograms were constructed for the Kailey, Main, and No. 9 domains using the capped gold composite intervals. Nugget sills were estimated from downhole variograms at a 2 m lag spacing. Pairwise 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 14-41, and an example is shown in Figure 14-41.

Table 14-41: Variogram Parameters for the Kailey Deposit

Zone	Axis	Nugget (C0)	Sill 1 (C1)	Range 1 (R1) (m)	Sill 2 (C2)	Range 2 (R2) (m)	Dip	Dip Azimuth	Pitch
Kailey	Major	0.15	0.25	20	0.2	80	85	170	90
	Semi-major			20		80			
	Minor			5		10			
Main	Major	0.25	0.17	30	0.2	90	90	162	117
	Semi-major			25		70			
	Minor			5		10			
No. 9	Major	0.20	0.30	30	0.25	90	90	155	128
	Semi-major			30		90			
	Minor			5		10			

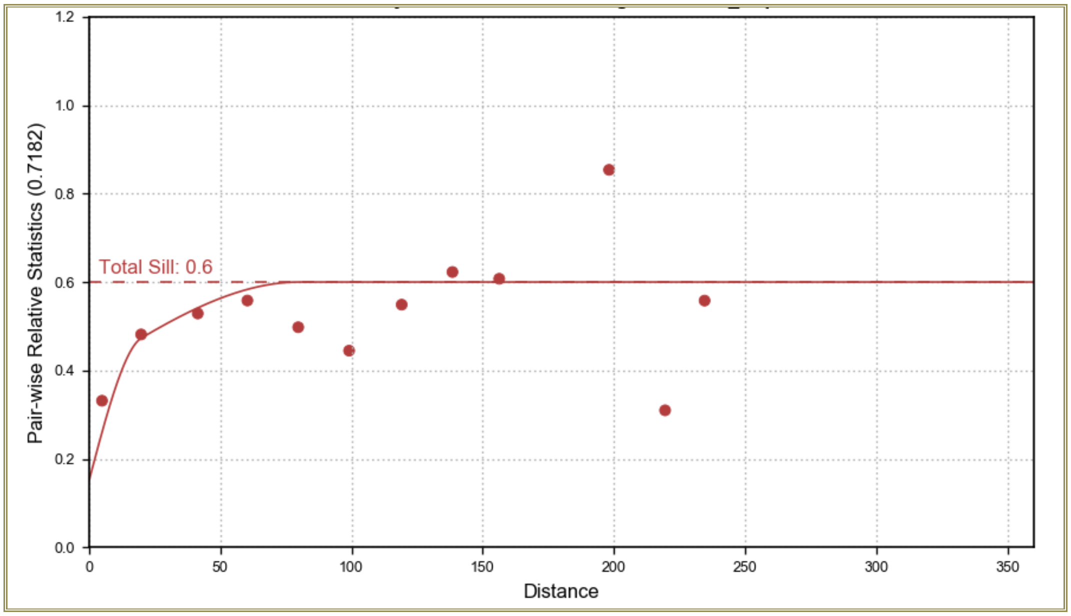


Figure 14-41: Example Variogram for the Major Axis of the Kailey Domains

Block Modelling

The block model definition is presented in Table 14-42. The upper limit was defined by the surface topography. The parent block size was based primarily on drill-hole spacing, envisaged 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 14-42 shows a plan view of the Kailey block model location.

Table 14-42: Kailey Deposit Block Model Attributes

Item	X	Y	Z
Origin Coordinates (m)	502,675.00	5,504,450.00	385
Block Extents (m)	1,200	890	690
Number of Parent Blocks	120	178	138
Parent Block Size (m)	10	5	5
Sub-Block Size (m)	2.5	1	1.25
Rotation		18° counter-clockwise	

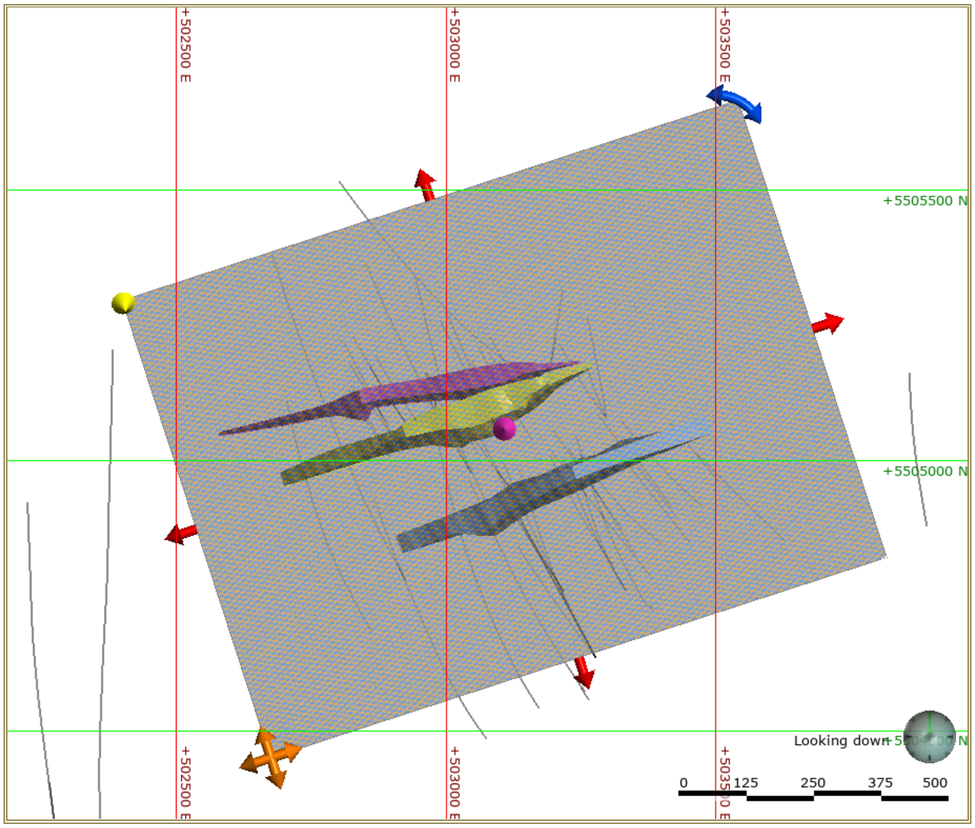


Figure 14-42: Plan View of Kailey Block Model, Wireframes, and Drill Traces

Bulk-Density Data

Bulk-density data were supplied by GGM in the form of a Microsoft 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 14-43 presents the bulk density data available for the Kailey deposit.

Table 14-43: Statistical Summary of Bulk-Density Data for the Key Lake Deposit

Deposit	Lithology	No. of Meas.	Mean Density (g/cc)	SD 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

Note: SD = standard deviation.

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.

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 midpoints 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, between 7 and 16 samples, and a maximum of 3 samples per drill hole was applied. This ensured that a minimum of 3 drill holes was required to estimate blocks in the first pass.

For Pass 2, between 4 and 16 samples, and a maximum of 3 samples per drill hole was applied. This ensured that a minimum of 2 drill holes was required to estimate blocks in the second pass.

For Pass 3, between 3 and 16 samples, and a maximum of 3 samples per drill hole was applied. This ensured that a minimum of 1 drill holes was required to estimate blocks in the third pass.

For Pass 4, between 1 and 16 samples, and a maximum of 3 samples per drill hole 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 14-44.

Table 14-44: Summary of Search Parameters—Kailey Deposit

Domain	Pass	X (m)	Y (m)	Z (m)	Min. Samples	Max. Samples	Max. Samples/DH	High-Grade Restraining			
								X (m)	Y (m	Z (m)	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: DH = drill hole.

Grade Interpolation

OK was the preferred estimator for the Kailey, No. 9, and Main domains. Variograms showed good structure and were readily interpretable. All blocks outside of these three domains were estimated using ID^2. Estimated blocks are shown in Figure 14-43.

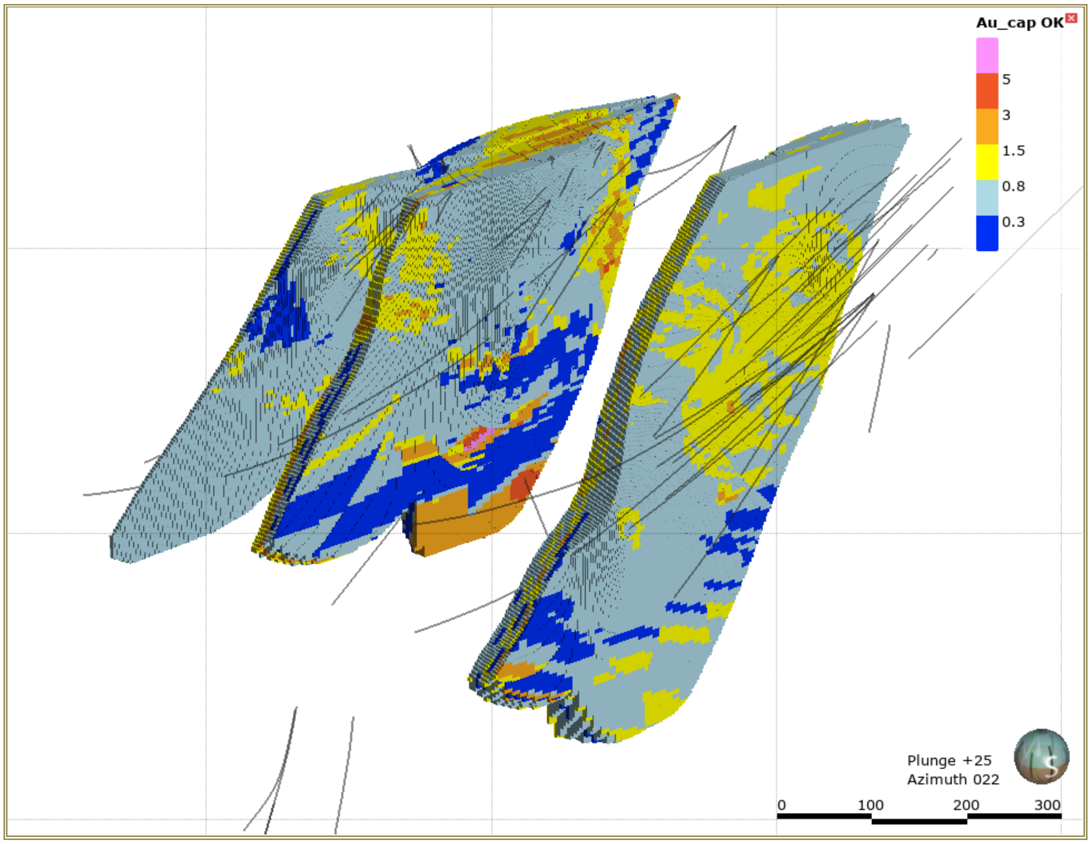


Figure 14-43: Kailey Gold Grade Distribution in Resource Block Model, Looking NW

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 14-26, and the chosen pit is shown in Figure 14-44.

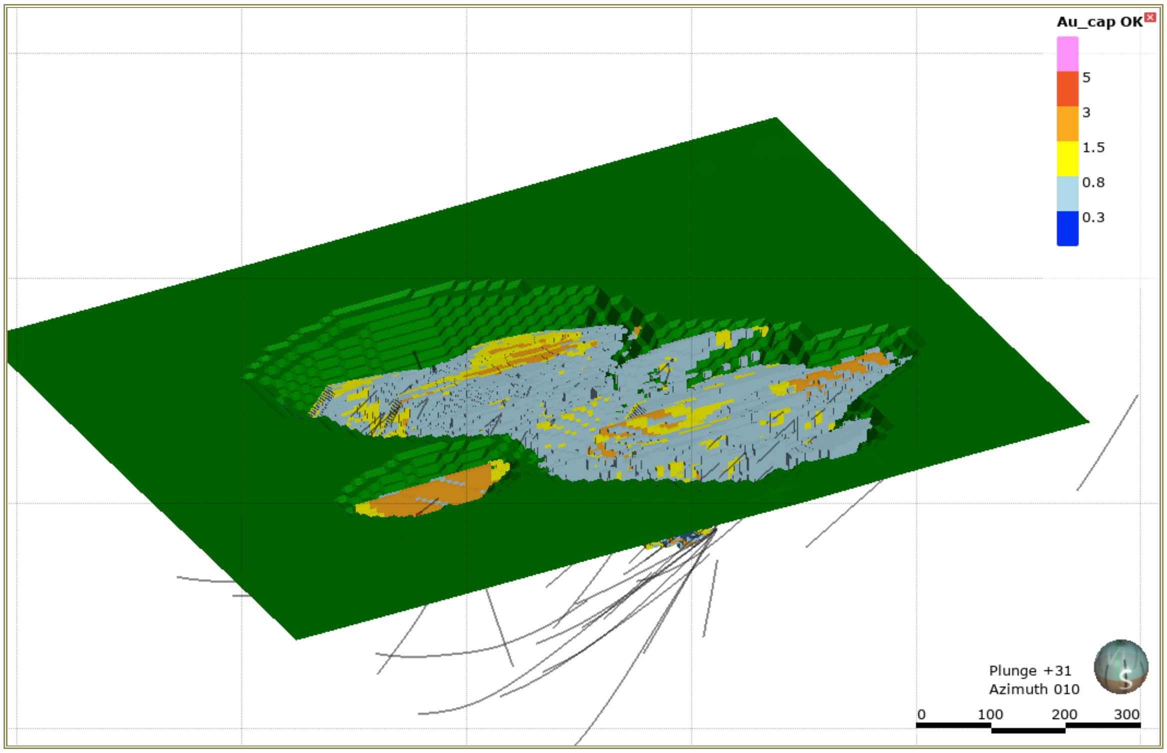


Figure 14-44: Key Lake Deposit Pit Optimization—$1,500 Pit Shell, MII Blocks

Underground Voids

There has been significant past production at the Kailey deposit, formerly known as the Little Long Lac Mine, which produced 1.78 Mt at an average grade of 10.6 g/t Au for 607 koz over an 18-year mine life. GMS was supplied an underground void model for the shaft, underground drifts, and a 3-D 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; the mine operated between 1934 and 1952. Figure 14-45 shows the underground workings in relation to the three modelled domains.

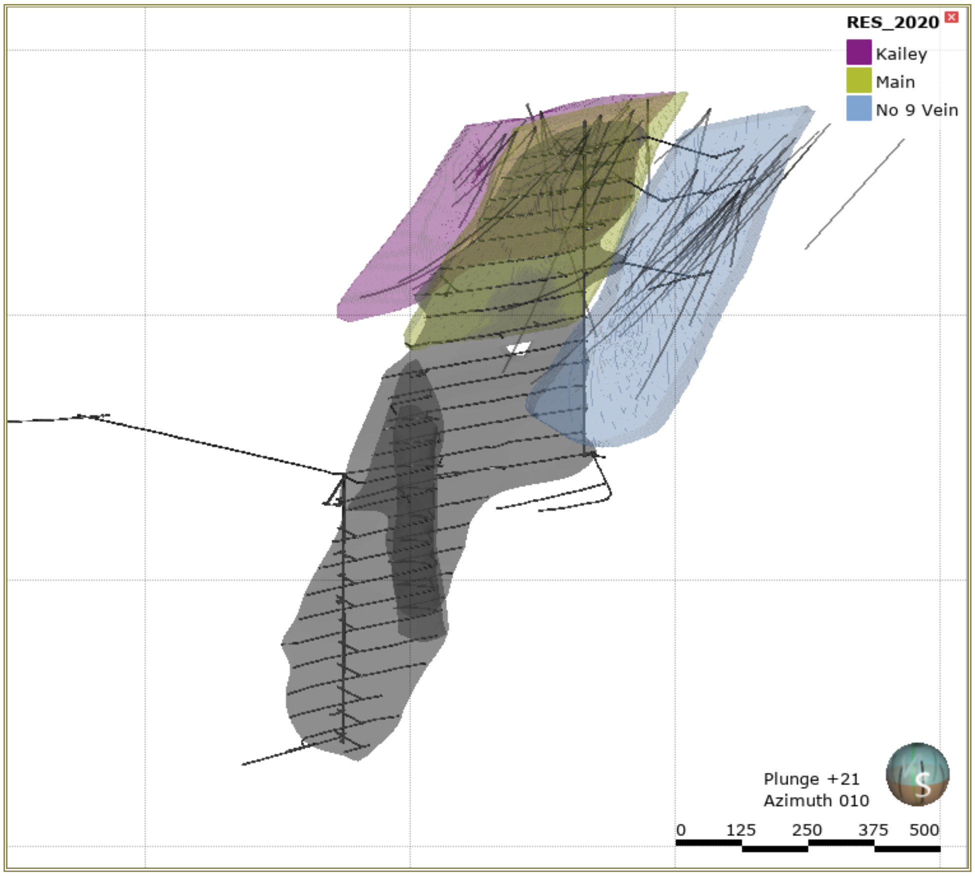


Figure 14-45: Underground Void Model (Grey) and Three Modelled Domains at Kailey

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 14-46. 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.

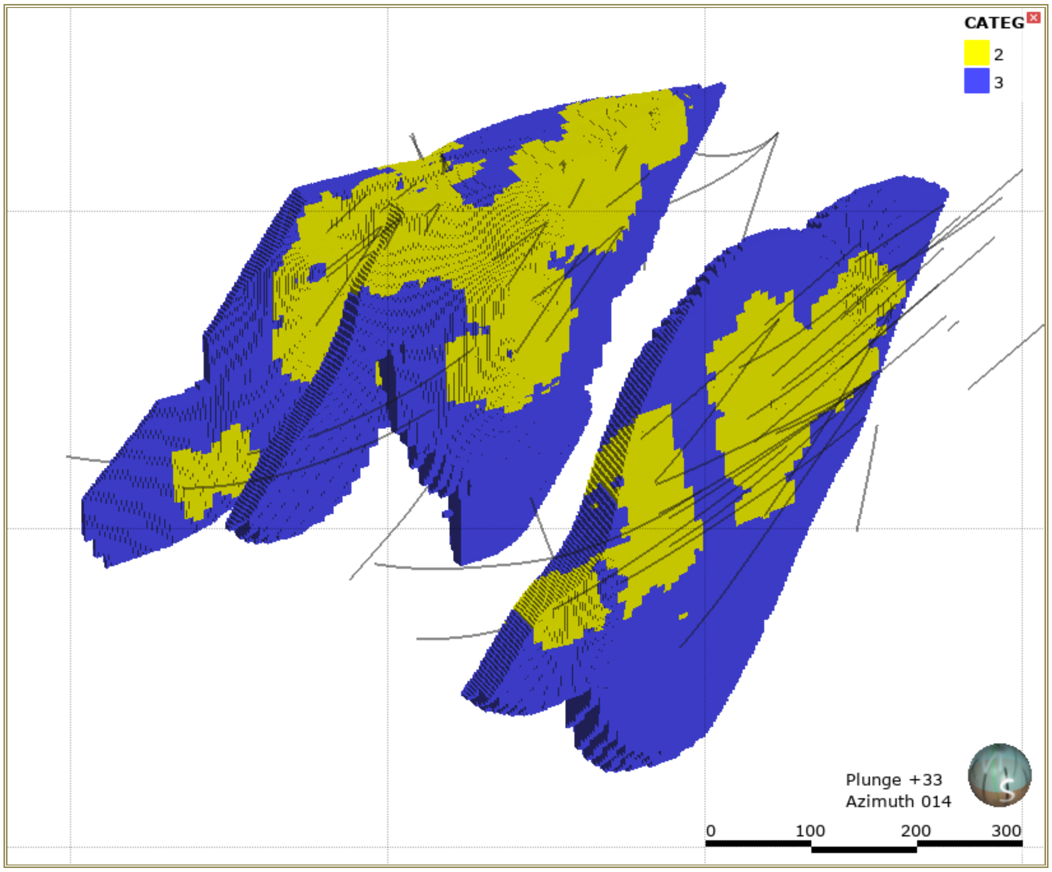


Figure 14-46: Kailey Deposit Coloured by Resource Category (Yellow = Indicated, Blue = Inferred)

Mineral Resource Statement

The Mineral Resources are summarized in Table 14-45 at a lower cut-off grade of 0.4 g/t Au 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 14-26.

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 14-45: Summary of Kailey Open Pit Mineral Resource

Mineral Resource Category	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)
Measured (M)	-	-	-
Indicated (I)	11,276	0.96	348
Subtotal M & I	**11,276**	**0.96**	**348**
Inferred	4,858	0.87	136

Notes:

- Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability.
- No Mineral Reserves are published at Kailey.
- The independent and qualified person for the Kailey MRE is Mr. Réjean Sirois, P.Eng., of GMS.
- The effective date of the Mineral Resource was adjusted to June 30, 2024.
- Open pit Mineral Resources are constrained within an optimized pit shell using a gold price of $1,500, a CAD/USD exchange rate of 1.3 and a metallurgical recovery of 90%. An incremental ore haulage cost of $1.30/t milled is also assumed for Kailey.
- No underground Mineral Resources are quoted.
- 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.

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 14-46.

Table 14-46: Kailey Open-Pit Mineral Resource Sensitivity

Cut-Off Grade (g/t Au)	Indicated Category			Inferred Category		
	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)
1.5	1,395	1.89	85	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

Qualified Person's Commentary

When compared to the 2020 MRE by Mr. James Purchase of GMS at that time, the 2024 MRE reported above by Mr. Réjean Sirois of GMS for the Kailey deposit is identical. Gold price used for the Mineral Resource evaluation remains the same as 2020, and since no new drilling, or interpretations or estimations were produced since that time MRE remains unchanged. Only the QP was changed and agreed to stand behind the current MRE.

15 MINERAL RESERVE ESTIMATES

The Mineral Reserve for the open pit mine at GGM is estimated at 144.7 Mt at an average grade of 1.23 g/t Au for 5.7 Moz of contained gold as summarized in Table 15-1. The MRE was prepared by GMS. The resource block model was also generated by GMS with an effective date of June 30, 2024.

The MRE and mine design have been completed to a level appropriate to declare Mineral Reserves. The Mineral Reserves are based on Measured and Indicated Mineral Resources, and do not include any Inferred Mineral Resources. The Inferred Mineral Resources contained within the mine design are classified as waste.

Table 15-1: Greenstone Open Pit Mineral Reserve Statement

Category	Diluted Ore Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)
Proven	6,817	1.16	255
Probable	137,846	1.23	5,445
Total P&P	**144,662**	**1.23**	**5,700**

Notes:

- CIM Definition Standards were followed for Mineral Reserves.
- Effective date of the estimate is June 30, 2024.
- Mineral Reserves are estimated at a cut-off grade of 0.30 g/t Au.
- Mineral Reserves are estimated using a long-term gold price of $1,550/oz and a CAD/USD exchange rate of 1.28:1.00.
- A minimum mining width of 15 m was used.
- Bulk density of ore is variable but averages 2.78 t/m^3.
- The average strip ratio is 5.5:1.
- Dilution factor is 17.2%.
- Numbers may not add due to rounding.

15.1 Mineral Resource Block Model

The Mineral Resource block model consists of blocks with percent attributes for intact rock mass, overburden, organics, historical 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 historical tailings overlay the pit footprint and have been modelled to allow for their tracking and management in the material movement plan.

15.2 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 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 Technical Report, Measured and Indicated Mineral Resource blocks were considered for optimization purposes and for mineable Mineral Resource calculations. However, sensitivities were run using the complete MRE.

15.2.1 Mining Dilution and Ore Loss

A mining dilution assessment was previously performed by evaluating the number of contacts for blocks above an economic cut-off grade. The block contacts were used to create a dilution skin around ore blocks to estimate an expected dilution value 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.

The resulting dilution percentage was equivalent to 17.2%, and the ore loss at 1.2%; those parameters were directly inputted in Whittle for the current optimization exercise.

15.2.2 Pit Optimization Parameters

A summary of the pit optimization parameters is presented in Table 15-2 for a milling rate of 27 kt/d based on a long-term gold price of $1,550/oz and a CAD/USD exchange rate of 1.28:1.00.

Table 15-2: Greenstone Pit Optimization Parameters

Description	Unit	Value
Nominal Milling Rate	t/d	27,000
Plant Throughput	kt/a	9,855
Exchange Rate	CAD/USD	1.28
Diesel Fuel Price Delivered	$/L	0.938
Gold Price	$/oz	1,550
Gold Price (local currency)	C$/oz	1,984
Transport and Refining Cost	$/oz	2.34
Royalty and Simulated Other Agreements	%	4.7
Metallurgical Recovery at Cut-Off Grade	%	91.1
Total Processing Cost	**$/t milled**	**6.98**
G&A	$/t milled	3.31
Rehabilitation and Closure	$/t milled	0.36
Sustaining Capital	$/t milled	0.45
Others G&A	$/t milled	0.10
Total Ore-Based Cost	**$/t milled**	**11.21**
Cut-Off Grade	g/t Au	0.30
Mining Rate	Mt/a	72.0
Mining Dilution	%	17.2
Mining Loss	%	1.2
Total Mining Reference Cost	**$/t mined**	**1.97**
Incremental Bench Cost	C$/10 m bench	0.03
Overall Slope Angle in Fresh Rock	°	55
Overall Slope Angle in Overburden	°	25

Note: G&A = general and administrative.

The gold selling cost includes a 3% royalty fee plus a transport and refining cost of $2.34/oz. The cost parameters were estimated based on first principle assumptions. The total ore-based cost is estimated at $11.20/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 $1.97/t with an incremental depth factor of $0.03/t per 10 m bench.

A physical hard boundary was imposed in the optimization process to prevent the pit from encroaching into the nearby lake (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.

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° based on a designed inter-ramp angle of 63.4°. The overall slope angle in overburden is 25°.

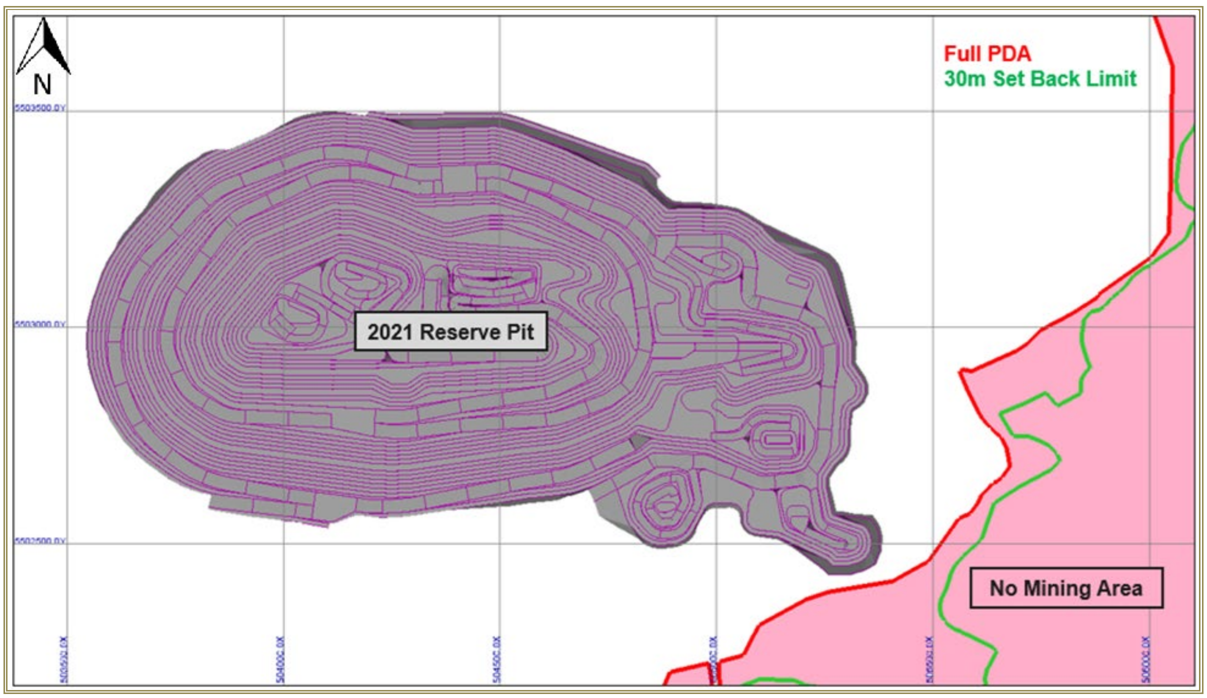


Figure 15-1: Pit Limit Physical Boundary Constraint

15.2.3 Cut-Off Grades

The cut-off grade resulting from the optimization parameters is calculated at 0.30 g/t Au which assumes an average metallurgical recovery of 91.1% and an average mining dilution of 17.2%. The cut-off grade is the break-even grade where revenue equals costs to carry the full operation while excluding direct mining costs as calculated using the following formula:

$$COG(g/t) = \frac{Cp + Ca + Cr + Com + Csibc + Cmc}{r*(P - Cs)}$$

Where:

r is the metallurgical recovery (%)

P is the gold price in $/oz

Cs is the cost of selling gold (refining and royalties) in $/oz

Cp is the total processing costs (fixed and variable) in $/t treated

Ca is General and Administration (G&A) cost in $/t treated

Cr is the cost of rehandle in $/t treated

Com is the difference between ore and waste mining cost in $/t treated

Csibc is non-mining sustaining capital in $/t treated over LOM

Cmc is mine closure cost incurred during the life of mine in $/t treated

15.2.4 Open Pit Optimization Results

Whittle pit optimization software produces nested shells, and the results are presented in Table 15-3 using only the Measured and Indicated Mineral Resources. The nested shells are generated by using revenue factors to scale up and down from the base-case selling price.

Table 15-3: Measured and Indicated Mineral Resource Whittle Shell Results

Shell No.	RF No.	Gold Price ($/oz)	NPV Best ($M)	NPV Spec. ($M)	NPV Worst ($M)	Pit Value ($M)	Ore (Mt)	Waste (Mt)	Total (Mt)	SR t:t	Au (g/t)	Au (koz)
1	0.30	465	922	922	922	1,007	20.5	47.6	68.2	2.3	1.55	1,026
2	0.32	496	1,005	1,003	1,003	1,105	23.3	56.0	79.3	2.4	1.52	1,140
3	0.34	527	1,067	1,063	1,063	1,181	25.5	62.7	88.3	2.5	1.50	1,229
4	0.36	558	1,195	1,184	1,184	1,335	31.2	76.3	107.5	2.4	1.42	1,422
5	0.38	589	1,302	1,283	1,283	1,468	35.8	91.3	127.1	2.5	1.38	1,592
6	0.40	620	1,315	1,294	1,294	1,484	36.5	93.4	129.8	2.6	1.38	1,614
7	0.42	651	1,813	1,758	1,758	2,150	59.9	183.7	243.6	3.1	1.30	2,499
8	0.44	682	1,859	1,800	1,800	2,216	62.2	194.4	256.6	3.1	1.29	2,589
9	0.46	713	1,900	1,834	1,834	2,276	64.7	205.0	269.8	3.2	1.29	2,676
10	0.48	744	2,198	2,101	2,101	2,737	84.9	286.3	371.2	3.4	1.24	3,372
11	0.50	775	2,252	2,148	2,148	2,824	88.0	306.5	394.5	3.5	1.24	3,503
12	0.52	806	2,389	2,252	2,252	3,051	98.5	359.4	457.8	3.6	1.22	3,871
13	0.54	837	2,547	2,375	2,375	3,331	111.6	423.0	534.7	3.8	1.21	4,332
14	0.56	868	2,655	2,437	2,435	3,536	121.8	479.2	600.9	3.9	1.20	4,683
15	0.58	899	2,790	2,539	2,515	3,814	135.3	555.9	691.2	4.1	1.19	5,178
16	0.60	930	2,793	2,540	2,515	3,821	135.7	557.6	693.4	4.1	1.19	5,190
17	0.62	961	2,803	2,542	2,517	3,842	137.3	563.7	701.0	4.1	1.19	5,232
18	0.64	992	2,830	2,555	2,518	3,901	141.0	587.1	728.1	4.2	1.18	5,356
19	0.66	1,023	2,852	2,556	2,517	3,949	144.0	607.3	751.3	4.2	1.18	5,459

Shell No.	RF No.	Gold Price ($/oz)	NPV Best ($M)	NPV Spec. ($M)	NPV Worst ($M)	Pit Value ($M)	Ore (Mt)	Waste (Mt)	Total (Mt)	SR t:t	Au (g/t)	Au (koz)
20	0.68	1,054	2,882	2,547	2,497	4,016	151.0	631.0	782.0	4.2	1.16	5,619
21	0.70	1,085	2,896	2,546	2,494	4,047	153.6	646.2	799.8	4.2	1.15	5,696
22	0.72	1,116	2,901	2,544	2,491	4,057	154.8	650.7	805.5	4.2	1.15	5,725
23	0.74	1,147	2,925	2,533	2,466	4,112	161.3	679.6	840.9	4.2	1.13	5,885
24	0.76	1,178	2,943	2,537	2,452	4,156	165.6	710.1	875.7	4.3	1.13	6,018
25	0.78	1,209	2,946	2,528	2,448	4,164	166.6	716.1	882.7	4.3	1.13	6,045
26	0.80	1,240	2,951	2,530	2,439	4,177	168.2	726.6	894.9	4.3	1.13	6,090
27	0.82	1,271	2,961	2,532	2,410	4,203	171.9	755.5	927.4	4.4	1.12	6,198
28	0.84	1,302	2,965	2,527	2,400	4,211	173.1	764.4	937.5	4.4	1.12	6,232
29	0.86	1,333	2,965	2,527	2,399	4,212	173.4	765.3	938.6	4.4	1.12	6,236
30	0.88	1,364	2,969	2,523	2,383	4,222	175.2	783.0	958.2	4.5	1.12	6,296
31	0.90	1,395	2,970	2,518	2,376	4,225	176.2	789.2	965.5	4.5	1.12	6,320
32	0.92	1,426	2,983	2,518	2,263	4,266	185.2	898.9	1,084.1	4.9	1.12	6,645
33	0.94	1,457	2,984	2,508	2,235	4,272	186.8	918.3	1,105.2	4.9	1.12	6,700
34	0.96	1,488	2,985	2,451	2,222	4,274	188.5	929.4	1,117.9	4.9	1.11	6,740
35	0.98	1,519	2,985	2,448	2,220	4,274	188.9	931.5	1,120.3	4.9	1.11	6,749
36	1.00	1,550	2,955	2,477	1,831	4,290	206.0	1,319.5	1,525.5	6.4	1.16	7,697

Note: NPV = net present value

The shell selection is presented in Table 15-4. Pit shell 23 was selected as the optimum final pit shell which corresponds to a $1,147/oz pit shell (Revenue Factor 0.74). This shell has a total tonnage of 840.9 Mt including 161.3 Mt of ore at an average grade of 1.13 g/t Au for 5.89 Moz of contained gold. The average strip ratio is 4.2:1.

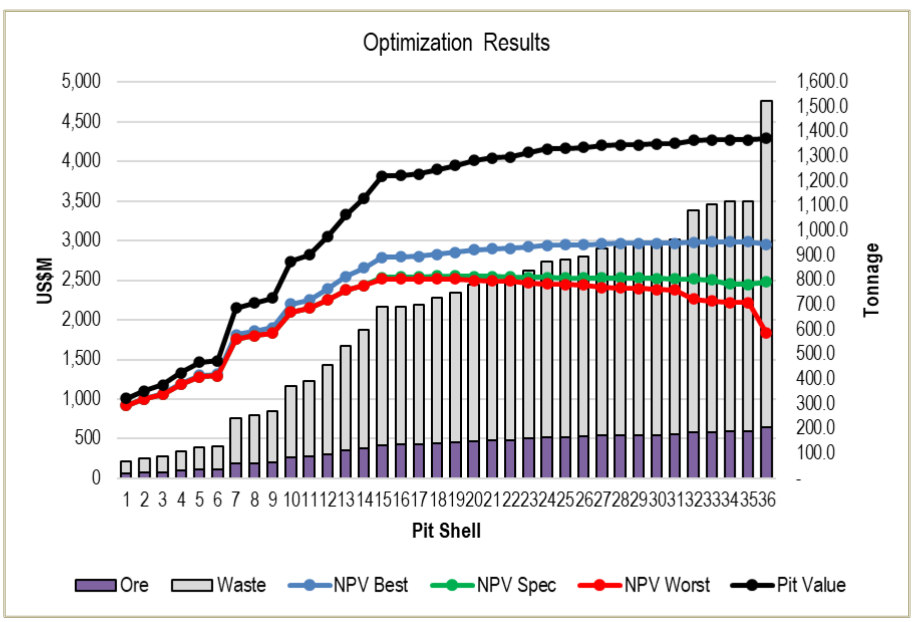


Figure 15-2: Pit-by-Pit Graph of Measured and Indicated Resources

Table 15-4: Measured and Indicated Mineral Resource Pit Shell Selection

Description	Unit	Value
Shell Number	-	23
Shell Revenue Factor	-	0.74
Shell Price	$/oz	1,147
Total Tonnage	kt	840.9
Waste Tonnage	kt	679.6
Strip Ratio	W:O	4.2
Ore Tonnage	kt	161.3
Grade	g/t Au	1.13
Contained Gold	koz	5,885

15.3 Mine Design

15.3.1 Underground Voids

The presence of underground stopes was considered when designing the pits, mainly for the open voids in the F Zone which are 150 m high and 30 m wide. Most of the other underground openings are backfilled with sand fill or rock fill.

When designing the open pit all attempts were made to avoid intersecting underground voids with the designed walls.

15.3.2 Ramp Design Criteria

The ramps and haul roads are designed for the largest equipment, which is a 250-tonne-class haul truck with a canopy width of 8.65 m. For double-lane traffic, industry best-practice was used to design a travelling surface of at least three times the width of the largest vehicle. The double-lane ramp width is 32.0 m, and the single-lane ramp is 18.5 m. Single-lane ramps are introduced in the pit bottom when the benches start narrowing and when the mining rates will be significantly reduced. Ramp gradients are established at 10%.

A shoulder barrier or safety berm on the outside edge will be constructed of rock to a height equal to the rolling radius of the largest tire using the ramp, which is the haul truck tire at a height of 1.8 m. These shoulder barriers are required wherever a drop-off greater than 3 m exists, and the barriers will be designed with side slopes of 1.1H:1V. A ditch on the highwall will capture run-off from the pit-wall surface and assure proper drainage of the running surface. To facilitate drainage of the roadway, a 2% cross slope on the ramp is planned.

15.3.3 Open Pit Mine Design Results

The final pit design is presented in Figure 15-3. The final pit is 2,300 m along strike, 1,000 m wide, and reaches a depth of 590 m from surface. The final pit design has two exits: one to the south 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 does not connect to the east ramp system. The ramp system introduces 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.

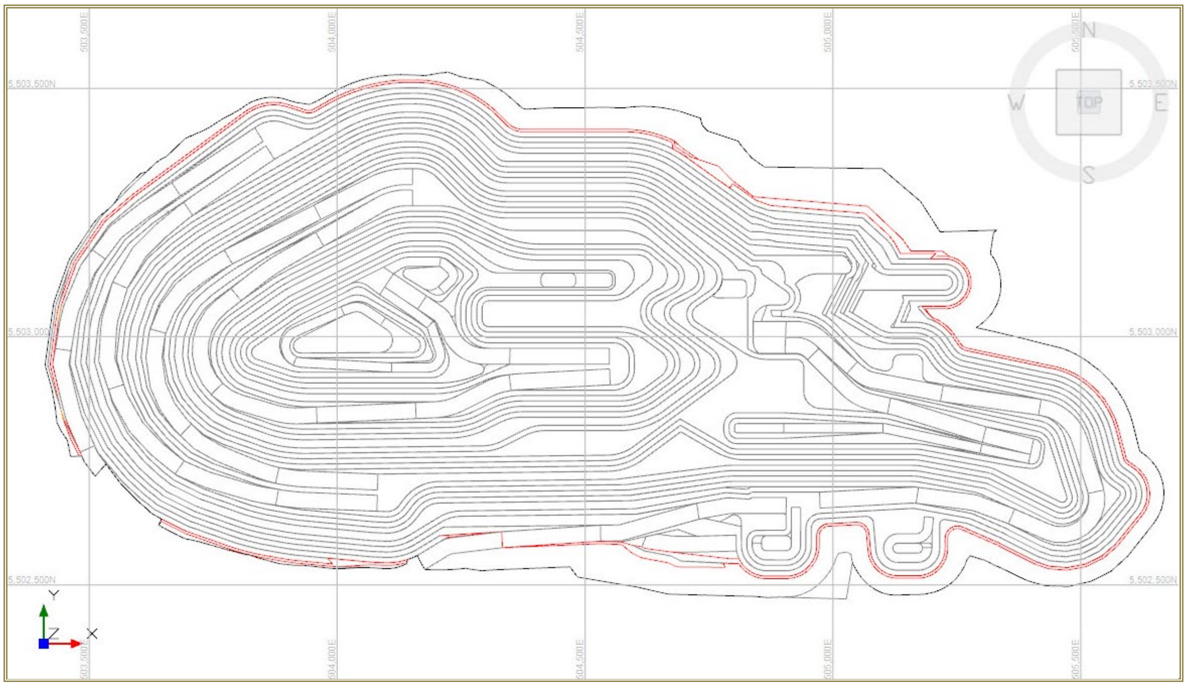


Figure 15-3: Final Pit Design

A three-dimensional view and longitudinal section are presented in Figure 15-4. Several of the underground voids will be entirely mined out by the final open pit, but certain voids will remain in the wall, such as those from the F Zone at depth.

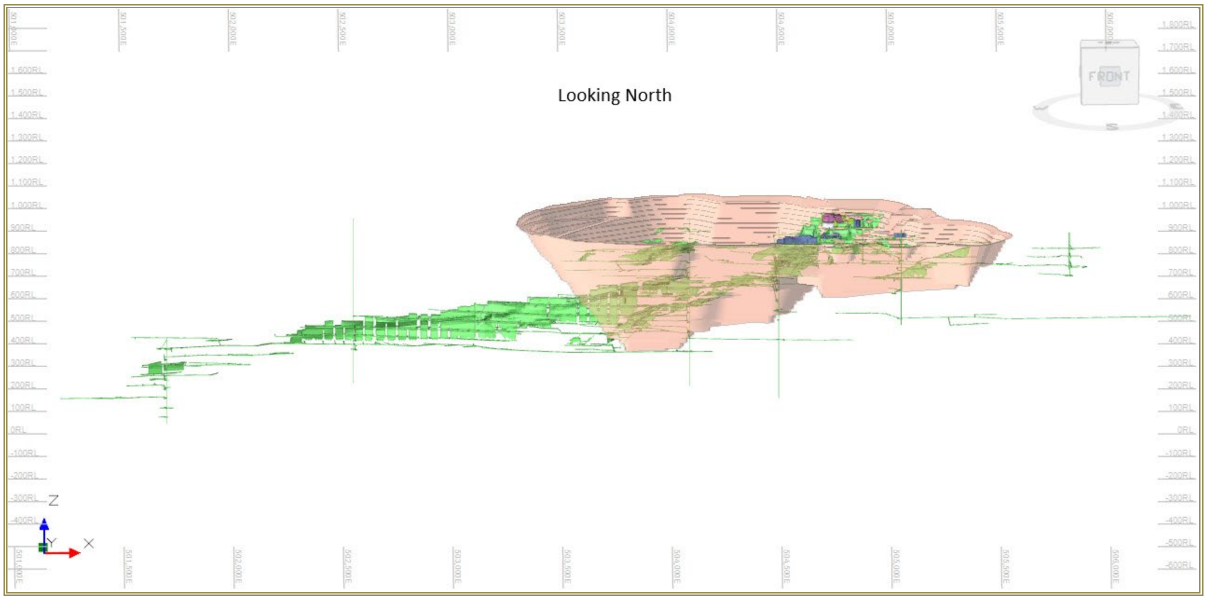


Figure 15-4: 3-D View of Final Open Pit with Historical Underground Voids

15.4 Waste Tonnage

The waste tonnage to be mined is estimated at 788,566 Mt for an average strip ratio of 5.5 that includes overburden, historical tailings, and underground backfill, as shown in Table 15-5.

Table 15-5: Waste Tonnages

Waste Material (including Inferred)	Value (kt)
Overburden	13,854
Historical Tailings	4,480
Underground Backfill	1,285
Waste Rock	768,946
Total Waste	**788,566**

16 MINING METHODS

Greenstone Mine consists of a developing open pit that will mine through the historical underground workings of the MacLeod-Cockshutt and Hard Rock Mines.

16.1 Mine Designs

16.1.1 Open Pit Phases

Mining of the pit will occur in five principal phases. The parameters of each phase is summarized in Table 16-1. The objective of pit phasing is to improve the economics either by feeding the highest grade during the earlier years or delaying waste stripping until later years. Phase 1A was designed to initiate mining before the Trans-Canada Highway was relocated from within the ultimate pit boundary.

Table 16-1: Pit Phase Design Summary

Content	Unit	Phase Design						Total Pit
		Phase 1A	Phase 1B	Phase 2	Phase 3	Phase 4	Phase 5	
Total Tonnage	kt	35,559	115,347	143,529	320,768	200,641	115,399	931,244
Overburden	kt	1,201	747	1,134	2,225	2,031	3,969	11,308
Tailings	kt	507	586	2,175	1,212	0	0	4,480
PAG Rock	kt	1,905	4,314	4,221	7,770	1,114	969	20,293
Waste Rock	kt	26,861	82,487	111,630	264,384	177,442	89,680	752,485
Diluted Ore	kt	5,085	27,212	24,369	45,176	20,054	20,781	142,678
Diluted Grade	g/t Au	1.43	1.26	1.38	1.23	1.32	0.90	1.25
In Situ Gold	koz	234	881	1083	1,786	850	599	5,432
Strip Ratio	W:O	6.0	3.2	4.9	6.1	9.0	4.6	5.5
% of Gold	%	4.3	16.2	19.9	32.9	15.6	11.0	100

The phase designs uses 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, which can create a greater 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 Phase 1 design is two-phase (1A and 1B) and commences with the constraint of the 50 m buffer from the Trans-Canada Highway 11 prior to its relocation, but without the constraint of avoiding the historical MacLeod High Tailings. Phase 1 pit uses the same ramp from the starter pit but descends significantly deeper. On the west side of Phase 1 is a flattened bench that will act as a plateau and dumping grounds for the historical tailings. Moving these historical tailings allows mining to progress to the north. This plateau will be preserved throughout Phase 2 until it is eventually removed.

Phase 2 is created with final pit wall parameters to mitigate extended stripping. Phase 2 is restricted by the historical tailings and maintains a 50 m buffer. The Trans-Canada Highway 11 will already be relocated at this point to allow mining to proceed northwards. The plateau developed in Phase 1 is retained, with an independent ramp for material removal.

Phase 3 begins a new temporary ramp for access to the ore body until the depth of the pit surpasses Phase 2. This explains the seemingly abrupt end to the Phase 3 temporary ramp when the switch is made from the Phase 2 ramp system.

Phase 4 is the maximum depth of the pit. An internal sub-pit is mined at the end of Phase 4 to ensure that the main haulage lane always has a side wall during production and is less prone to impacts from geological faults and sloughing. At the end of Phase 4 the main haulage lane is reduced to a single lane for a portion of the ramp. This can only be done once the haulage from the main portion of the pit is completed.

Phase 5 is the eastern extension of the pit and has a new independent ramp that will act as the main haulage lane for the phase. This ramp will connect to and replace the main haulage lane created in Phase 2. At the end of Phase 5 two sub pits are mined on the edge of the Phase 5 pit. There are no constraints as to when they are mined; when they are mined is determined by the optimized schedule.

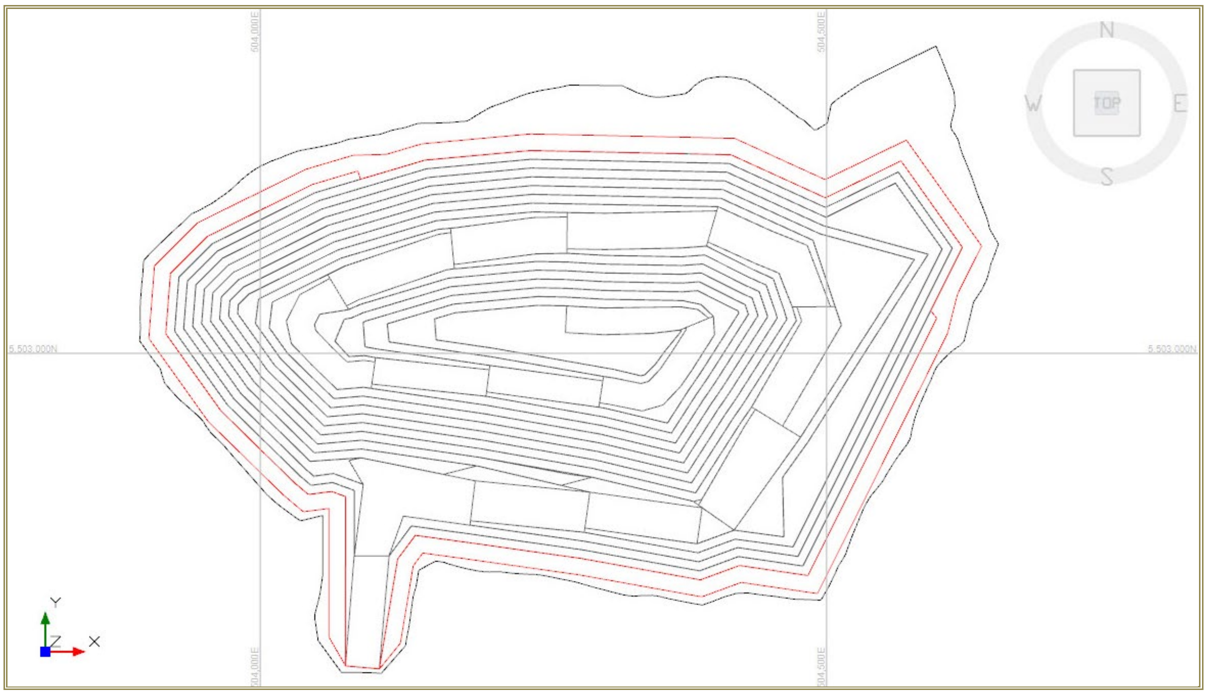


Figure 16-1: Phase 1A Phase Design

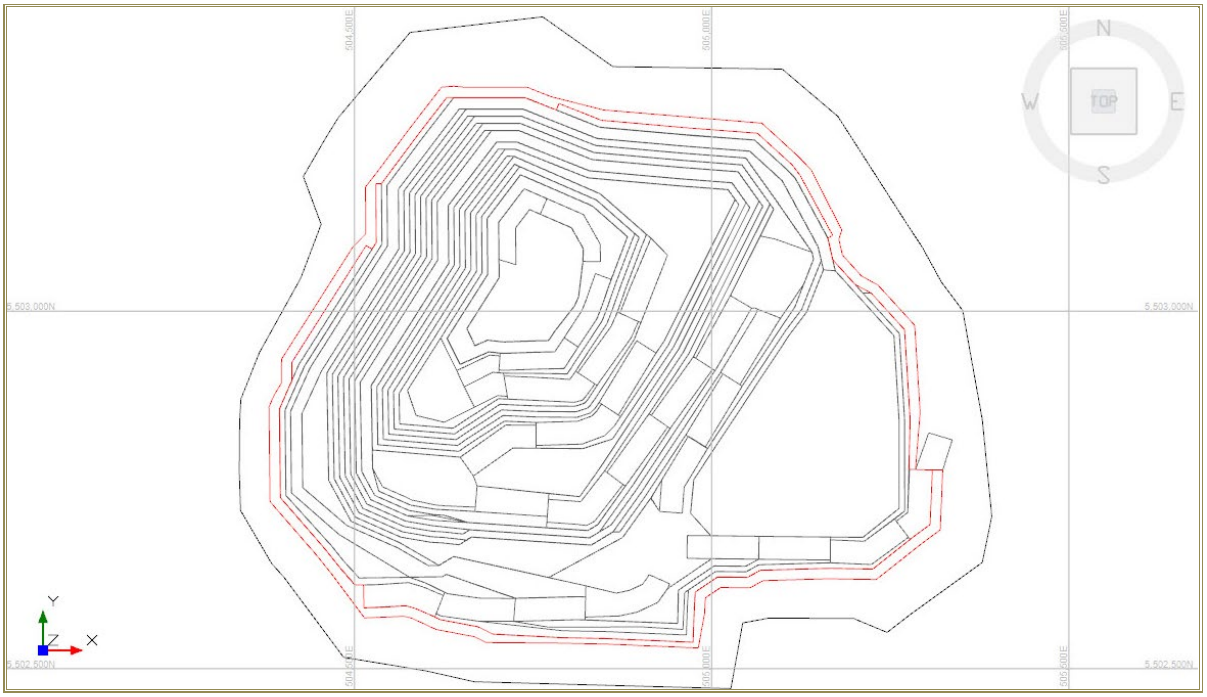


Figure 16-2: Phase 1B Design

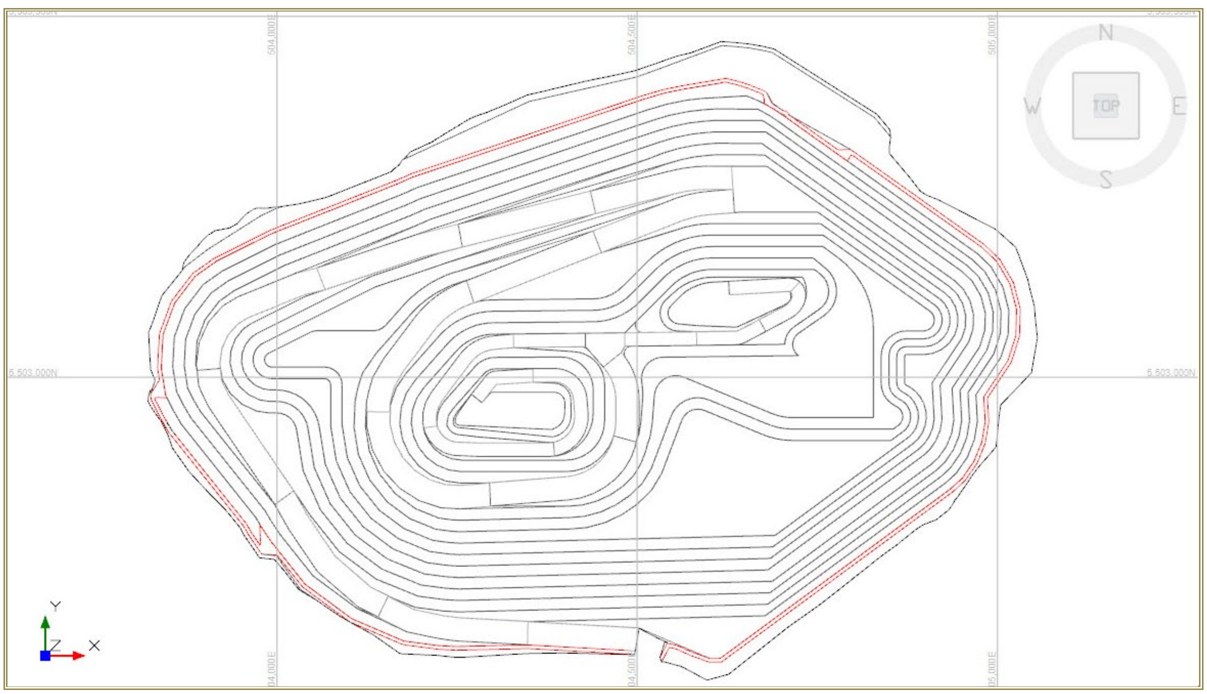


Figure 16-3: Phase 2 Design

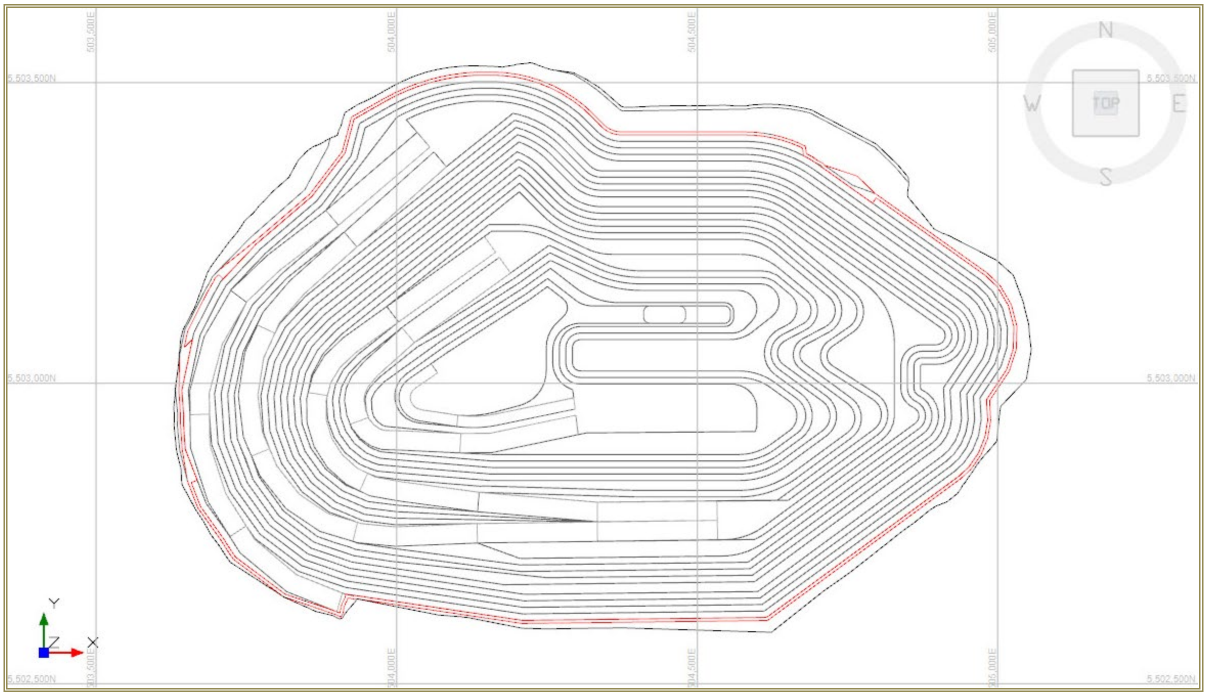


Figure 16-4: Phase 3 Design

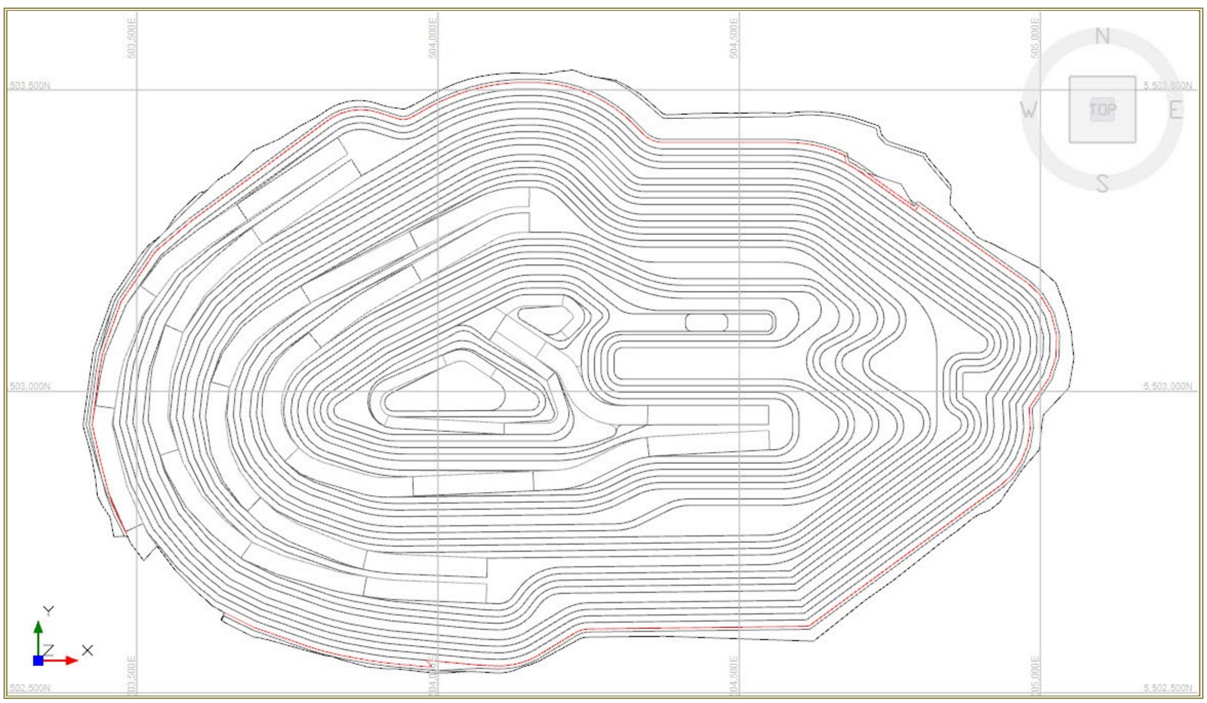


Figure 16-5: Phase 4 Design

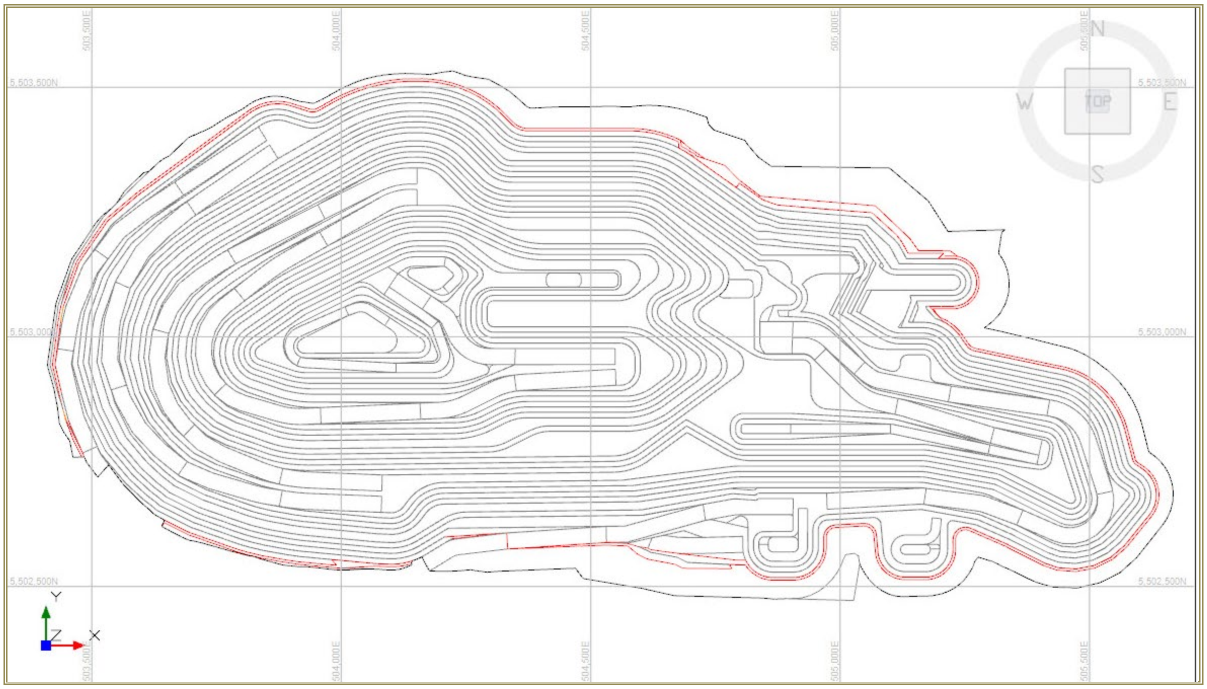


Figure 16-6: Phase 5 Design

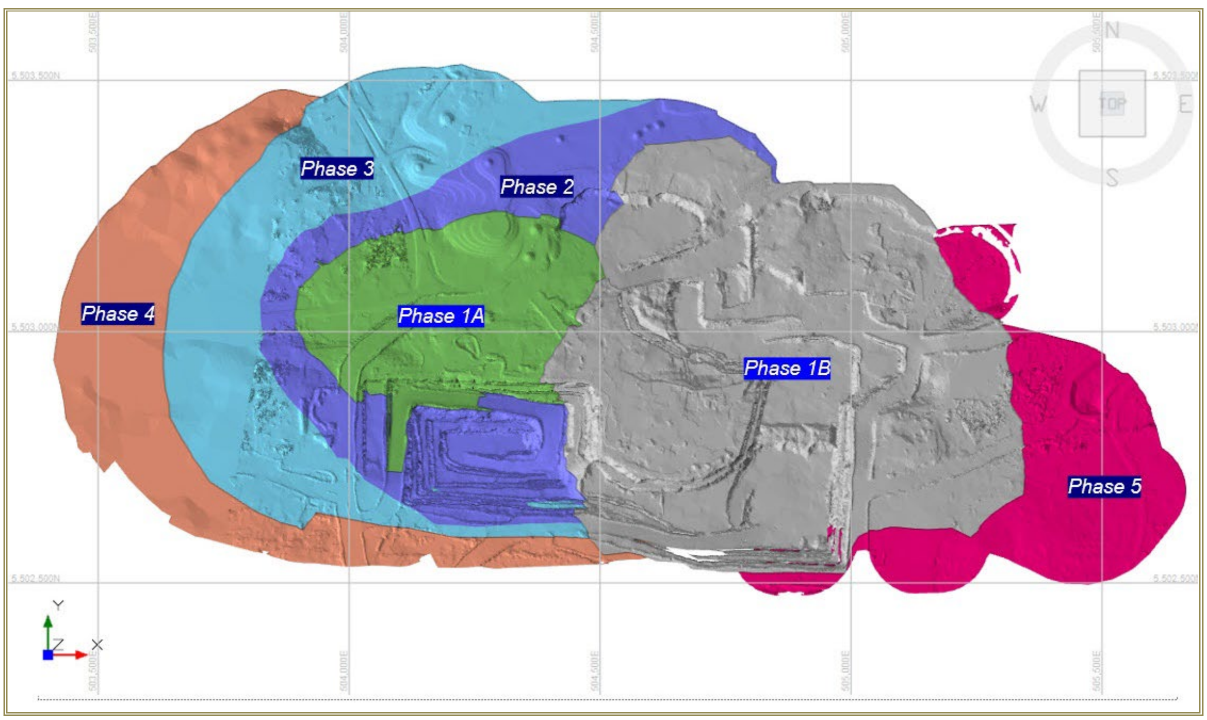


Figure 16-7: All Phases

16.1.2 Pit Slope Geotechnical Assessment

Golder was mandated to produce a feasibility study-level pit-slope design study to support the mine designs. Golder's conclusions have been used in the pit optimization and design process. Their scope of work included reviewing geotechnical field investigations, carrying out follow-up field investigations, and providing slope designs.

The open pit will be developed in a rock mass rated "good" to "very good," where rock mass failure is not a concern. Historical underground longwall mining within the ultimate limit of the pit has demonstrated the high quality of the rock mass. The locations of the underground workings and whether they are backfilled or remain as open voids are well understood.

Pit slope stability is governed by structural controls, the most significant being foliation control on the bench-face angle and the potential control of flat joint-sets on the bench-crest back-break angles. No major faults have been identified that will adversely daylight through the final pit walls. While there are localized differences in the orientations of the discontinuity (joint set) populations, they do not justify distinctly different slope designs. Double benching will have to be done with vertical pre-split drilling, no sub-grade drilling; well controlled blasting practices are required.

For pit optimization, an overall slope angle of 55° was used to simulate the inclusion of ramps in the pit design. In the overburden and historical tailings materials, an overall slope angle of 25° was selected.

The final pit was designed using a double benching configuration to a final height of 20 m. The pit slope profile is based on Golder's recommendations, given in Table 16-2. The slope profile is based on vertical batter angles with a 10 m catch bench width for an inter-ramp angle of 63.4°. A 16 m geotechnical berm is introduced every 100 m, where ramp segments do not pass in the slope to reduce the vertical height.

Table 16-2: Pit Wall Geotechnical Design Criteria

Slope Parameters	Unit	Temporary Pit Walls	Final Pit Walls
Final Bench Height	m	20.0	20.0
Bench Face Angle	°	90	90
Average Design Catch Bench Width	m	15.5	10.0
Inter-ramp Angle	°	52.2	63.4
Overall Slope Angle[1]	°	52.2	60.8
Geotechnical Benches	m	16.0[2]	16.0[3]

Notes: [1] Overall slope angle calculation does not include ramps. [2] Geotechnical catch berms designed as required on temporary walls. [3] Geotechnical catch berms will be 16.0 m at every 100 m for final walls.

At the bedrock-overburden contact, the overburden slope is 2H:1V. The overburden slopes will comprise fluvial or glacial cohesionless or cohesive material of sufficient strength to maintain this slope angle. 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 Golder reported, the rock mass is assumed to have a very low permeability, and the rate of groundwater infilling of the historical underground workings is unknown. For slope-stability assessments, it has been assumed that the pit slopes will be partially saturated with draw-down cones similar to another open pit in the region.

16.1.3 Overburden and Waste Rock Storage

Waste rock will be disposed of in four distinct waste rock storage areas (WRSA or waste rock dumps) of which three are located around the pit and one further to the south. An overburden stockpile is also located near the pit. The open pit generates 788.6 Mt of waste rock, which includes 1.3 Mt of underground void backfill, 4.5 Mt of historical tailings, 13.9 Mt of overburden, and 17.7 Mt of PAG material. The tailings material will be transported for disposal within the TMF.

The design criteria of each waste rock dump have been adjusted based on foundation stability assessments, which may include foundation preparation. All waste rock dumps have 20 m high lifts to allow for wider catch benches to facilitate reclamation. Design criteria for the waste rock dumps are shown in Table 16-3. All waste dump storage capacities are shown in Table 16-4.

Table 16-3: Waste Rock Dump Design Criteria

Waste Rock Storage Area (Dump)	Average Catch Bench Width (m)	Slope Angle (°)	Overall Slope Angle (H:V)	Maximum Elevation (m)	Approximate Height (m)
A	13.5	37	2:1	430	100
B	13.5	37	2:1	430	100
C	13.5	37	2:1	450	110
D	13.5	37	2:1	550	220

Note: H:V = horizontal to vertical.
Shear key excavations are being constructed to stabilize the dumps where ground conditions are inadequate to support the slope angles.

Table 16-4: Waste Rock Dump Storage Capacities

Waste Rock Storage Area (Dump)	Remaining Storage Capacity (Mt)	Remaining Storage Capacity (Mm^3)	Surface Area (ha)	% Filled at Start	% Filled at End
A	29.0	15.1	41	18	100
B	42.3	22.4	44	0	100
C	99.0	51.1	95	7	100
D	592.3	307.1	301	0	96
Total	**762.6**	**395.7**	**481.4**	**3.2**	**96.8**

Notes: Remaining storage is based on the effective date of June 30, 2024.
Values do not include waste rock that was used for initial Mine construction purposes (i.e., crusher ramp, access roads, aggregates, TMF).

16.1.4 Ore Stockpiles

The ore stockpile is designed with a maximum storage capacity of 12.9 Mt. The stockpile reaches a maximum of 10,659 kt in 2032; however, in most years the stockpile is well below capacity with sufficient margin to account for changes in mine plan or stockpiling. There are four gold grade-bins of stockpiled material ranging from marginal to high grade. These bins are to be stored within the main stockpile in separate piles to reduce dilution. There is adequate space to have smaller stockpiles of different grades on the pad.

Table 16-5: Ore Grade-Bin Cut-Off Grades

Grade Bins	Cut-Off Grade (g/t)
Bin 1	1.10
Bin 2	0.50
Bin 3	0.35
Bin 4	0.30

The ore stockpile design criteria are presented in Table 16-6. 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)	Maximum Height (m)
Stockpile	7.1	2:1	400	70

The ore 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 will be built to ensure consistent and minimal rehandling times.

16.1.5 Mine Haul Roads

The Mine haul roads from the pit to the dumps, crusher, and TMF are built; however, additional haul roads will be built during operations as the pit evolves. Over the LOM, an additional 5.2 km of haul roads will be built.

16.2 Production Schedule

The LOM Mine production schedule is calculated monthly for the remainder of 2024 and all of 2025. For 2026 and 2027, the schedule is developed on a quarterly basis, and on an annual basis for 2028 and beyond.

The objectives of the LOM plan are to maximize discounted operating cash flow subject to several constraints:

- Supply best grade ore to the plant and feed at a nominal capacity, which ultimately reaches 27,000 t/d (9.86 Mt/a).
- Limit the mining rate to approximately 72 Mt/a.
- Limit the vertical drop-down rate within the pit to approximately eight benches per phase, per year.
- Limit peak haul truck requirements.

- Use a grade segregation and stockpiling strategy, with a maximum stockpile of 10.7 Mt that is roughly equivalent to one year of milling.

Year 1 (2024) of mining is currently in the ramp-up phase. The peak mining rate of approximately 72 Mt will be achieved in 2025 and will be maintained for ten years (2025 to 2034). In 2035, the rate gradually declines as either sufficient ore for the mill is available or to limit peak truck requirements. The annual LOM mine production, stockpile inventory, mill production, and gold production are presented in Figure 16-8 to Figure 16-11. Figure 16-12 to Figure 16-15 present the end of period mine infrastructure status at different dates.

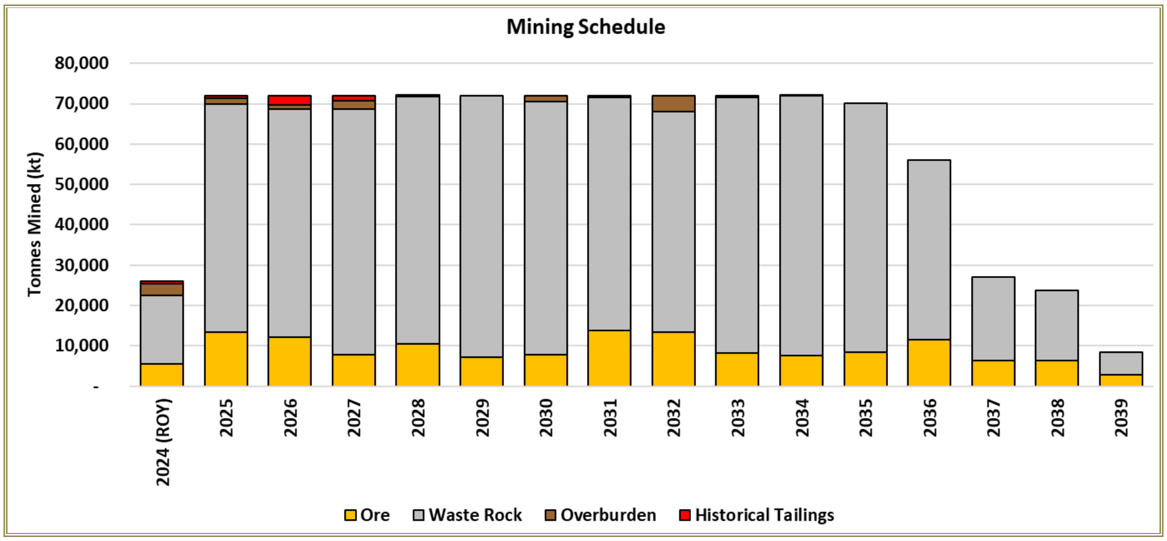


Figure 16-8: Annual LOM Mine Production

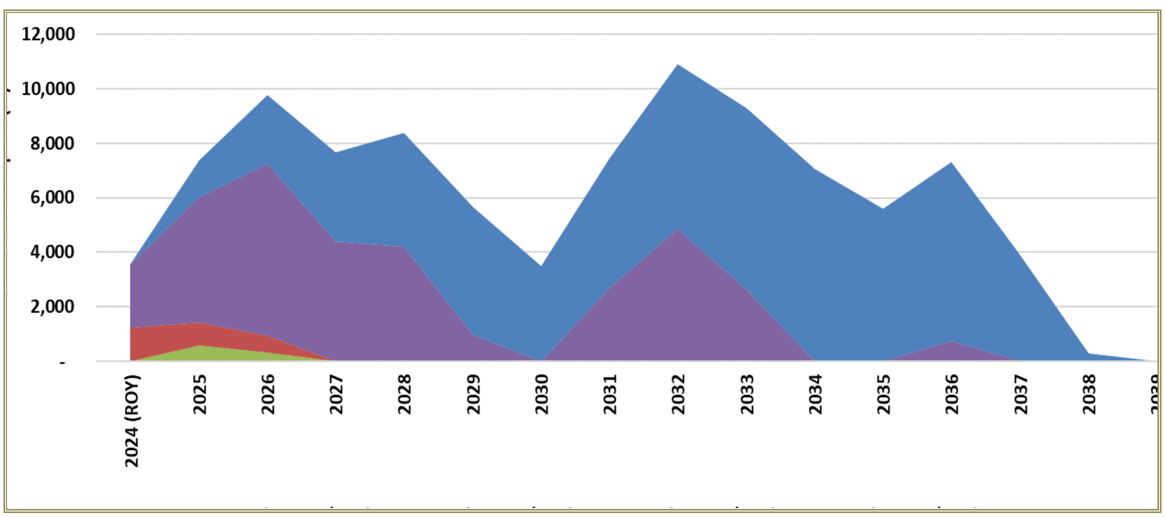


Figure 16-9: Annual LOM Ore Stockpile Inventory

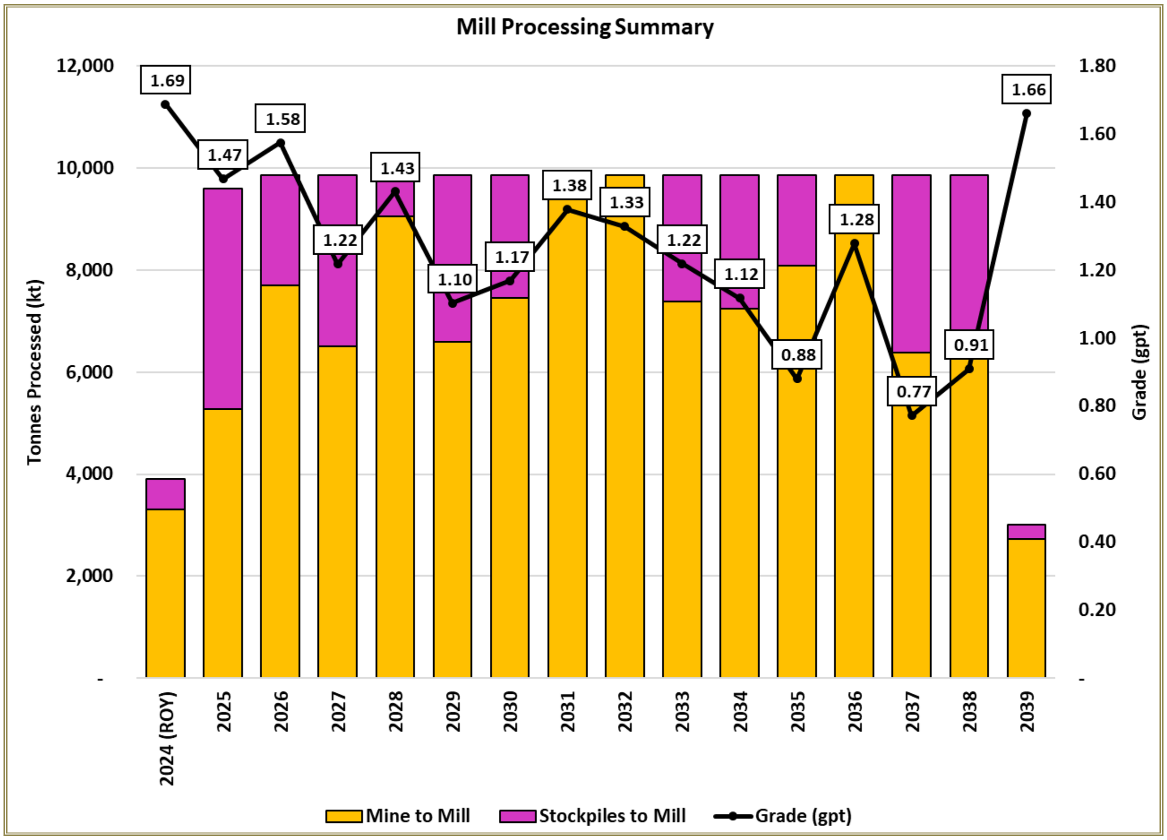


Figure 16-10: Annual LOM Mill Production

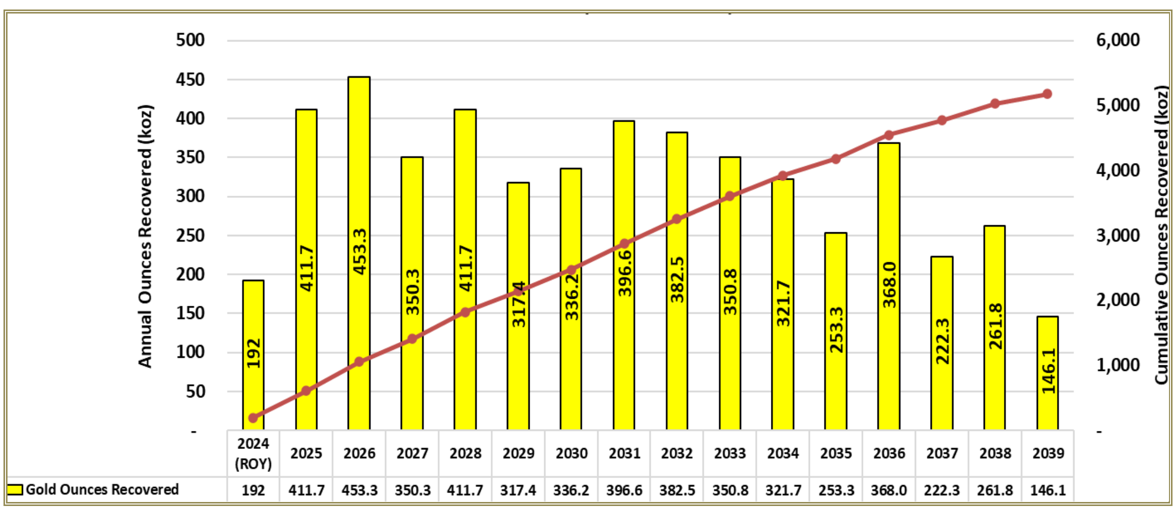


Figure 16-11: Annual LOM Gold Production

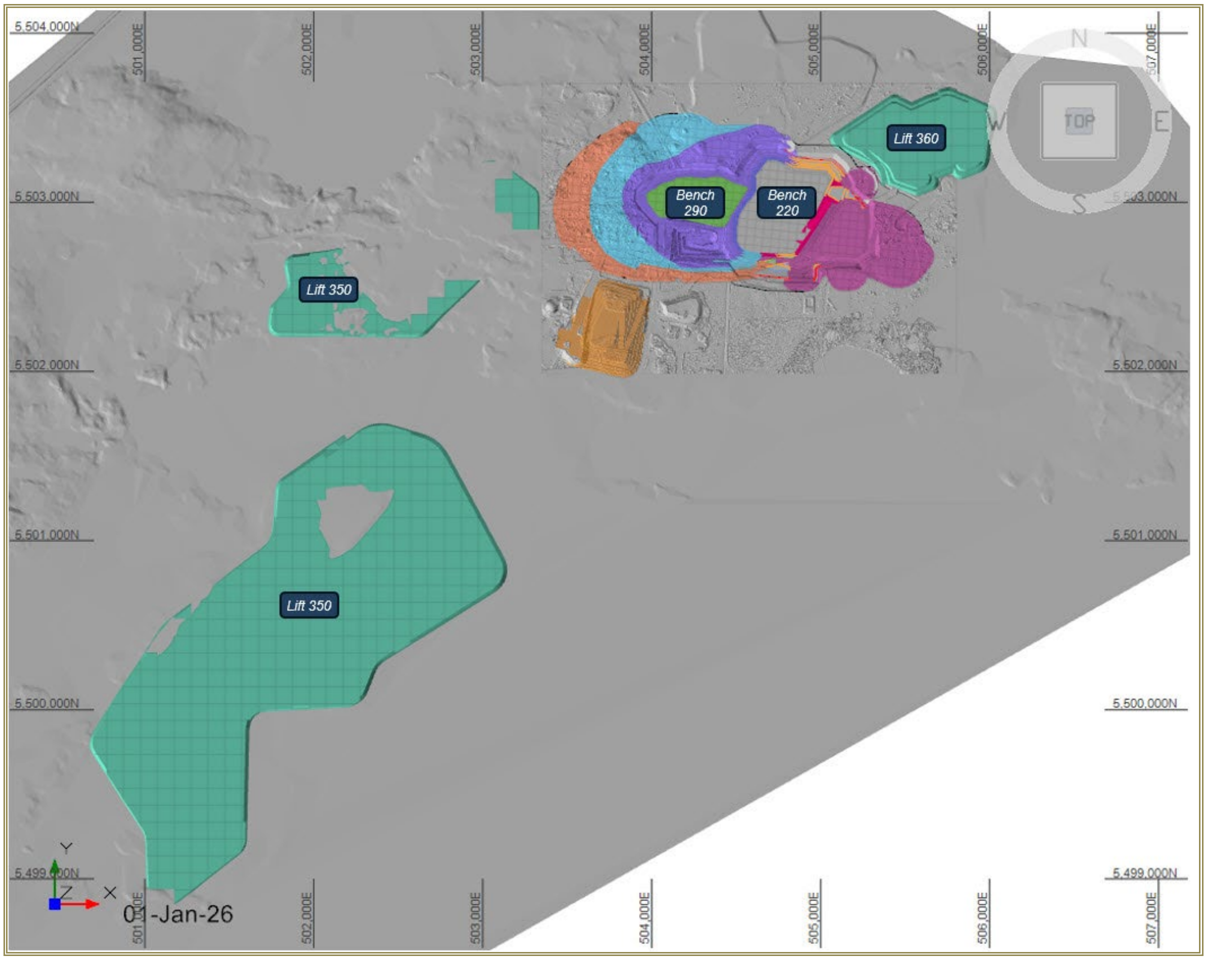


Figure 16-12: Pit Production Schedule—Year -1

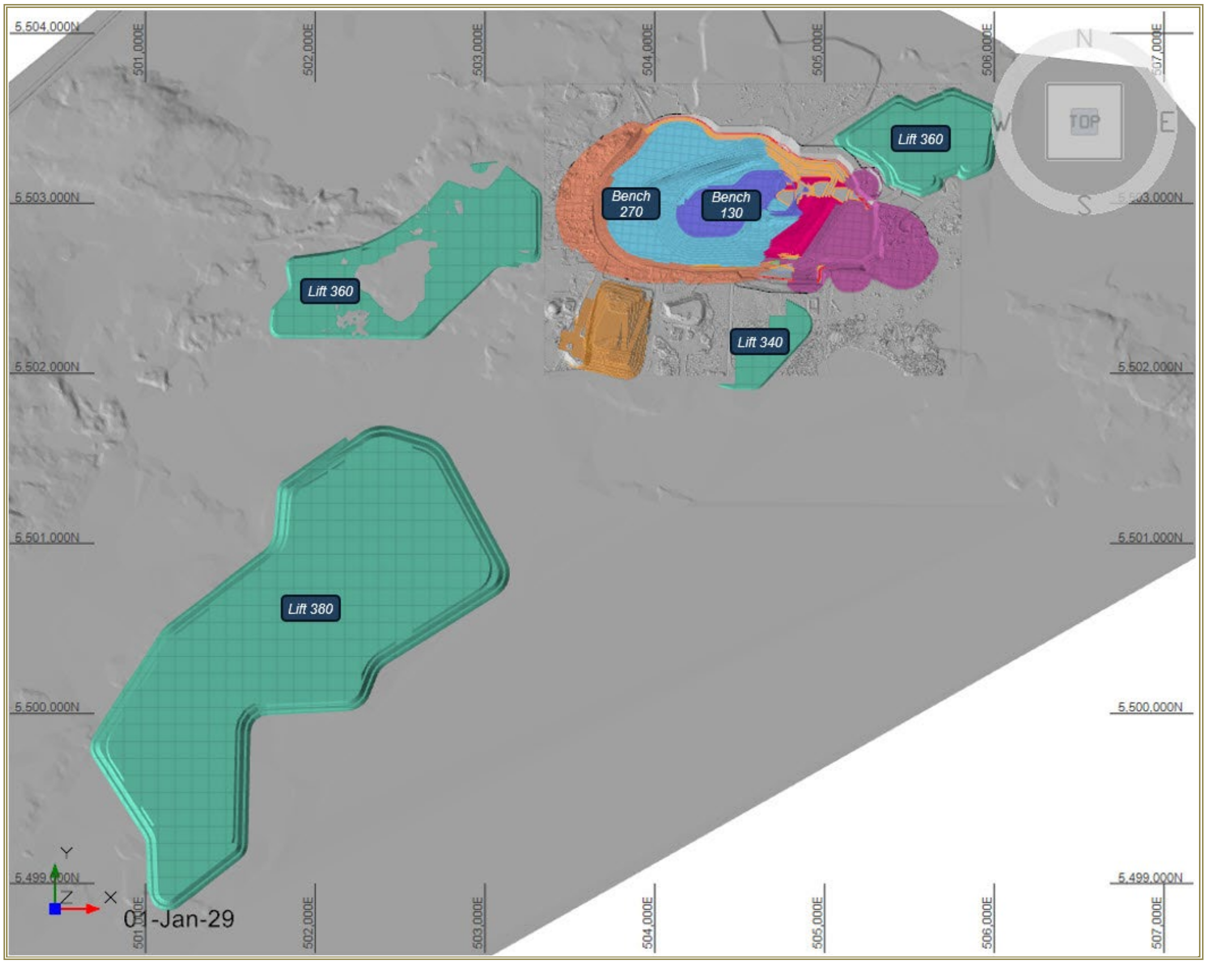


Figure 16-13: Pit Production Schedule—Year 5

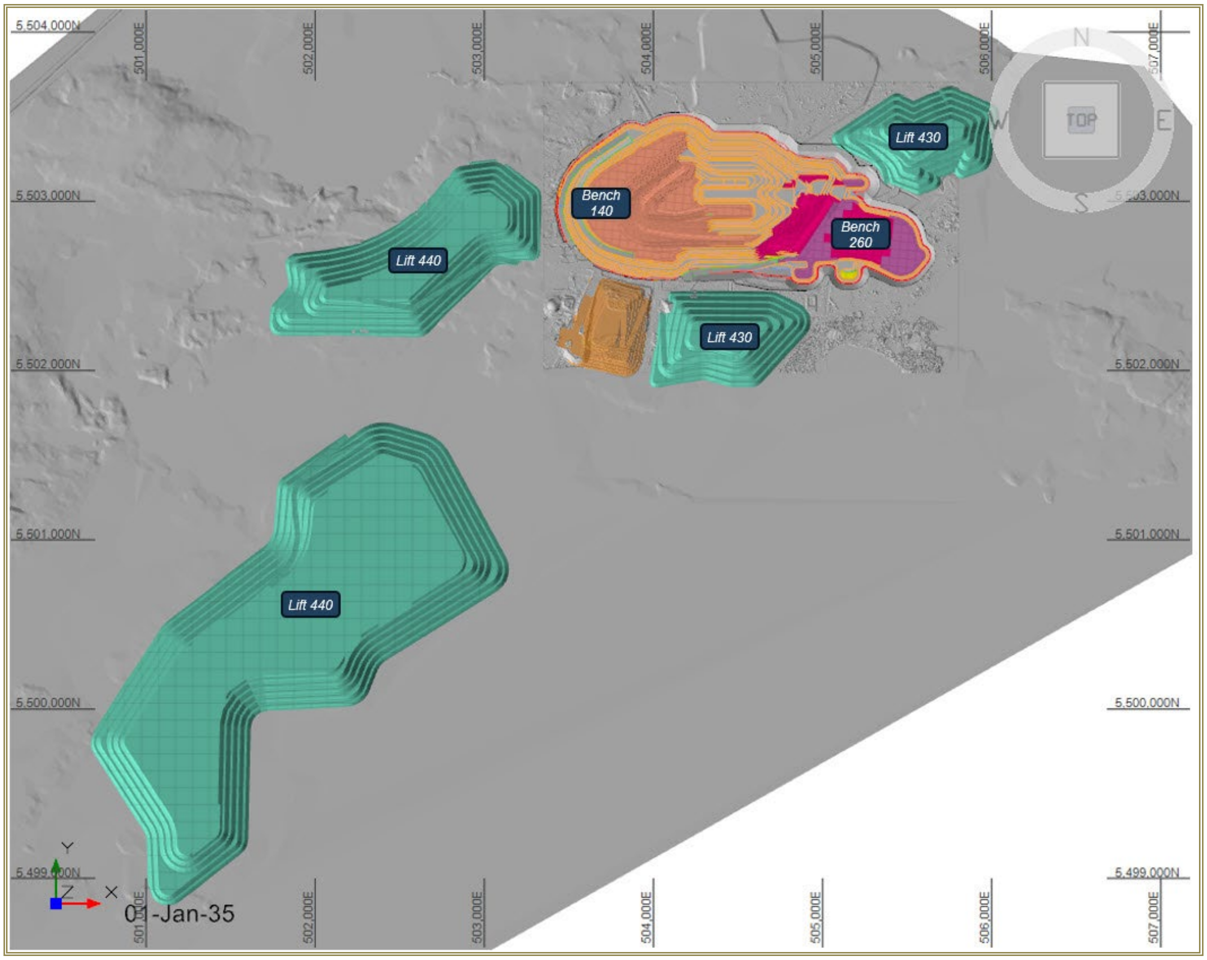


Figure 16-14: Pit Production Schedule—Year 10

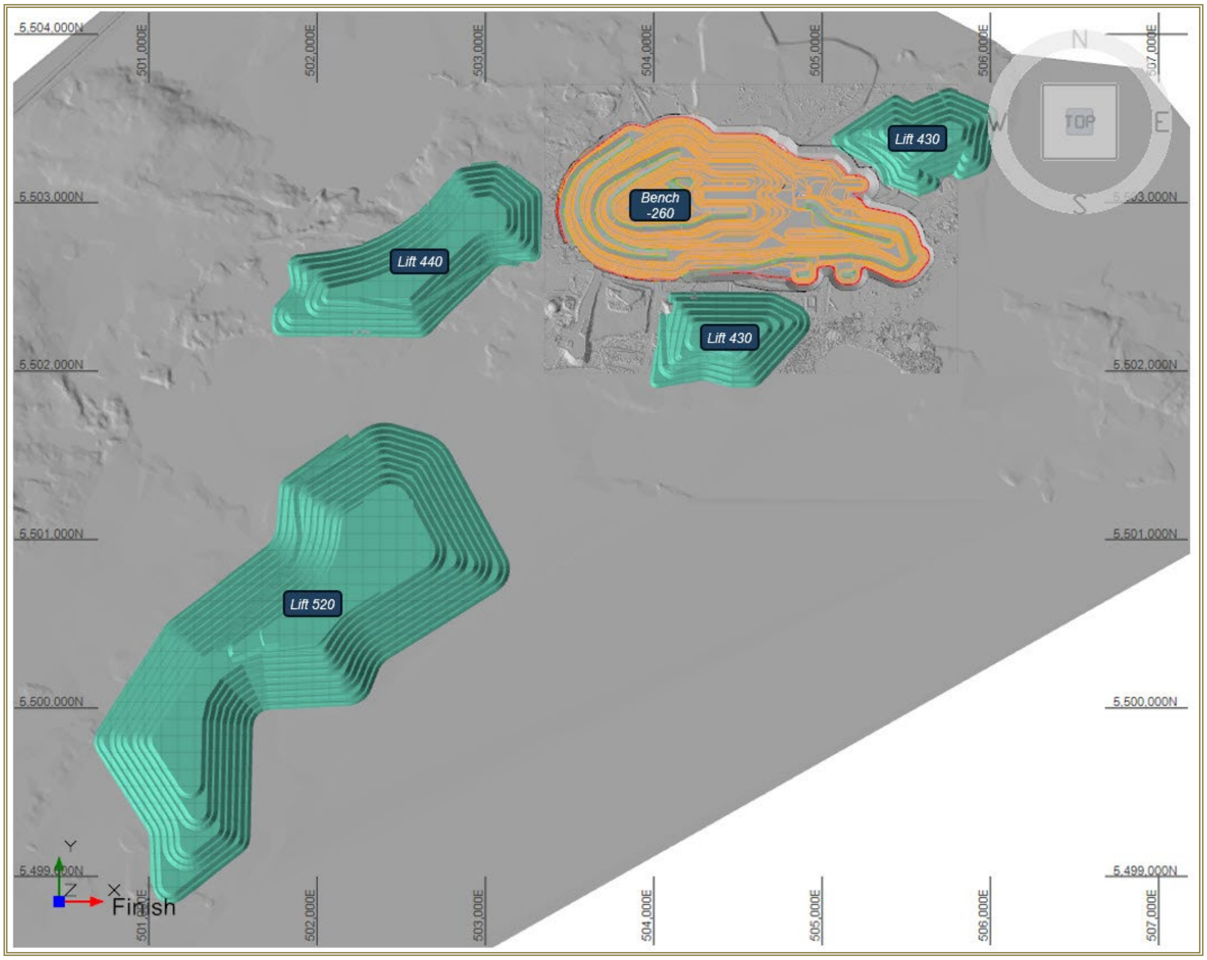


Figure 16-15: Pit Production Schedule—End of LOM

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 389 koz from January 2025 through December 2029, with an average head grade of 1.36 g/t Au and an average metallurgical recovery of 90.8%. Over the LOM 5.18 Moz of gold are produced, an increase from the 5.05 Moz contemplated in the previous study.

Table 16-7: LOM Production Schedule

Year	Mining					Processing			
	Ore Mined (kt)	Grade (g/t Au)	Contained Gold (koz)	Waste Mined (kt)	Total Mined (kt)	Ore Milled (kt)	Grade (g/t Au)	Contained Gold (koz)	Recovered Gold (koz)
2024 (ROY)	5,492	1.17	206.7	20,537	26,029	3,904	1.69	211.9	192.4
2025	13,428	1.22	528.8	58,568	71,996	9,605	1.47	453.5	411.7
2026	12,229	1.31	513.6	59,756	71,985	9,857	1.58	499.2	453.3
2027	7,780	1.32	329.7	64,216	71,996	9,857	1.22	385.8	350.3
2028	10,547	1.36	459.7	61,447	71,994	9,857	1.43	453.4	411.7
2029	7,131	1.37	313.1	64,867	71,998	9,857	1.10	349.5	317.3
2030	7,714	1.39	345.6	64,287	72,000	9,857	1.17	370	.336.2
2031	13,800	1.09	485.6	58,193	71,994	9,857	1.38	436.8	396.6
2032	13,315	1.08	462.7	58,679	71,994	9,857	1.33	421.2	382.5
2033	8,244	1.37	363.0	63,728	71,972	9,857	1.22	386.3	350.8
2034	7,628	1.32	324.0	64,372	72,000	9,857	1.12	354.3	321.7
2035	8,405	0.98	263.9	61,718	70,124	9,857	0.88	278.9	253.3
2036	11,560	1.14	424.4	44,538	56,098	9,857	1.28	405.3	368.0
2037	6,412	1.01	208.0	20,612	27,023	9,857	0.77	244.9	222.3
2038	6,268	1.25	251.6	17,472	23,741	9,857	0.91	288.3	261.8
2039	2,725	1.80	158.0	5,576	8,301	3,013	1.66	160.9	146.1
Total	**142,678**	**1.23**	**5,638**	**788,566**	**931,244**	**144,662**	**1.23**	**5,700**	**5,176**

Note: ROY = rest of year.

16.3 Mine Operations and Equipment Selection

16.3.1 Mine Operations Approach

Mining is performed using conventional open pit methodology with GGM's hydraulic shovels, wheel loaders and haul trucks. Certain support activities, such as explosives manufacturing and blasting activities, are outsourced.

16.3.2 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 blasthole size is used for a 5.6 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. Blastholes 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.

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. The average drilling productivity is estimated at 2,195 t/h.

Table 16-8: Drill and Blast Parameters

Description	Unit	Production Holes
Drill Pattern		
Explosive Type	-	Emulsion
Explosive Density	g/cm^3	1.2
Hole Diameter	in	9.0
Diameter	m	0.229
Burden	m	5.6
Spacing	m	6.5
Subdrill	m	1
Stemming	m	4.2
Bench Height	m	10
Blasthole Length	m	11
Pattern Yield		
Rock Density	t/bcm	2.74
BCM/hole	-	364
Yield per Hole	t/hole	997
Yield per Metre Drilled	t/m drilled	90.7
Powder Factor	kg/t	0.33
Weight of Explosives per Hole	kg/hole	335
Drill Productivity		
Re-drills	%	5
Pure Penetration Rate	m/h	39.1
Hole Length	m	11
Overall Drilling Factor	%	62
Overall Penetration Rate	m/h	24.2
Drilling Productivity	t/h	2,364
Drilling Efficiency	holes/h	2.20

The blasthole rig selected for production drilling has a hole size range of 152 to 270 mm, with a single-pass drill depth of 12.2 m and a 40 ft tower configuration. This rig will have both rotary and down-the-hole (DTH) drilling capability, and 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 is being implemented.

With the automation package, the drill rigs are controlled remotely through a remote operator station (up to three drills per remote station). 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 no longer required.

Blasting activities are outsourced to an explosives provider who is responsible for supplying and delivering explosives down the hole. The Mine engineering department is responsible for designing blast patterns and relaying hole information to the drills via the wireless network or manually.

16.3.3 Grade Control

The ore control program consists of staking dig limits in the field for all material types to guide loading-unit operators. A high-precision system combined with an arm geometry system allows shovels to target small dig blocks and perform selective mining. The system gives operators a real-time view of dig blocks, ore boundaries, and other positioning information.

To have optimal ore–waste boundary identification, RC drilling will target 100% of all ore material, and also capture an average of 25% of the total waste in the pit. Concurrent with the RC sampling, production blastholes will be used to supplement grade control. Currently, sampling targets 100% of all ore and waste material within the mineralized mine trend, and more selective sampling is completed in waste rock outside of the mine trend.

The samples collected are sent to GGM's laboratory in Geraldton for sample preparation and assaying. All samples are collected on the bench and properly tagged by grade-control samplers on each shift.

The grade-control information obtained through RC drilling and blasthole sampling will undergo post-blast adjusting using the blast movement monitoring (BMM) system. A blast movement monitoring system has been included in the blasting cost.

16.3.4 Pre-Split Drilling

Pre-split drill and blast will maximize stable bench faces and inter-ramp angles along pit walls as prescribed by Golder's geotechnical pit slope study. 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 line, then a trim shot will be used to clean the face. Pre-split holes are spaced 1.5 m apart and will be 20 m in length and drilled with a smaller diameter of 127 mm.

As presented in Table 16-9, blasting of the pre-split holes will use a special packaged pre-split explosive, and be internally traced with detonating cord that ensures fast and complete detonation of the decoupled charge. A designed 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

Parameters	Unit	Pre-Split Holes
Drill Pattern		
Hole Diameter	inch	5
Diameter	m	0.127
Spacing	m	1.5
Bench Height	m	20
Hole Length	m	20
Face Area	m^2	30
Explosives Charge	kg	25
Charge Factor	kg/m^2 face	0.83
Cartridge Charge		
No. 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/h	41.2
Overall Drilling Factor	%	58
Overall Penetration Rate	m/h	23.9
Drilling Efficiency	holes/h	1.2
Metres of Drilling per metre Crest	m/m of crest	13.33

The drill selected for this application is a 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.

16.3.5 Loading

Most of the loading in the pit is performed by four 29 m^3 face shovels. The hydraulic shovels are matched with a fleet of Caterpillar 793 mine haul trucks. The shovels will be complemented by one production front-end wheel loader (FEL) with a 30 m^3 bucket. To reduce dilution and increase flexibility, one 15 m^3 backhoe-configured excavator will be added to the loading fleet.

The loading productivity assumptions for both types of loading tools in ore, waste, and overburden are presented in Table 16-10.

Table 16-10: Loading Specifications

	Unit	Loading Unit			
		Shovel (29 m³) Ore/Waste	Shovel (29 m³) Overburden	Excavator (15 m³) Ore/Waste	Loader (30 m³) Ore/Waste
Material					
Rated Truck Payload	t	216	216	216	216
Rated Truck Heaped Volume	m³	152	152	152	152
Bucket Capacity	m³	29.0	29.0	15.0	30.0
Bucket Fill Factor	%	90	90	90	90
In Situ Dry Density	t/bcm	2.75	2.00	2.75	2.75
Moisture	%	3	5	3	3
Swell	%	40	25	40	40
Wet Loose Density	t/bcm	2.02	1.68	2.02	2.02
Actual Load Per Bucket	t	52.7	43.8	27.3	54.5
Passes (calculated)	No.	4.1	4.9	7.9	4.0
Passes (rounded)	No.	4	5	8	4
Actual Truck Wet Payload	t	216	219	218	216
Actual Truck Dry Payload	t	210	208	211	210
Production and Productivity					
Average Production (dry tonnes)	t/h	2,594	2,155	1,500	2,950

16.3.6 Hauling

Haulage is performed with a combination of 224-tonne (Caterpillar 793-08) and 216-tonne (Caterpillar 793F) mine haul trucks. The truck fleet productivity was estimated using Deswik.LHS software. Several haulage profiles and haul routes were used to simulate cycle times. Cycle times were estimated for each mining period and all possible destinations, including several waste storage areas.

On production benches, a speed limitation of 30 km/h is imposed to reflect the lack of proper roads and less favourable rolling conditions in addition to having historical underground workings 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 is constrained at 50 km/h.

Multiple waste dumps were used to help level the truck requirements for the mine. During the critical years of the mine, levelling was achieved by sending waste rock to the closest dumps.

Table 16-12 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, and is also dependant on the dumping schedule and the distance each dump is from the pit. The dump schedule was planned so that cycle time tends to plateau at a maximum limit to allow for a consistent fleet over most of the mine life. Another factor for variable cycle time is the phasing, where new phases have a lower depth and shorter cycle time associated with them, and it is common for three or more phases to be active within the same year. The average cycle time shown in Table 16-12 is a weighted average of all the phase cycles to all the waste dumps. Multiple waste dumps were used to help level the truck requirements for the mine and, during the critical years of the mine, levelling was achieved by sending waste rock to the closest dumps.

Figure 16-16 depicts the tonnage moved by year and the associated cycle times per material.

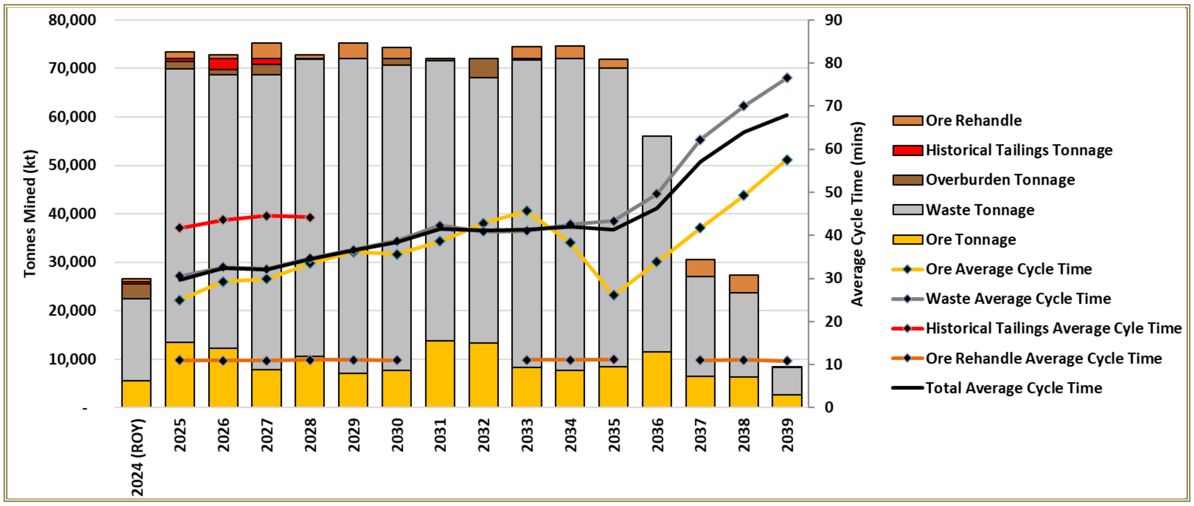


Figure 16-16: Haul Truck Cycle Time by Category and Material Moved

The truck fleet reaches a maximum of 37 units in 2031 and remains at this level until 2034 before it starts decreasing because of a decreasing mining rate.

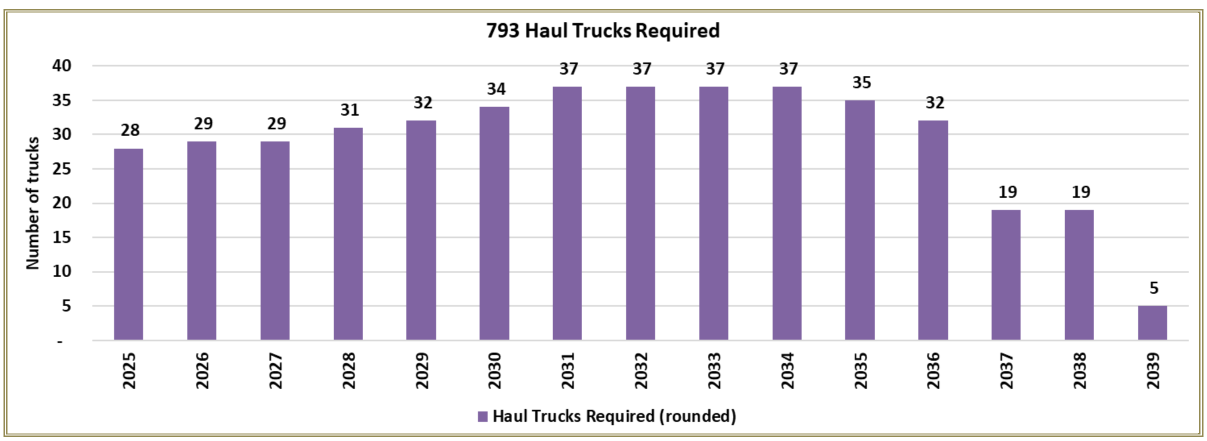


Figure 16-17: Haul Truck Requirements

16.3.7 Equipment Usage Assumptions

The typical equipment usage assumptions are established by equipment groupings as presented in Table 16-11. The annual net operating hours vary between 5,000 and 6,000 h/a.

Table 16-11: Mining Equipment Usage Assumptions

Equipment	Unit	Shovels	Loaders	Trucks	Drills	Ancillary
Length of Period	d	365	365	365	365	365
Availability	%	85.0	82.5	87.5	85.0	85.0
Use of Availability	%	90.0	90.0	90.0	90.0	85.0
Utilization	%	76.5	74.3	78.8	76.5	72.25
Effectiveness	%	87.0	85.0	87.0	85.0	80.0
Overall Equipment Effectiveness	%	66.6	63.1	68.5	65.0	57.8

16.3.8 Pit Dewatering

The open pit dewatering strategy is to use the existing underground stopes and the connectivity of the past underground mines (Hard Rock and Mosher) to keep the water level below the working benches in the pit. The existing Macleod shaft will be used to dewater to the 200 m level, after which dewatering will need to be performed from the Mosher shaft (to reach the −265 m level). 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.

16.3.9 Road and Dump Maintenance

Waste, ore storage, and loading areas will be maintained by a fleet of six Komatsu D375A-8 track-type dozers. A Caterpillar 824 wheel dozer is also dedicated to mine roads and the loading areas.

Mine roads will be maintained by three 16 ft-blade motor graders. A water and sand truck will spray roads to suppress dust or spread road aggregate during winter months. Two small water trucks will spray the smaller site roads to suppress dust and fill the production drills with water.

16.3.10 Support Equipment

All construction-related work, such as berm construction and water-ditch cleaning, will be done by two 49 tonne excavators and one 95 tonne excavator for pit-wall scaling.

16.3.11 Mine Maintenance

The Mine's maintenance department manages and performs maintenance planning and training of employees. Reliance on dealer and manufacturer support is planned for the initial five years of operation, and support for major components exchange programs will be maintained throughout the LOM. Tire monitoring, rotation, or replacement is outsourced to a contractor who is permanently on site.

A computerized maintenance management system is 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 key performance indicators (KPI) and cost-tracking purposes.

16.3.12 Mine Management and Technical Services

The Mine Manager is responsible for the overall management of the Mine. Superintendent positions in operations and engineering, a maintenance manager, and a geology and exploration manager report directly to the Mine Manager.

The mine operations department is composed of two foremen per crew (eight in total); one for loading and hauling and another for drilling and blasting. A mine dispatcher is 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 engineering and geology team 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.

An equipment simulator has been purchased to assist with the with the number of truck operators required to be trained at the start-up and operations phases.

16.3.13 Roster Schedules

Rotating schedules of 7 days on, 7 days off, and 14 days on, 14 days off are worked based on a twelve-hour shift per day. Four crews are required to operate on a continuous basis of 24 h/d and 365 d/a.

16.4 Fleet Management

A fleet management system (Minestar) has been implemented to manage the operation, monitor machine health, and track KPIs. The system is managed by a dispatcher on each crew who controls the system that will sends onscreen instructions to the operators to target peak production efficiency.

A high-precision GPS for machine guidance is used to mitigate the associated risk of working around historical underground workings. This system will enable shovel operators to navigate safely in potentially hazardous areas. Similar high-precision drill navigation systems are installed on the production drills and auxiliary drills to guide rigs into position, and assure holes are drilled to the correct depth and location.

16.5 Pit-Slope Monitoring and Voids Management

16.5.1 Pit-Slope Monitoring

Rock mass failure is not considered a high risk due to the high overall rock mass strength. However, slope movement monitoring will be installed in the open pit to gather measurements and confirm engineering assumptions to assure safe working conditions. Initial slope-movement monitoring will consist of using prisms read by manual or automated total stations. The initial prism monitoring will provide movement response data to verify visual observations, and that the slope is performing adequately.

Pit-wall mapping using routine digital techniques and physical geological mapping has also been implemented.

16.5.2 Voids Management

Underground void management is required to identify high-risk potential areas from historical underground stopes prior to mining, and then progressively mining through these areas. Each underground stope requires a detailed assessment to determine the best operating practices for safe mining. The hazards related to these historical underground workings include:

- Sudden and unexpected collapse of the open pit floor or walls
- The loss of people or equipment into unfilled or partially filled underground workings
- Loss of explosives from charged blastholes that have filled cavities connected to the blasthole
- Overcharging blastholes where explosives have filled cavities connected to the blasthole
- Risk of flyrock from cavities close to the pit floor and adjacent blastholes.

The historical underground workings are well documented and are not a concern for the overall stability of the final pit walls. For the pit designs, as far as is practical, the ramps will be kept away from the known historical underground openings, and larger berms will be designed to create access points around the bigger underground openings at different heights.

16.6 Mining Equipment

The type and number of major mining equipment for each production year are presented in Figure 16-18.

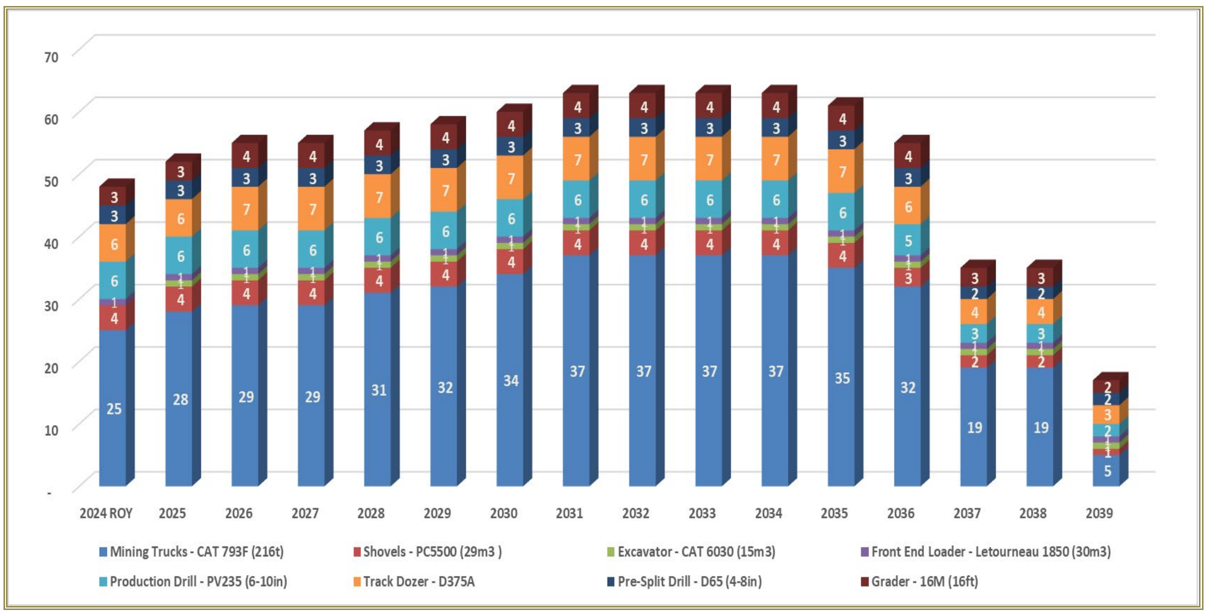


Figure 16-18: Mine Equipment LOM Requirements

The annual Mine equipment purchase schedule is shown in Table 16-12.

Table 16-12: Annual Mine Equipment Purchase Schedule

Equipment Purchase Schedule	Total	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2036	2037	2038	2039
Major Equipment																			
Haul Truck CAT 793F (216 t)	12	-	3	1	-	2	1	2	3	-	-	-	-	-	-	-	-	-	-
Production Shovel PC5500 (29 m³)	1	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Production Excavator CAT 6030 (15 m³)	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Front-End Loader L1850 (30 m³)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Production Drill PV235 (6-10 in)	5	3	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Track Dozer D375A-8	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Motor Grader CAT 16M (16 ft)	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Auxiliary Pre-split Drill D65 (4-8 in)	1	-	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Total	**22**	**4**	**6**	**3**	**0**	**3**	**1**	**2**	**3**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
Support Equipment																			
Excavator (49-t)	2	-	-	-	1	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Excavator (90-t)	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Hydraulic Hammers for Excavator 49 t	3	-	-	1	-	-	1	-	1	-	-	-	-	-	-	-	-	-	-
Utility Wheel Loader—(250 HP)	1	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Small Water Truck (Hook lift)	4	2	-	-	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-
Boom Truck (28 t crane)	2	1	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Knuckle Boom Truck (10 t)	1	-	-	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-
Telehandler	1	-	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Mechanic Service Truck	3	-	-	-	2	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Welding Truck	2	-	-	-	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-
Fuel/Lube Truck	4	2	-	-	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-
Lube Truck	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pickups	67	-	8	29	-	-	-	-	30	-	-	-	-	-	-	-	-	-	-

Equipment Purchase Schedule	Total	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2036	2037	2038	2039
Pit Busses	7	-	2	2	-	-	-	-	2	1	-	-	-	-	-	-	-	-	-
Mobile Air Compressor 375 CFM	2	-	-	1	-	-	-	-	1	-	-	-	-	-	-	-	-	-	-
Electric Welding Machine	2	-	-	-	-	-	2	-	-	-	-	-	-	-	-	-	-	-	-
Mobile Welding Machine	4	-	-	-	2	2	-	-	-	-	-	-	-	-	-	-	-	-	-
Lighting Towers	18	-	4	6	-	-	-	4	4	-	-	-	-	-	-	-	-	-	-
Genset 6 kW	3	-	-	-	3	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Genset 60 kW	4	-	2	-	-	-	1	1	-	-	-	-	-	-	-	-	-	-	-
Dewatering Pump—10 in	10	-	-	-	-	4	-	-	2	4	-	-	-	-	-	-	-	-	-
Dewatering Pump—6 in	4	-	1	1	-	-	-	-	2	-	-	-	-	-	-	-	-	-	-
Trash Pump 3 in	10	-	-	4	-	4	-	-	2	-	-	-	-	-	-	-	-	-	-
Diesel Powered Air Heaters	4	2	-	-	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-
Industrial Sweeper/Floor Cleaner	2	-	1	-	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-

16.7 Mine Workforce Requirements

Table 16-13 presents the mine operations workforce requirements for the LOM with a reduction occurring when production decreases in 2036. The mine workforce reaches a peak of 498 individuals during 2031.

Table 16-13: Annual Workforce Requirements

Department	2024 (ROY)	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039
Mine Operations	234	270	268	280	291	322	326	344	344	340	340	327	299	194	189	110
Mine Maintenance	82	92	92	94	106	110	110	110	110	110	110	94	78	53	51	30
Mine Geology	19	20	20	18	18	18	18	18	18	18	18	16	16	15	15	12
Mine Engineering	25	27	27	27	26	26	26	26	26	26	26	25	21	20	20	14
Total Workforce	360	409	407	419	441	476	480	498	498	494	494	462	414	282	275	166

Note: Mine engineering includes technical services for the TMF.
ROY = rest of year.

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 P_{80} 90 µm). The grinding circuit includes an HPGR, two identical ball mills, and two identical gravity concentrators. The mill operation schedule is 24 h/d, 365 d/a, 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. The key general process design criteria are presented in Table 17-1.

Table 17-1: Key General Process Design Criteria

Parameter	Unit	Value
Throughput—Design	t/a	9,855,000
Throughput—Design	t/d	27,000
Throughput—Design	t/h	1,223
Design Grind Size (P_{80})	µm	90
Crusher Utilization	%	67
Process Plant Availability	%	92
Operating Time	d/a	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 testwork program determined grinding characteristics for the various lithologies. Based on the ROM expected composition (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 percentile of hardness was 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 from 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 sample's BWI was 15.5 kWh/t, which was used for the design of the ball mills. Modified BWIs, considered to be more accurate for the prediction of the grinding circuit behaviour, 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, which confirmed that the design value of 15.5 kWh/t was satisfactory and conservative. Furthermore, the variability between the results obtained for each lithology was small, justifying the use of only one value of BWI to represent the entire deposit for the design of the ball mills. The HPGR will also generate 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	Unit	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)

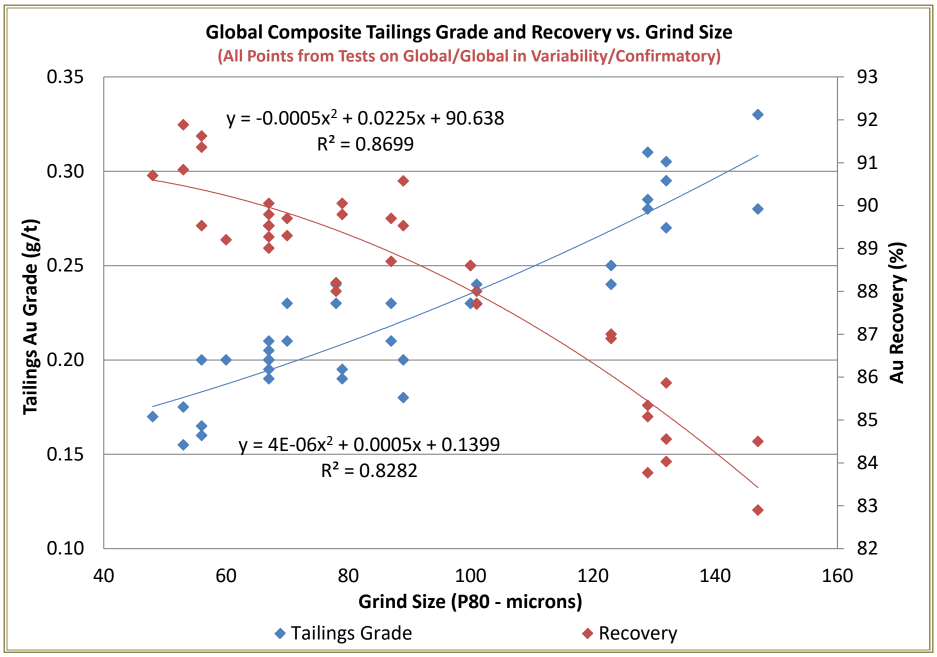
Comminution Parameters	Unit	Value
Bond Rod Work Index (RWI) (80th percentile)	kWh/t	16.8
Standard Bond Ball Mill Work Index (BWI) (75th percentile)	kWh/t	15.5
Modified Bond Mill Work Index (BWI) (75th percentile)	kWh/t	15.5
Abrasion Index (Ai) (average)	g	0.127
Unconfined Compressive Strength (UCS) (80th percentile)	MPa	242
Unconfined Compressive Strength (UCS) (range)	MPa	206 to 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 testwork established a correlation between grind size and gold recovery whereby a finer grind results in a higher recovery.

The global composite was considered to be the most representative of the ROM over the life-of-mine (LOM), and the results of the leach tests on the global composite were used to determine the optimal grind size (Figure 17-1).

The analysis was also conducted on the results from leach tests on 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 were used to evaluate the impact of ore variability.



Source: Soutex/WSP (2014, July 21).

Figure 17-1: Global Composite Tailings Grade and Recovery vs. Grind Size

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 was made between grind fineness and throughput, and a 90 µm grind was considered optimal. The lower recovery between 72 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 hence 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 a reduction in recovery. A multivariate linear regression analysis was used to determine the correlation between leach residue grade and mineralogical composition. Multivariate regression models made it possible to describe how one variable (response) reacts to simultaneous changes in other variables (predictors). The method enabled the combined impact of each predictor variable to be quantified on the response variable, which was not possible via simple regression analysis.

The results of the leach tests conducted during the feasibility study stage, described in Section 13.2.2, and the basic engineering stage, described in Section 13.2.5, 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 sulphur head sample grades. The strong correlation between the residual gold grade and arsenic and sulphur 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. Composites A to I and WCE were produced for the feasibility study testwork completed in 2014. 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 data set for the model since it would lead to bias.

Table 17-4: Composite for Multivariate Analysis

Composite	No. 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 due to 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 sulphur head grades on the residual gold grade was modelled via the following multivariate linear regression equation:

$$Au_{Tailings(g/t)} = -0.0435 + 0.0349Au_{Head(g/t)} + 0.660As_{Head(\%)} + 0.0312S_{Head(\%)} + 0.000516P_{80(\mu m)}$$

The limits of application of the model in terms of arsenic and sulphur values are set based on the range of arsenic and sulphur 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 arsenic and sulphur value is calculated using the minimum ratio and the head gold 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

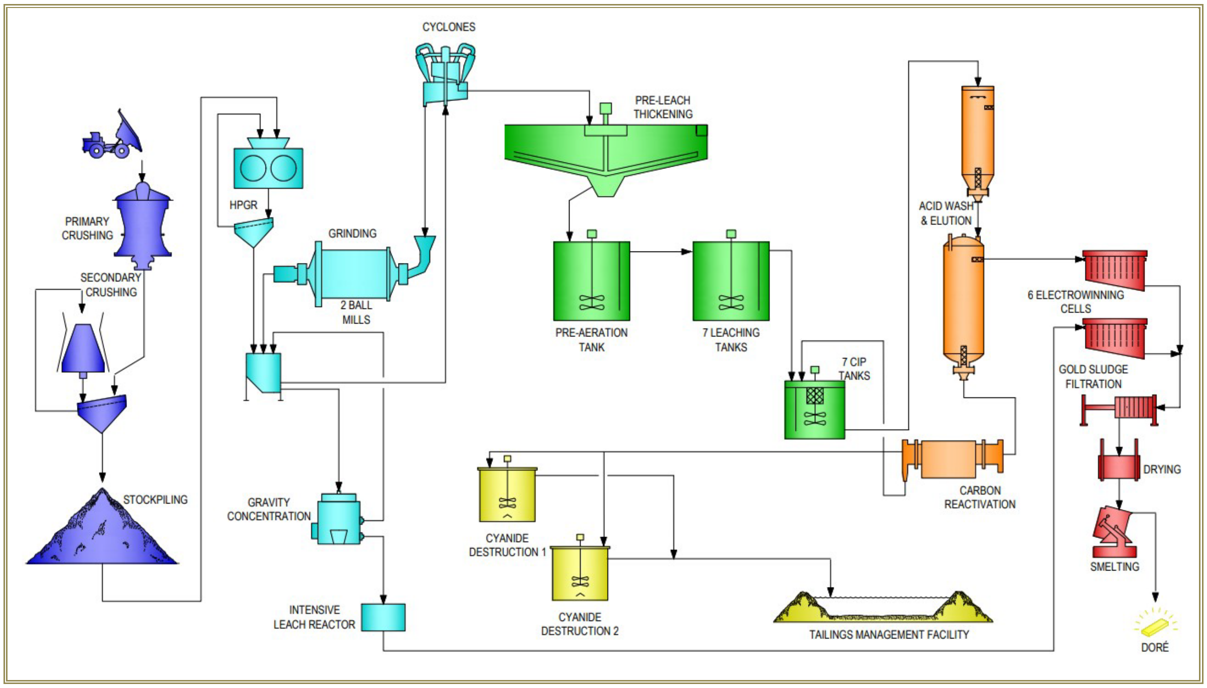
This formula has been used in the block model and open pit optimization process to calculate the gold recovery of each individual block based on chemical composition and grind size. The overall gold recovery can then be computed for a determined time period.

17.2 Flowsheet and Process Description

The gold recovery process for the Greenstone Mine consists of a crushing circuit (primary gyratory and secondary cone); an HPGR and ball mill grinding circuit with gravity recovery; pre-leach thickening; cyanide leaching; CIP adsorption; elution and regeneration; 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 on-site sewage treatment plant processes domestic wastewater, discharging to the environment. Tailings 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 shown in Figure 17-2.



Source: Soutex/WSP (2016, January 11).

Figure 17-2: Process 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 ROM 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.

Primary Crushing

ROM ore is delivered by mine haulage trucks, then 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 is used to break oversized rocks. The 1,300 x 1,800 mm 450 kW gyratory crusher reduces the ore from a 1,200 mm top size (P_{80} 275 mm) to P_{80} 140 to 160 mm. The primary crusher operates with an open-side setting of 165 to 200 mm. The crushed ore falls into a steel discharge pocket with a 1.6 times truckload-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-tonne 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 so 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 used for all of the tramp removal systems.

Secondary Crushing and Screening

The secondary crusher is a 950 kW standard cone crusher with a 45 to 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 x 8,500 mm double-deck secondary crushing screen is fitted with a top deck 75 x 100 mm closing screen opening, and a bottom deck with 50 x 100 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 P_{80} 35 mm. 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 is installed 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 metal into a separate chute. These measures prevent tramp metal 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 tonne-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 100 tonne-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.

Crushed Ore Stockpile and Reclaim

The crushing circuit product is stored in an 80,884-tonne total-capacity stockpile which provides 66 hours of operation, including 21,196 tonnes of live capacity for 17 hours of operation. The stockpile is located north of the secondary crushing circuit and the process plant. There is a single stockpile-reclaim tunnel with three apron feeders 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 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 and 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 metal still present.

17.2.2 HPGR/Grinding and Gravity Recovery Circuit

The HPGR/grinding circuit crushes and grinds ore to the optimal size to maximize gold recovery in the leach and CIP circuit. The gravity recovery circuit removes gravity-recoverable gold in the grinding circuit 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, and tanks. The design factor accounts for the process fluctuations and upsets, ensuring that mill throughput is met.

High-Pressure Grinding Rolls

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 to ensure that the weight bin choke-feeds the HPGR. The HPGR is equipped with two 2580 kW motors for a total of 5160 kW. The HPGR roll dimensions are 2.2 m diameter by 2.0 m long, and have a rotating speed of 22 rpm. The HPGR discharge falls onto the wet-screen feed-bin conveyor, 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 x 28 mm apertures, and the bottom screen panels have 6 x 27 mm apertures. The screen undersize is about P_{80} 2.7 mm, which falls into its respective ball mill pump-box.

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 100-tonne capacity overhead crane is installed to service the HPGR and the secondary screen in the HPGR building, and a 5-tonne capacity overhead crane services the COS reclaim-conveyor head pulley, electromagnet, and HPGR feed bin. A sump pump is also installed in the HPGR and secondary-screen area. The wet-screen-oversize return conveyor is 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-tonne overhead crane are installed to service the HPGR wet screens area and grinding area.

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.

The cyclone overflow is at P_{80} 90 µm. The ball mill 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 totalling 10,500 kW per mill. Both mills are 6.7 m diameter (inside liners) with 12.2 m effective grinding 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 line and slowing the HPGR rpm to match the throughput.

A single liner-handler can be used for the liner changes in either ball mill. Each mill has its own ball-kibble lifting frame and inching drive. Each ball kibble is loaded with 65 mm-diameter balls. 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.

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 pump-box. Screen undersize flows to the gravity feed chute onto each gravity concentrator. A bypass 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 gravity concentrators and transferred to the intensive-leach reactor (ILR). The gravity concentrator rejects stream flows 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-tonne overhead crane is installed to service the gravity screens, with each gravity concentrator having a dedicated monorail.

Intensive Leach Reactor

The gravity concentrate from both gravity concentrators is transferred to the single intensive-leach reactor. This equipment 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 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.

Pre-Leach Thickening

The cyclone overflow from each ball mill circuit feeds a dedicated trash screen above the pre-leach thickener. The screen-undersize gravity-flows 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 outdoors; the tank underside is cladded and heated. The thickener increases the slurry density from 31% to 55% solids, and recycles water to the grinding circuit. To promote sedimentation, 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). Thickener underflow is pumped to the leach circuit and sampled prior to leaching.

Leach Circuit

The leach circuit consists of one agitated pre-aeration tank and seven agitated leach tanks, all of which are located outdoors. All tanks are 18.6 m diameter x 23.5 m high, 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 30.3 hours, and an additional 4.3 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, and in the second, third, and fifth leach tank to readjust the pH value between 10.5 and 11.0, as required. A 23% sodium cyanide solution is added to Leach Tanks 2, 3, and 4 to ensure an initial concentration of 0.4 g/L NaCN, 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 18 mg/L concentration of dissolved oxygen. The leach discharge flows by gravity to the CIP circuit.

Leach tanks and associated equipment are serviced by a mobile crane, as required. Drive-through access is provided along both sides of the leach containment area, which is hydraulically linked with the pre-leach thickener containment area.

Carbon-in-Pulp

The leach circuit discharge flows into the CIP launder 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 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 diameter by 12.0 m high). 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 adjusted at 33 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 standby).

Each day, the lead CIP tank is taken off line, and the tank contents are 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 12 tonnes.

The loaded-carbon screen is 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 then 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 TMF.

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 CN_{WAD} concentration in the tailings' slurry to less than 50 mg/L to reach the ICMI code end-of-pipe requirement. Liquid sulphur dioxide and gaseous oxygen are injected at the bottom of the reactors and are 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 2 mg/L. Copper sulphate is used as a catalyst for the reaction and is added to the slurry as a 12% copper solution. About 20 mg/L of copper ions are required nominally (40 mg/L design). Lime is added to control the pH of the reaction to approximately 8.5. Sulphur dioxide is added at a ratio between 4.5:1 and 5:1. The CN_{WAD} concentration is measured in the feed and also in each reactor to ensure the target of 50 mg/L is achieved by maintaining the concentration between 30 and 50 mg/L. The residence time required to achieve the desired reaction is 60 minutes. If one tank is offline, the time required for the reaction can still be met with the 69 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.

Final Tailings

The cyanide destruction reactors overflow to the tailings pump box. The pump box also collects various tailings streams from the process, including sump pumps. Reclaim water is added to the pump box when the plant throughput is low to maintain adequate fluid velocity in the tailings pipeline.

The tailings pump box discharge is pumped by two parallel pump-trains of two pumps installed in series (one train in operation and one train on standby) pumping the tailings to the TMF via a 28-inch diameter HDPE pipeline. The pipeline is separated into two sections to fulfill the pressure requirements, with two pumps in series at the process plant. A spigotting system distributes tailings into the TMF.

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-tonne capacity.

Carbon Acid Wash and Elution

Loaded carbon from the CIP circuit is pumped to the loaded-carbon screen 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, on 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 solution (0.1% NaCN) is prepared in the elution-solution tank and is circulated through the column to strip the carbon at a temperature of 140°C. Solution is heated through a natural-gas heater. A carbon elution cycle is completed within eight to twelve hours.

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 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 a 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-tonne 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 two 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.

Electrowinning

Pregnant solution from the gravity concentrate-leaching is pumped to its dedicated electrolyte tank 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 pump box 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.

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.

Compressed Air

The compressed-air system is composed of three air compressors (two operating, one on standby). 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 various equipment requiring compressed air (filters, vents, dust suppression systems, etc.).

Oxygen (O_2)

The oxygen requirements for leaching and cyanide destruction are met by a vacuum-pressure swing-adsorption (VPSA) plant. The VPSA plant is installed outside the plant, adjacent to the leach tanks. Two liquid-oxygen tanks (total 100 tonnes or 4 days consumption at design rate) are also installed on site as a back-up source of oxygen.

Cyanide (NaCN)

Cyanide is delivered in isotainers (or equivalent) containing 18 tonnes of solid cyanide 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 to produce a 23% w/v cyanide 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 standby). Cyanide solution is distributed to the gravity-concentrate leaching circuit, the leach circuit, and the elution circuit.

Caustic (NaOH)

Caustic is delivered by 28-tonne bulk tanker truck, in liquid form at a 50% solution concentration. The solution is transferred to the storage tank, which can hold approximately one-and-a-half tanker trucks. The live storage volume is 30 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.

Quicklime (CaO)

Pulverized quicklime is received in bulk. It is transferred to a 298 m^3 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 and 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 pump box where it is further diluted to produce 23% lime slurry. The lime pump box is equipped with two transfer pumps (one operating, and one standby). The transfer pump pumps to a storage tank from which there are two distribution pumps (one operating, and one standby). 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.

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 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 standby).

Hydrochloric Acid (HCl)

Hydrochloric acid is delivered by 22-tonne bulk tanker trucks as a 33% concentrated solution. It is unloaded to a storage tank that has the capacity to hold a one-and-a-half truck deliveries, 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.

Copper Sulphate ($CuSO_4 \cdot 5H_2O$)

Copper sulphate pentahydrate is received in 500 kg bulk bags. The 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 12% w/w concentration solution. The copper sulphate storage tank is 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 21 m^3, which is about 23 hours storage at 27,000 t/d plant throughput. Two distribution pumps are installed (one operating, and one standby) to transfer copper sulphate to its addition point in the cyanide detoxification distribution-box.

Sulphur Dioxide (SO_2)

Liquid sulphur dioxide is delivered by a 30-tonne bulk tanker-truck. It is transferred to two 50 tonne-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 eight 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.

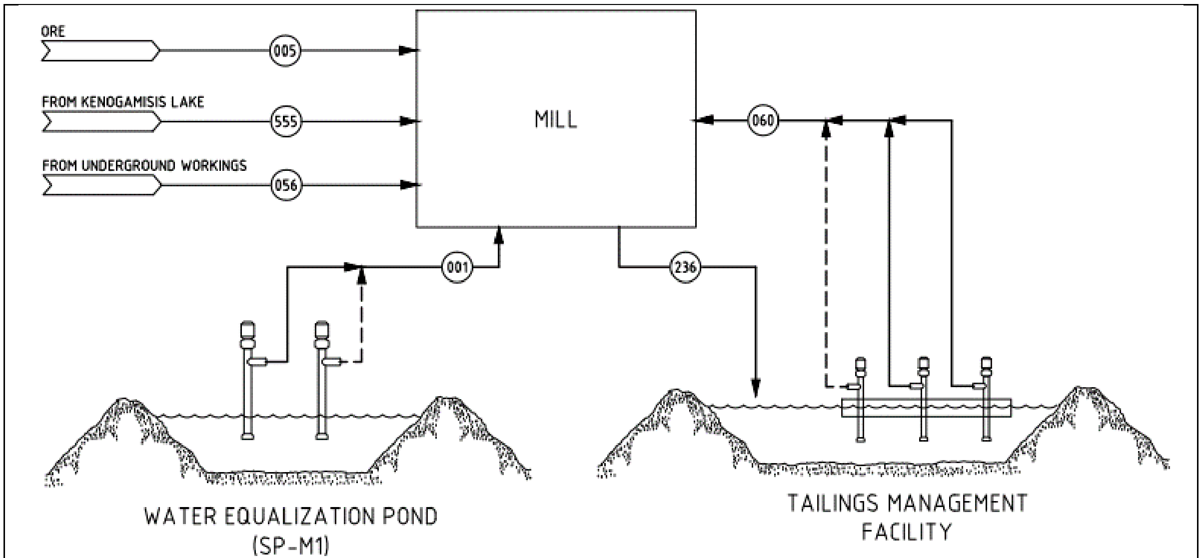
Antiscalant

Antiscalant is received in a liquid form in 1 m^3 ISO containers. Containers will be directly fitted with pumping systems from where the antiscalant 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 for fresh water will come from the underground workings collected in the water-equalization Pond M1. No water is planned to be withdrawn from Kenogamisis Lake to supply the process plant. 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, which contains a significant amount of water. Figure 17-3 outlines the water balance results calculated for the process plant.



Stream No.	001	005	056	060	236	555
Solids (t/h)	0.0	1,222.8	0.0	0.0	1,227.7	0.0
Solution (t/h)	200.0	37.8	8.6	797.5	1,038.0	0.0
Slurry (t/h)	200.0	1,260.6	8.6	797.5	2,260.7	0.0
% Solids	0.0	97.0	0.0	0.0	54.1	0.0
SG Slurry	1.00	2.66	1.00	1.00	1.53	1.00
Slurry (m^3/h)	200.0	473.4	8.6	797.5	1,473.5	0.0

Source: Souther/WSP (2016, January 29).

Figure 17-3: Process Plant Water Balance

17.4 Power Requirements

The power requirement for the process plant was determined 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 (34 MWh/h) at 27,000 t/d.

17.5 Process Plant Layout

17.5.1 Location

The process plant site is located southwest of the open pit. The site main entrance is on the southwest 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 infrastructure in consideration of the prevailing winds.

17.5.2 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 13 make-up air units (five for the process plant; seven for the crushing, COS, and HPGR building; and one for the ore reclaim tunnel).

A primary/secondary 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:

- 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.

17.5.3 Fire Protection

The fire-protection water reserve is 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 are 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 are installed throughout the plant. The electrical room is protected by an inert gas system. For unheated conveyor galleries, fire protection is provided by dry-pipe sprinkler system.

17.5.4 Electrical Distribution

Power to the process plant is supplied from the on-site power plant at 13.8 kV.

The power plant consists 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, with one unit on hot standby and one unit in maintenance. The availability of power plant will be 100%, excluding a major fault at the 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 step-down transformers installed inside modular, prefabricated and pre-assembled electrical rooms (E-room). The E-rooms house the medium- and low-voltage MCCs, lighting & services transformers, instrumentation transformers, VFDs, soft-starters, and so forth, to distribute power to the dry-end equipment, including crushers and HPGR, ball mills, process plant, and so forth.

The power plant also supplies power to outlying pumping stations, shops, and service buildings via the 13.8 kV overhead power lines.

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 the power plant for grounding detection and coordination.

17.5.5 Control System

The process control system is a dedicated, microprocessor-based, scalable, deterministic, control system with the ability to monitor and control centrally and remotely, and is a PLC/SCA$A solution and referred to as the process control system (PCS). Touch screens are used for local operator interface (HMI).

18 PROJECT INFRASTRUCTURE

This section describes the infrastructure and service facilities required to support the Greenstone Mine mining and processing facilities, including: tailings and water management; power generation; roads; camp; mine equipment maintenance shops (truck shop); warehouses; communication systems; laboratories and offices; diesel fuel, and the natural gas distribution pipeline.

18.1 Tailings Management

WSP (and its predecessor companies WOOD and Golder Associates) performed specialized geotechnical and hydrologic engineering services for the design of the TMF, including geotechnical foundation investigation programs, design of the tailings dams and ancillary hydraulic structures, and tailings deposition planning. Since 2022, WSP has been responsible for the ongoing design and construction quality assurance (CQA) and Engineer of Record (EOR) services during the staged construction of the TMF.

Tailings impoundment is provided by the construction of dams with a final maximum height of 35 m and crest length of about 7,400 m. The dams are being constructed primarily using waste rock from mining operations. The upstream slope of the dams comprises a low permeability compacted glacial till core keyed into low permeability foundation soils. In places, a Deep Soil Mixing (DSM) seepage cutoff wall has been constructed to connect the core into the low permeability foundation soils. The till core, key trench and DSM wall serve as a low permeability element to mitigate seepage through the dams and their foundations. A seepage collection system is provided at the downstream toes of the dams to collect any seepage for pumping back into the TMF reservoir. The TMF dams will be periodically raised using downstream construction methodology and the core, internal filter zones and rockfill sections of the dams will be extended with each expansion.

18.1.1 Geotechnical Subsurface Investigations

Environmental and hydrogeological baseline investigations were initially carried out in the TMF area by Stantec. Subsequently, WOOD carried out the subsurface characterization of the TMF dam footprint with geotechnical investigations from 2014 to 2019. The investigations also included searches for sand and gravel filter material and glacial till core required for the initial and future dam construction. Several additional geotechnical investigations have been carried out by WSP since construction began to support construction activities and detailed design of future raises of the TMF dams.

The geotechnical investigations have included the excavation of numerous test pits and the drilling of numerous boreholes. Soil samples were collected from boreholes and test pits and were tested in a geotechnical laboratory. In-situ hydraulic conductivity testing of foundation soils were performed. Piezometers/monitoring wells were installed in the boreholes at various depths to facilitate groundwater monitoring and testing. The groundwater table was found to vary across the TMF area from near surface to about 3.5 m below ground surface. Bedrock encountered was generally good to excellent quality based on RQD measurements.

The Quaternary stratigraphy at the TMF consists of five 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 (predominantly silt with minor clay), deltaic (sands) and recent deposits (aeolian, organics, fluvial). Isolated pockets of sand and gravel (possibly washed glacial till) were also encountered below the glacial till in the Southwest Dam. A glaciolacustrine deposits of silt interbedded with thin clay layers (typically 1 mm to 10 mm in thickness) (referred to as 'GL Silt') was generally encountered underlying the upper sand deposits in the low-lying areas of the foundations at all dam sections, except at the North Dam.

18.1.2 Design Criteria

The TMF is currently designed to receive approximately 145 Million tonnes (Mt) of mill and historical tailings at an average dry density of 1.34 t/m^3. A cyanide destruction system is used to process all tailings water before it is sent to the TMF. An allowance has been made within the TMF to store historical tailings and contaminated soils which are being relocated from the open pit area.

In accordance with the *Lakes and Rivers Improvement Act* (LRIA) Hazard Potential Classification (HPC), the TMF 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.

Dam design criteria includes 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).

18.1.3 Dam Design

The cross-section design for the TMF dams consists of an inclined upstream low permeability glacial till core, with a filter and transition layers downstream of the core, and a downstream rockfill shell. The filter and transition zones prevent the migration of core material into the downstream shell. A blanket filter was constructed between the foundation soils and the rockfill embankment to protect against potential "piping" of foundation soil fines into the rockfill due to seepage forces. A non-woven geotextile and riprap erosion protection were placed on the exposed upstream side of the core to protect it from surface water erosion due to waves or run-off until tailings beaches are established against the slope of the dams.

The TMF dams will be constructed in stages, with annual dam raises in the downstream direction. Construction of the TMF starter dams was completed in 2023. The first (Stage 1) dam raise will be completed in 2024 to a crest elevation of 344 m, and the planned ultimate crest elevation will be 365 m.

The in-situ testing and the advanced laboratory testing programs have indicated that parts of the GL Silt deposit are potentially liquefiable. Out of conservatism, post-liquefied stability analyses were carried out for all dam sections where the GL Silt was present. In the starter dam stage, this required the placement of upstream rockfill stability berms pending the establishment of upstream tailings beaches. For the starter dams and the Stage 1 dams raise, downstream rockfill stability berms were constructed within the ultimate dam footprint. For 2025 and subsequent dam raises, it will likely be necessary to undertake ground improvement measures to obtain the target downstream Factors of Safety for the presumed post-liquefaction condition. The ground improvement measures could include construction of shear keys (i.e., trenches excavated through the GL Silt and backfilled with mine rock) or possibly arrays of DSM under the downstream dam toes in the critical sections.

An Operations Management and Surveillance (OMS) Manual following the guidelines of the Mining Association of Canada has been put in place for the TMF. An extensive system of geotechnical instrumentation has been installed to monitor the performance of the TMF dams, including vibrating wire piezometers to measure pore water pressure within the dams and foundations, inclinometers to measure dam deformations, and magnetic settlement systems to monitor settlement within the dams.

18.1.4 Tailings Characteristics

Tailings geochemistry indicates that less than 10% of the ore is considered potentially acid generating (PAG). This amount will be reduced through oxidization during ore processing, thereby reducing the overall acid rock drainage (ARD) potential for the tailings.

18.1.5 Tailings Deposition Plan

Tailings are deposited in the TMF from the dam crests as a conventional slurry to produce a wide exposed beach. This beach will displace the tailings pond away from the dams towards natural ground along the northern edge of the facility to enhance long term dam stability. A barge mounted pump system, located near the north side of the TMF, reclaims water from the TMF pond and pumps back to the mill.

Figure 18-1 shows the general arrangement of the TMF and the final tailings and pond surfaces.

18.1.6 Water Management

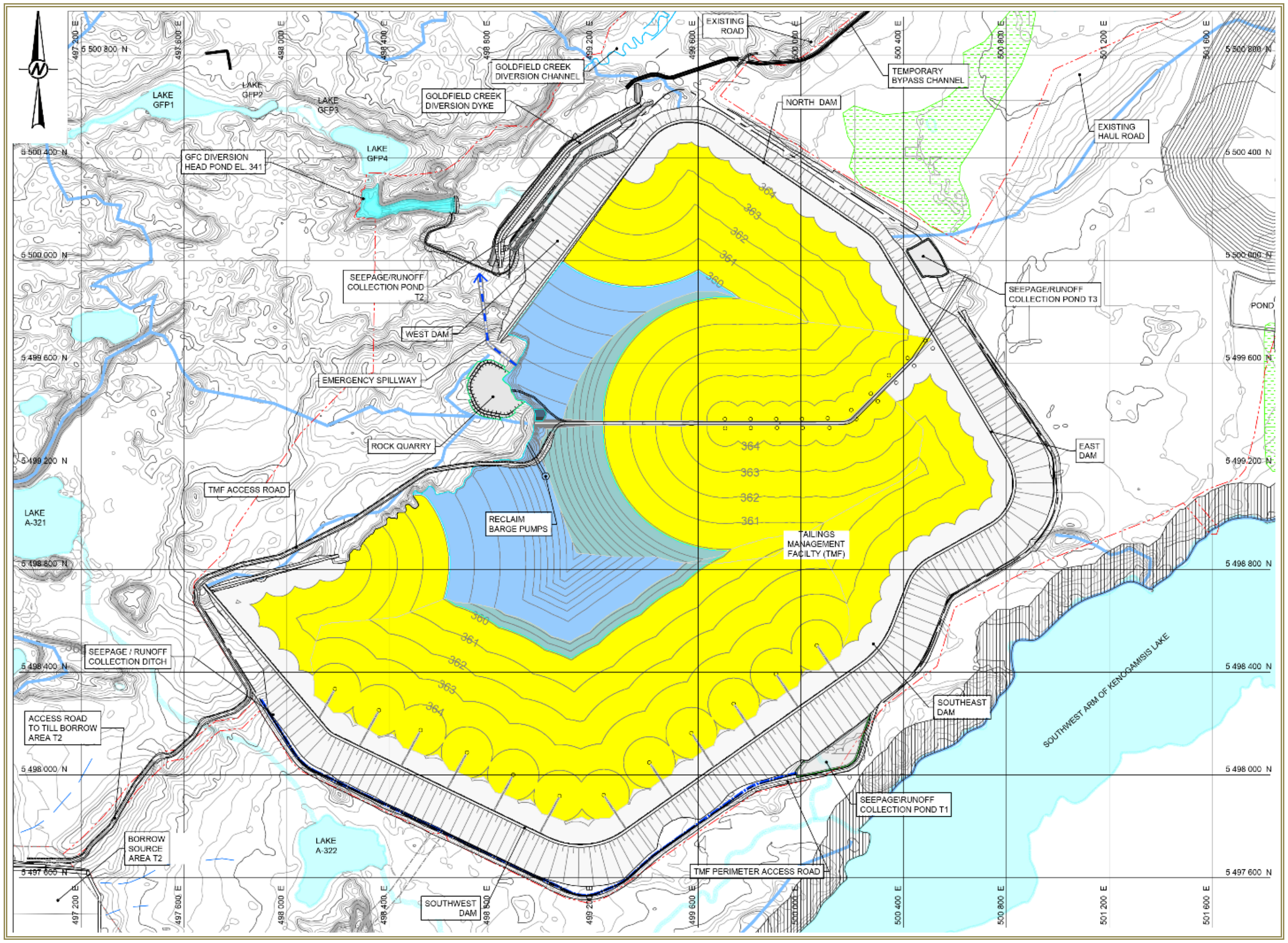
The Greenstone site and TMF have a positive water balance. Reclaim water from the TMF is expected to supply the majority of the mill makeup water for most of the years of operation. The balance of mill make-up water will be sourced from other ponds within the project area such as waste rock storage area ponds, and the historical underground workings. The TMF's emergency spillway invert levels will be established at the high-water level corresponding to the EDF event at all stages of operation to ensure sufficient storage capacity to contain the EDF without any discharge to the environment.

A TMF emergency spillway is currently located between the North and East Dams. In 2026, the TMF emergency spillway will be relocated to the west abutment of the West Dam such that any emergency overflows will be directed towards the Goldfield Creek diversion head pond. After closure, once the quality of the pond water is acceptable to discharge, the spillway invert will be lowered and the TMF pond will discharge by gravity through the final spillway.

18.1.7 Seepage Mitigation and Control

Seepage mitigation measures have been included in the TMF design and construction. At the Southwest Dam, a contiguous Deep Soil Mixing (DSM) wall was constructed to penetrate through the upper sand and silts and to key into the underlying, low permeability GL Silt layer. A short section of DSM wall was also constructed under part of the East Dam. At all other dam locations, a key trench was excavated to connect the low permeability dam core into a low permeability soil formation (i.e., the glacial till or GL Silt) or directly onto a prepared bedrock surface. The key trench was backfilled with compacted, low permeability glacial till.

A system of perimeter seepage collection ditches and ponds was constructed downstream of the TMF dams to capture foundation seepage and surface water runoff from the external dam slopes. These perimeter ditches drain to three seepage collection ponds, and any collected seepage in these ponds is pumped back into the TMF.



Source: WSP (2024).

Figure 18-1: General Arrangement of Tailings Management Facility

18.2 Goldfield Creek Diversion

Goldfield Creek (GFC), a small watercourse that previously drained through the TMF footprint and into Kenogamisis Lake, was rerouted into the Goldfield Creek diversion system for construction of the initial TMF. During the 2023 spring freshet, significant erosion of the realigned channel occurred, resulting in the transport and deposition of sediment downstream of the constructed channel. Subsequently, a temporary bypass channel was constructed in 2024 to bypass flow around the eroded Goldfield Creek diversion while the realigned channel is redesigned and remediated. The temporary bypass channel is regarded as a temporary feature, and will remain in active service until Goldfield Creek has been restored, vegetated, and considered stable to receive flow.

18.2.1 Design Criteria

The GFC diversion channel and temporary bypass channel were designed to pass a minimum of the 500-year return hydrologic event. The Goldfield Creek diversion pond and diversion dyke were designed to convey the PMF event without overtopping. The diversion channel was also designed and constructed to carry the Probable Maximum Flood (PMF) from the catchment area by flooding along the banks.

18.2.2 Diversion Dyke Design

The GFC diversion dyke has been designed and constructed in accordance with the Lakes and Rivers Improvement Act (LRIA) and Canadian Dam Association (CDA) guidelines. The GFC Diversion Dyke was designed considering a “Very High” hazard potential, consequently, the dyke was designed and constructed for the most severe flood criteria, being the PMF.

The diversion dyke comprises a central core of glacial till with upstream and downstream shells of sand and gravel. An inclined chimney filter of sand is downstream of the core and a blanket filter of sand covers the foundation soil downstream of the core. Erosion protection underlain by geotextile was applied to both the upstream and downstream faces. Due to foundation soil difficulty during construction, the dyke was built without a seepage cutoff wall, but a seepage reduction key trench was excavated beneath the core and backfilled with compacted glacial till and extends to a variable depth (typically at least 0.5 m) below the foundation. In areas where bedrock was encountered in the key trench, the bedrock was cleaned and slushed grouted; in areas where the foundation glacial till was shallow, the dam was keyed into the till.

As expected, clean water from the Goldfield Creek head pond seeps under the diversion dyke into the adjacent seepage collection pond, which lies between the diversion dyke and the TMF West Dam. Water within the pond is pumped into the TMF, and the pumping keeps the water level in the pond well below that in the head pond, which precludes any contaminated seepage from the West Dam from entering the diversion system. Field pumping measurements have indicated that the volume of seepage under the diversion dyke is about 415 m^3/d.

18.2.3 Closure Considerations

The Goldfield Creek diversion is a permanent realignment of the Goldfield Creek system, and as such will remain in place after mine closure. Once the water quality of the tailings pond is deemed suitable for discharge to the environment, runoff from the TMF will be directed through the overflow spillway into the Goldfield Creek diversion head pond. Reclamation of the temporary bypass channel will occur during operations and will involve backfilling and regrading of the channel to pre-construction grades.

18.3 Historical Tailings

The Mine site includes the historical MacLeod high and low tailings and Hardrock tailings which were deposited on land and into Kenogamisis Lake during historical mining activities in the 1930s through 1970s. Relocation of portions of these historical tailings from the open pit area to the TMF has been incorporated into the Mine design. The TMF has been sized to store up to 8.3 Mt of historical tailings, waste rock (for access road construction) and contaminated soil.

Relocation of historical MacLeod tailings commenced in January 2024 and is ongoing. Tailings removal is being completed using a combination of dozers and excavators and loading of Mine haul trucks for transportation and disposal into the northwest portion of the TMF. Prior to placement of these tailings in the TMF, waste rock is used, as needed, to construct access roads within the TMF to allow the haul trucks to ingress the designated areas for deposition. Once deposited, the historical tailings will be covered with new tailings in subsequent years of operations.

18.4 Water Management

18.4.1 Administrative Water Services

The mine site and camp draw water from a freshwater intake in the Southwest Arm of Kenogamisis Lake. Under an MECP permit to take water, the water is pumped to a small water treatment plant located within the process plant area to provide potable water and service water to the buildings.

The wash bay water at the truck shop is provided from the treated discharge of the effluent treatment plant. The wash bay is equipped with a filtering system to recirculate most of the water to the wash bay. Sludge is disposed through the process plant tailings management system.

18.4.2 Collection Ponds

Several collection ponds were excavated in key areas within the mine site to collect any seepage and runoff by gravity flow, 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, receives pumped flows from the other six remote collection ponds and open pit.

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 EDF (24-hour duration) 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: Collection Ponds—Existing

Aspect	Pond M1*		Pond A2*	Pond B1*	Pond B2*
Catchment Area and Size	Open Pit WRSA-C [81.9 ha]	Open Pit WRSA-C (south) [107.8 ha]	WRSA-A [61.5 ha]	Plant Site, Ore Stockpile [51.1 ha]	WRSA-B [31.2 ha]
Maximum Operating Volume (m^3)	50,000	43,000	8,500	19,000	8,000
EDF Runoff Volume (m^3)	4,000	103,000	34,000	37,000	15,000
Volume at Emergency Spillway Invert (m^3)	54,000	146,000	42,500	56,000	23,000

Table 18-2 summarizes the collection ponds to be constructed in the future and prior to the development of the waste dumps and overburden stockpiles.

Table 18-2: Collection Ponds—Future

Aspect	Pond A1	Pond C1	Pond D1	Pond D2
Catchment Area Size	Overburden Stockpile [75.4 ha]	WRSA-C (north) [38.3 ha]	WRSA-D (north) [134.1 ha]	WRSA-D (south) [82.3 ha]
Maximum Operating Volume (m^3)	14,500	5,500	19,000	11,000
EDF Runoff Volume (m^3)	36,000	37,000	128,000	79,000
Volume at Emergency Spillway Invert (m^3)	50,500	42,500	147,000	90,000

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.

18.4.3 Effluent Treatment Plant

The effluent treatment plant provides treatment of contact water from waste rock, ore, and overburden stockpiles, as well as from the historical underground workings and the open pit dewatering. The treatment reduces contaminants from the influent flow, such as total suspended solids (TSS) and metals. The plant is designed to be capable of managing a minimum treatment flow of 1,900,800 L/d (22 L/s), typical flow of 12,182,400 L/d (141 L/s), and a maximum flow of 19,008,000 L/d (220 L/s), at influent TSS levels between 200 mg/L to 2,000 mg/L and at dissolved arsenic raw water quality of up to 0.115 mg/L. The effluent treatment plant can be operated at a maximum flow rate of 19,008,000 L/d (220 L/s).

The effluent treatment plant has three steps in the water treatment train: TSS and metal removal, final filtration and neutralization, and potential future ammonia treatment. If ammonia concentrations are a concern in the future, a moving bed biofilm reactor or equivalent technology can be added to the treatment train upstream of the Actiflo clarifier. Ammonia is primarily from blasting residuals.

Sludge collected from metal and arsenic removal is sent to a geotextile sock for dewatering. The geotextile socks will be deposited in the TMF.

18.5 Power Supply and Distribution

Power is generated on-site by a natural gas-fueled power plant, consisting of a combined heat and power. The designed capacity of the plant is 48 MW and the average demand is 35.2 MW, for an operating load of 73% to 80%. Heat recovery is estimated at 5 MW thermal and is used for heating buildings.

The power plant has a total of six, 9.6 MW generators having an output voltage of 13.8 kV at 60 Hz. Five generators provide enough power to meet peak and average power demand, with one unit on standby and to facilitate maintenance. A seventh unit has been purchased and will be installed in 2025 to provide an additional backup. The plant has a black start generator to provide power needed to start the plant after an outage. Backup power, in the event of an outage (planned or unplanned), is supplied or supplemented by portable generators.

Annual natural gas requirements for the operation of the power plant are estimated at 1,450,000 gigajoules (GJ) during ramp up, and 2,800,000 GJ at peak production. Gas is supplied from the existing TC Energy mainline pipeline located approximately 12 km north of the Mine, and the natural gas is brought to the Mine via a new distribution pipeline.

Power is distributed to facilities further afield, such as the TMF and dewatering points, via an overhead 13.8 kV distribution line network. For the closer process facilities and support facilities, the power supply is stepped down to 4160 V and 600 V and distributed by power line, cable trays or underground with buried utilities.

18.6 Other Infrastructure

18.6.1 Truck Maintenance Shop and Warehouse

The truck shop has eight heavy duty maintenance bays: four bays for servicing mining haul trucks, and the other bays used for smaller support equipment, large loader, drill rigs and wash bay. Three 30/5 t overhead cranes service the maintenance bays. Compressed air is provided throughout the maintenance facility. Lubricants are stored in a specific enclosed area and are distributed to four reel stations. The warehouse is used to store consumables and maintenance parts. Office space is provided within the warehouse.

18.6.2 Site Mixed Emulsion Plant

The site mixed emulsion (SME) plant is located along the haul road to the TMF.

18.6.3 Sewage Treatment Plant

The sewage treatment plant is located near the process plant. It treats sewage flows from the mine site offices, mine dry building and process plant. The plant is sized to handle 300 persons at any given time, based on a design flow rate of up to 250 L/person/d and for a total estimated sewage flow of 75,000 L/d, with a maximum discharge rate not exceeding 0.9 L/s. Treated sewage effluent is co-discharged with the treated mine effluent to the Southwest Arm of Kenogamisis Lake.

18.6.4 Fuel Supply Storage and Distribution

Fuel is stored at the process plant and the heavy duty truck fuel station near the pit to serve light vehicles and mine equipment, respectively. The process plant fuel station includes a double walled split tank (50,000 L diesel and 20,000 L gasoline). The heavy duty truck fuel station has three double walled, above ground 50,000 L diesel storage tanks and one double walled, above ground 50,000 L urea storage tank. Liquid urea is used as an additive with diesel fuel to reduce nitrous oxide emissions of the mine fleet.

18.6.5 Communications Network

A site-wide, fibre optic communications network has been installed for site communications. Cellular phone coverage is provided by a 100-metre tall tower and hosts all cellular, radio and mine dispatch systems.

18.6.6 Main Access Road

The site is accessed via Trans-Canada Highway 11, approximately 275 km northeast of Thunder Bay and 600 km west of Timmins/Matheson. A new site access road has been constructed off Trans-Canada Highway 11 to the administration, mineral processing, power generation, and shop facilities.

18.6.7 Assay Laboratory

Assaying requirements are divided amongst a third party offsite arrangement for sample preparation and gold assaying, an 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 Mine are outsourced.

18.6.8 Administration Building

The administration offices complex is located near the process plant building and has change rooms for both men and women, offices, conference rooms, lunchroom and first aid station.

18.6.9 Fire Protection

The process plant site fire protection system consists 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. 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. The fire protection system is fed from a dedicated fire pumping station with centrally controlled automatic fire detection and alarms.

18.6.10 Security

Access to the processing, power, and administration area are secured by a remotely operated vehicle gate, controlled by security guards on 24-hour duty. The processing facilities and truck shop are monitored by CCTV surveillance. There is 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 has an additional level of security.

18.6.11 Site Camp

The camp is located on GGM-owned land approximately 2.5 km north of the process plant site on Old Arena Road near the intersection with Michael Power Boulevard. The camp currently has an average occupancy of approximately 400 persons, with a peak of approximately 600 persons. The camp currently requires the collection and trucking of wastewater to a local disposal/treatment facility but will be connected to the upgraded municipal system in the future.

19 MARKET STUDIES AND CONTRACTS

GGM has a standard industry contract for the sale of gold doré. The gold markets are mature global markets with reputable smelters and refiners located throughout the world. Gold is a principal metal traded at spot prices for immediate delivery. GGM has existing contracts for the supply of major consumables, including diesel fuel, electricity, cyanide, and explosives, as well as for major equipment and construction services.

19.1 Doré Refining and Gold Bullion Sales

The current contract for refining doré from the Mine is as follows:

- 0.01% to 0.05% deduction
- 99.95% to 99.99% payable.

GGM has a gold bullion sales contract with Asahi Refining USA, Inc. The average selling cost (for refining and transportation) is $1.87/oz and is valid until March 31, 2026.

19.2 Contracts and Agreements

19.2.1 Diesel Fuel Supply Agreement

GGM has a tri-party agreement with Imperial Oil and Long Lake #58 First Nation for fuel supply, and this agreement is valid until August 31, 2027.

19.2.2 Power (Electrical) Supply Agreement

GGM owns and operates a 56 MWe (gross) power plant at the Mine that consisting of six Wärtsilä 20V34SG natural gas-powered generators and associated auxiliaries. GGM has a service agreement with Wärtsilä Canada Inc., for optimized maintenance and operational support services. The current energy service agreement is valid until November 17, 2028.

19.2.3 Cyanide Supply Agreement

GGM has an agreement with Cyanco for solid sodium cyanide supply. This agreement is valid until December 31, 2027, and has a one-year option to renew.

19.2.4 Explosives Supply Agreement

GGM has an agreement with Dyno Nobel Canada Inc. to supply explosives and services. The initial term of the contract is 60 months and is valid until August 22, 2028, with renewals for successive one-year periods.

19.2.5 Drill Equipment Service Agreement

GGM has an agreement with Epiroc Canada Inc. to perform maintenance and provide parts for the Epiroc-branded drill equipment. The current three-year agreement is valid until July 19, 2025.

19.2.6 Tailings Management Facility Construction Services

GGM has an agreement with Amik Ltd. (Cloutier Contracting) for the Stage 1 TMF dam expansion construction. The current agreement is valid until December 31, 2024.

19.2.7 Mining Equipment Support Services

GGM has a service support and parts agreement with Toromont Caterpillar, a division of Toromont Industries Ltd., for the Caterpillar-branded equipment. The current service and parts agreements are valid until December 31, 2024.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

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 all Mine phases:

- Federal environmental assessment decision statement received December 10, 2018, and amended on February 10, 2021
- Provincial environmental assessment notice of approval received March 12, 2019
- Closure Plan filed with Ministry of Mines (Mines) on March 30, 2021, with amendments filed December 29, 2023, and August 8, 2024
- Construction and operation phase permits and approvals
- Environmental issues that may impact the operation and closure of the Mine.

From 2013 to 2021, environmental baseline studies were conducted to support the federal and provincial environmental approvals process and permitting. The data were used to identify environmental constraints, siting, and layout of Mine infrastructure, as well as to consider design alternatives from an environmental management and approvals perspective. Data from these studies are being used to determine incremental changes and predict environmental effects associated with the Mine.

The Mine was subject to both federal and provincial environmental assessment (EA) (see Section 20.3.1). The final environmental impact statement/environmental assessment (EIS/EA) (Stantec, 2017) was completed in 2017 and provided to regulatory agencies, Indigenous communities, and the public for review and comment. Thirteen valued components (VC) supported the EIS/EA: 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. Mine interactions with the VCs were analyzed to determine potential environmental effects associated with Mine construction, operation, and closure phases. In addition to the VCs, the effects assessment also considered effects of the environment on the Mine, accidents and malfunction scenarios, and cumulative effects. The federal decision statement was issued on December 10, 2018, and the provincial notice of approval was issued on March 12, 2019. The EIS/EA was amended December 2019 to incorporate additional information, commitments, studies, reports, and modelling that were submitted after the formal submission of the EA in accordance with Condition 27 of the provincial notice of approval. The federal decision statement was amended on February 10, 2021, to accommodate minor design changes during detailed Mine design.

A series of Environmental Management and Monitoring Plans (EMMP) were developed and implemented for the Mine (refer to Section 20.5). The objective of the EMMPs is to provide appropriate and thorough processes of verifying predicted project effects and the effectiveness of mitigation measures. The EMMPs include an adaptive management process that will alert the management team of changing conditions, and assess the need for additional investigation when there is the potential to significantly affect the receiving environment—all aimed at planning appropriate mitigation, if it should ever be required. The EMMPs include commitments made in the EIS/EA, condition requirements in permit approvals, industry best-management practices, and policies and procedures implemented by GGM. The EMMPs are "living" documents as the Mine progresses.

The Closure Plan was approved by Mines on March 30, 2021, with amendments filed December 29, 2023, and August 8, 2024. The Plan details progressive rehabilitation, rehabilitation measures, monitoring, and expected site conditions following closure. A summary of the closure, decommissioning, and reclamation of the Mine is provided in Section 20.6.

As stated in the final EIS/EA, with the identified mitigation measures implemented, the Mine will not cause significant adverse environmental effects, including effects from accidents and malfunctions, effects of the environment on the Mine, and cumulative effects. GGM has completed slight modifications of Mine components compared to that presented in the final EIS/EA as detailed engineering advanced and construction was completed, which formed the basis for the final Mine plan used for this Technical Report. Permit amendments required for these engineering advances have been obtained.

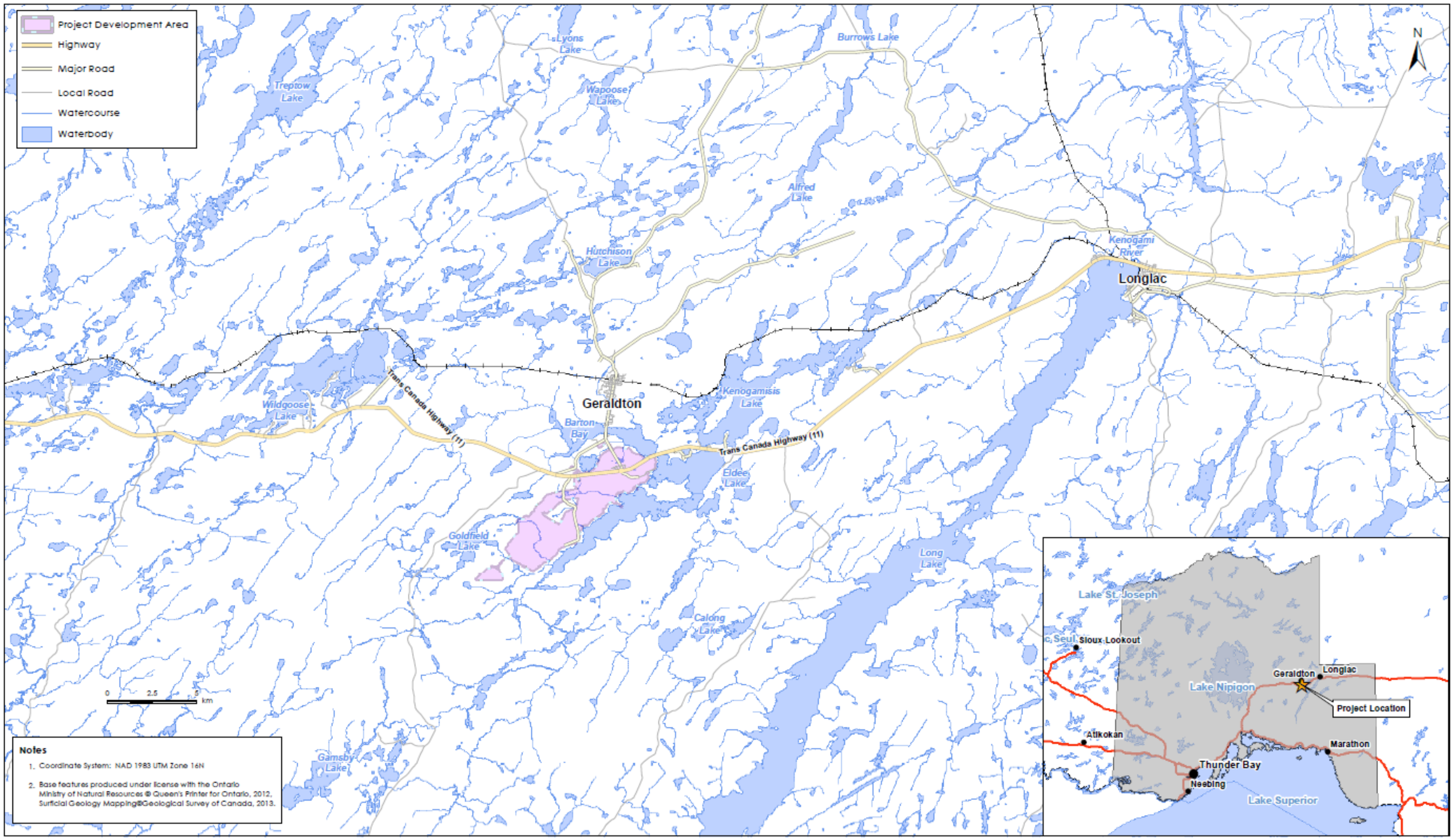
Consultation with stakeholders (community members, agencies, interested parties) and Indigenous communities is an integral part of the Project. Active participation through consultation helps to achieve an open and transparent process, build trust, enhance awareness of the Mine, and strengthen the quality of results. Consultation occurred throughout Mine planning, permitting, and Mine construction, and GGM is committed to maintaining stakeholder relationships through operations and into closure. Impact benefit agreements have been established with five Indigenous communities. Consultation and engagement activities are described in Section 20.4.

20.1 Environmental Studies

Baseline environmental studies were completed to characterize the natural, social, economic, cultural, and built environment that may have been impacted by the Mine or affect Mine design and approvals. Baseline studies for the Mine commenced in 2013. Tree clearing for Mine construction commenced in March 2021, with effluent discharge from the temporary effluent treatment plant (ETP) commencing on September 15, 2021. Therefore, the baseline conditions for the Mine are considered to have been those prior to March 2021.

During the EIS/EA process a Project development area (PDA) was identified (Figure 20-1) that encompasses the Mine footprint and the anticipated area of physical disturbance associated with Mine construction and Mine operation.

The next section summarizes the baseline studies completed from 2013 to 2021, and the long-term monitoring and physical and biological environmental settings of the Mine and surrounding area. The baseline studies were documented in Stantec (2017) unless otherwise noted.



Stantec	100-300 Hagey Boulevard, Waterloo, ON N2L 0A4	Project Development Area	Author: Brian Cowper Date: 24/09/16

Figure 20-1: Location of the Greenstone Gold Mine and Project Development Area (PDA)

20.1.1 Geology and Geomorphology

Physiography

The Mine 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 due to the presence of bedrock outcrops.

Topography is relatively flat to gently rolling, with 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. The PDA is bounded to the south, east, and north by Kenogamisis Lake, which forms the main watershed to which the Mine is located. Local water features and topography were an important consideration in the siting and design of key Mine components, including the TMF and associated watercourse diversions and the waste rock storage areas (WRSA).

Surficial Soils and Geology

The surficial soils and geology are typical of the boreal forest region overlying the Precambrian shield in northern Ontario. Soils are relatively young, exhibiting less than 10,000 years of development, comprising organic muck (about 36% of the PDA) and well-drained brunisols over thin bedrock (comprising about 35% of the PDA); poorly drained gleysols account for 13% of the PDA. Developed land and water make up the remaining 16% of the PDA.

Surficial geology consists of large areas of glacial till, glacial outwash, and glaciolacustrine and glaciofluvial deposits. Unique to this area is the high percentage of calcareous (carbonate-rich) substrates. Carbonates are commonly found throughout all modes of soil deposition within the PDA and surrounding area. Till and other discontinuous drift (gravelly silty sand to sandy silt) is mapped in the PDA's northern and western portions, generally near the 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 1 and 3 m 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.

Bedrock Geology

A detailed description of the bedrock geology and controls on mineralization is presented in Section 7.

20.1.2 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 testing, was used for mine planning and development of the Waste Rock Management Plan. The following subsection presents a summary of the results of the 2013 to 2019 geochemical testing program (Stantec, 2017; Appendices E6.1, E6.2, and E6.3; Stantec, 2020).

Overall, the ore, waste rock, and tailings materials contain relatively low acid rock drainage (ARD) potential but will still require consideration of how best to 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 Project's Water Management and Monitoring Plan.

Overburden

Overburden is classified as non-PAG material and is unlikely to generate acidic leachate. Metal- and organic-impacted soils (i.e., mainly antimony, arsenic, boron, cobalt, copper, nickel, and molybdenum as well as petroleum hydrocarbons and polychlorinated biphenyls) are within the area of the historical MacLeod–Mosher and Hardrock plant sites. A Soil Management and Monitoring Plan has been implemented that defines soil types and management options for the defined soil types.

Waste Rock

Waste rock was characterized into seven rock types. No PAG samples were identified in two of the identified seven rock types evaluated—clastic sediments or intrusive rock (Stantec, 2017, Appendix E6.1). These two rock types account for approximately 88% of the predicted waste rock. PAG waste rock is mainly associated with the sulphide replacement zones. Overall, the geochemical testing results indicated that the majority of the predicted waste rock is non-PAG (98.9%) with high carbonate content, which will have the ability to neutralize acidic drainage created by the small quantity of PAG material present in the stockpiles. Stantec (2017, Appendix E6.1) estimates the minimum onset time for acidification of PAG rock to be 70 years after exposure to the atmosphere. Therefore, acid generation is not likely to occur during the life of mine, and additional mitigation measures can be implemented during mine rehabilitation so that the potential for acid generation is reduced. Co-deposition of PAG and non-PAG waste rock has been identified as the preferred management option, as outlined in the Mine Waste Rock Management Plan.

Geochemical characterization also included evaluating the potential for metal leaching from the waste rock. The results from the initial weeks of kinetic testing (2 to 5 weeks of testing) on individual rock types identified arsenic, antimony, aluminum, cobalt, and uranium to be the constituents of potential concern (COPC) based on comparison to the Ontario Provincial Water Quality Objectives (PWQO) and interim PWQOs (Stantec, 2017, Appendix E6.2). The COPC leaching rates were noted to be decreasing with time, with long-term results (i.e., 35 to 40 weeks of testing) suggesting that only arsenic, antimony, and aluminum exceeded the PWQO and interim PWQOs (Stantec 2017, Appendix E6.2). The geochemical test results indicates the majority of the rock in the waste rock storage areas have low leaching potential. The decline for arsenic leaching rates was 3.8 times lower in clastic sediments and 2.5 times lower in intrusive waste rock, which comprise 88% of the total predicted waste rock by volume.

During operation, waste rock runoff and seepage will be collected and treated prior to discharging to the environment. At closure, contact water from WRSAs will be diverted to the open pit to expedite filling of the open pit.

A portion of waste rock was used to construct the Mine, particularly the TMF. Waste rock will be required for future Mine and TMF construction activities for the remainder of the life of mine. Criteria were developed to define waste rock suitable for construction purposes, which considered the rock's metal leaching and ARD characteristics. A correlation between leachable and arsenic concentrations in solids was developed. Approximately 4,000 individual samples of waste rock and ore were sampled and analyzed with multi-elemental IC-MS. In general, the concentration of arsenic in leachate are below the PWQO of 0.1 mg/L for samples having a solids content of less than 40 ppm arsenic. Therefore, waste rock classified as non-PAG with an average arsenic grade below 40 ppm was considered suitable for construction purposes. Review of the data indicated that material with elevated concentrations of other COPCs such as antimony, aluminum, cobalt, and uranium was generally associated with the elevated arsenic, and therefore metal leaching potential for other COPCs were accounted for using the average arsenic grade control.

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 acidification 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 will be addressed through progressive rehabilitation and closure of the TMF.

In the TMF pond, concentrations of metals and total cyanide are predicted to meet Metal and Diamond Mining Effluent Regulation (MDMER) criteria based on results of ageing tests. Unionized ammonia, cobalt, copper, arsenic, antimony, silver, and free cyanide were identified as COPCs during operation based on comparison with the PWQO. TMF water will not discharge directly to the environment, and seepage through the TMF dams 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 its filling. Once the pit 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.

20.1.3 Atmospheric Environment

The Mine is in a rural location of northern Ontario where air quality is primarily influenced by the Town of Geraldton and traffic on Highway 11. Measured concentrations of nitrogen dioxide, sulphur dioxide, and inhalable particulate matter were below the applicable provincial criteria. The maximum measured concentrations of total suspended particles and metals with Ministry of the Environment, Conservation and Parks (MECP) air quality criteria were well below those criteria. The maximum measured concentrations of 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 aromatic hydrocarbons (PAH) 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 concentrations is conservative and overestimate actual background concentrations in the Geraldton area.

20.1.4 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 the natural environment at night. 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 record the characteristics of a Class 3 acoustical environment as defined in the MECP Environmental Noise Guideline—Stationary and Transportation Sources—Approval and Planning (NPC-300).

20.1.5 Groundwater

Field activities to confirm baseline hydrogeological conditions were completed from 2013 to 2021 and included borehole drilling and groundwater monitoring-well installations, 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 1 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 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; these are typical of groundwater in Ontario and are reflective of the natural mineralization and geochemical processes in the area. Overburden and bedrock water quality was generally of good quality away from historical mining areas, with parameters occasionally above the Ontario Regulation 169/03 Ontario Drinking Water Quality Standards, reflective of location conditions.

Several historical or existing land uses have contributed to the degradation of water quality in the Mine 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, CA$mium, lead, nickel, and silver. The concentration of cyanide, CA$mium, lead, nickel, and silver exceed the MECP Aquatic Protection Values in historical Hardrock tailings water quality but are less than the MECP Aquatic Protection Values in historical MacLeod water quality. In addition, a small area of the historical Hardrock tailings is considered acid generating (referred to as the reactive tailings). Seepage from both the historical tailings areas has been identified as affecting water quality within Barton Bay and the central basin of Kenogamisis Lake, particularly arsenic concentrations, which the concentration of arsenic in historical tailings groundwater are well above the PWQO.

The open pit water management strategy is to dewater the historical underground workings to allow precipitation and runoff that flow into the open pit to be stored in the underground workings, which will allow dewatering and water treatment to occur at a consistent rate. Baseline 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. Baseline water quality associated with the Mosher Shaft No. 1 had concentrations of iron and zinc that consistently exceeded the PWQO, and concentrations of arsenic that consistently exceeded the interim PWQO. Dewatering from the MacLeod Shaft No. 1 commenced in September 2021 and dewatering was switched from MacLeod Shaft No. 1 to Mosher Shaft No. 1 in October 2023.

20.1.6 Soil Quality

To support the approved EIS/EA, baseline soil sampling investigations were undertaken to characterize soil quality in the area of the former mining operations. Additional soil quality investigations were conducted in 2018 and in 2020 to support the development of the Soil Management Plan, which classifies soil into six types (Type A, A2, A3, B, C, and D) each with a defined reuse or disposal destination as summarized in Table 20-1.

Table 20-1: Soil Classification and Management Summary

Soil Classification	Classification Requirement	Reuse or Disposal Destination
Type A	Meets or is below Table 8.1 ESS and LSLs	Unrestricted use within the PDA. Use where high quality soil is required (e.g., surficial cover).
Type A2	Parameters exceed the Table 9.1 ESSLSLs but are below the Table 7.1 ESS/LSLs	Use for reclamation within PDA where >30 m from a water body and >250 m from PDA boundary. Where not possible, bury >1.5 m BGS in WRSA.
Type A3	Parameters exceed the Table 8.1 ESSLSLs but are below the Table 9.1 ESS/LSLs	Use for reclamation within PDA and >250 m from PDA boundary. Can be used within 30 m of a water body. Where not possible, bury >1.5 m BGS in WRSAs.
Type B	Parameters exceed the Table 7.1 ESS but are below the Table 71. LSLs and APVs	Bury > 1.5 m below ground surface in WRSAs.
Type C	Metals parameters exceed the applicable ESS and LSLs/APVs for the intended reuse (i.e., Tables 7.1, 8.1 or 9.1). No other parameters exceed the applicable ESS for the intended reuse.	Dispose within the TMF (metals impacted soil only). Type C soils may be temporarily stored in the starter pit prior to disposal in the TMF.
Type D	Exceeds the applicable ESS for non-metal parameters (and potentially metals parameters as well) for the intended reuse.	Treat on-site under an appropriate Environmental Compliance Approval (ECA) or dispose of off-site at a licensed waste management facility.

Notes: ESS = excess soil standard in Ontario Regulation 406/19; LSL = leachate screening level in Ontario Regulation 406/19.

Detailed soil characterization has been completed on 8.8 Mm^3 of the 13.1 Mm^3 of soil to be disturbed across the PDA as part of Mine development. Of the 8.8 Mm^3 of soil characterized across the PDA, 60% (5.2 Mm^3) was classified as Type A, 35% (3.1 Mm^3) as Type B, 4% (0.4 Mm^3) as Type C, and 1% (0.09 Mm^3) as Type D. Type B and C soil are typically associated with elevated concentrations of antimony, arsenic, boron, cobalt, copper, nickel, and molybdenum. Type D non-metal parameters are predominantly petroleum hydrocarbons, with a smaller amount of polychlorinated biphenyls. The majority of soil in the PDA's southern portion (i.e., south of the Southwest Arm Tributary) was classified as Type A, with a minor amount of Type B. Soil within the PDA's northern portion (i.e., north of the Southwest Arm Tributary) was classified as a mix of Type A, B, and to a minor extent Type C. Type D soil was associated with historical land uses such as the MTO Yard, Dan's General Store, and the historical MacLeod–Mosher and Hardrock plant sites. These areas with Type D soil are within the footprint of the open pit.

The majority of the 3.5 Mm^3 of soil remaining to be characterized is within the footprint of the open pit, where it is anticipated to be consistent with Type A and B soil, with a minor component of Type C.

20.1.7 Historical Tailings Quality

Mining activities associated with the historical Hardrock Mine, MacLeod–Mosher Mine, and Little Long Lac Mine resulted in tailings deposition on land and portions of Kenogamisis Lake, altering the natural shoreline. The historical tailings within the PDA are referred to as the MacLeod tailings and Hardrock tailings.

The land-based historical MacLeod and Hardrock tailings were delineated, and quality of seepage assessed, as part of the EIS/EA. The historical MacLeod land-based tailings cover an area of 120.78 ha, with a volume of 8,393,700 m^3. Groundwater from these historical MacLeod tailings is interpreted to flow primarily toward Barton Bay, with a portion of flow directed to the south and east toward Central Basin and Southwest Arm of Kenogamisis Lake. The mean concentration of arsenic, cobalt, iron, nickel, and silver in groundwater recharge from the MacLeod tailings exceeds the PWQOs. The mean concentration of arsenic in groundwater recharge from the historical MacLeod tailings is 10,400 µg/L, with generally higher concentrations observed with depth. Seepage from the historical MacLeod tailings is impacting the water quality of Barton Bay. As part of Mine development, 23% of these tailings will be relocated to the TMF, where seepage will be captured via a seepage collection system, thus mitigating effects of the land-deposited historical tailings on the receiving environment.

The Hardrock land-based historical tailings extend over an area of 26.98 ha, with a volume of 455,200 m^3. Groundwater from these historical Hardrock tailings is interpreted to flow toward Central Basin and Southwest Arm of Kenogamisis Lake. The mean concentration of cyanide (free), arsenic, CA$mium, cobalt, iron, lead, nickel, silver, and zinc in groundwater recharge from the Hardrock tailings exceeds the PWQOs. The mean concentration of arsenic in groundwater recharge from historical MacLeod tailings is 11,400 µg/L, with generally higher concentrations observed with depth in the tailings. Seepage from these historical tailings is impacting the water quality of the Central Basin Band Southwest Arm of Kenogamisis Lake. As part of Mine development, 70% of these tailings will be relocated to the TMF, where seepage will be captured via a seepage collection system, thus mitigating effects of the historical tailings on the receiving environment.

In 2022 and 2023, the lake-based historical tailings were delineated and characterized to confirm the overall effect of relocating a portion of land-based historical tailings on the water quality of Kenogamisis Lake (Stantec, 2023). The total volume of historical MacLeod tailings (on land and submerged) is estimated to be 417,000 m^3 over an area of 68.38 ha, with approximately 10% by volume (41,000 m^3) submerged within the Central Basin. The total volume of historical Hardrock tailings is estimated to be 8,255,300 m^3 over an area of 155.68 ha, with approximately 5% by volume (418,000 m^3) submerged within Barton Bay east. The submerged tailings represent 19% of Barton Bay east and approximately 0.5% of Central Basin by area. Comparing top and bottom water samples under winter ice found no evidence for increased concentrations of solutes that would be associated with a substantial loading of these elements from the submerged tailings to the overlying lake water either through advective groundwater seepage or diffusion. Given the downward vertical hydraulic gradient associated with the submerged tailings areas, the mass loading of arsenic and iron from advection dominated fluxes (i.e., seepage) from the submerged tailings is minimal.

20.1.8 Surface Water

Hydrology

The Mine is in the Kenogamisis River watershed, adjacent to Kenogamisis Lake. The lake is long, narrow, and shallow, consisting of four main basins referred to here as the Southwest Arm, Barton Bay basin, the Central Basin and Outlet Basin. Water levels within the 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 and 329.70 masl with two Cautionary Compliance Zones to provide flexibility during winter and spring freshet conditions.

The Kenogamisis River flows into the Southwest Arm of Kenogamisis Lake. Its watershed area upstream of Kenogamisis Lake is 760 km^2, which contributes approximately 92% of total inflow into the Southwest Arm of Kenogamisis Lake, and 65% of total inflow into the Outlet Basin. 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 in the PDA are the Southwest Arm tributary and Goldfield Creek. The Southwest Arm tributary is a second-order tributary, the main branch of which 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 eastward to Kenogamisis Lake. In November 2022, Goldfield Creek was diverted to the headwaters of the Southwest Arm tributary to allow construction of the TMF. Other areas of the PDA drain towards Mosher Lake and Barton Bay and the Central Basin of Kenogamisis Lake.

Surface Water Quality

Historical surface-water quality demonstrated that Kenogamisis Lake has been affected by human activities, primarily runoff and seepage from the historical mines and effluent discharge from the Geraldton municipal wastewater treatment plant (WTP). Temporal and spatial trends in water quality are apparent in the historical record. Surface-water quality has improved over time, as indicated by a decrease in the number of parameters that exceed guidelines for the protection of freshwater aquatic life (PWQO and Canadian Water Quality Guidelines for Freshwater Aquatic Life [CWQG-FAL]). Prior to 1990, provincial and federal guideline exceedances were common for CA$mium, cobalt, lead, copper, arsenic, iron, and phosphorus. Since 1990, only arsenic, iron, and phosphorus have routinely exceeded guidelines. Despite improvements in water quality for a number of parameters, no major change in arsenic, iron, and phosphorus concentrations has been observed over the 40-year record, with concentrations of arsenic, iron, and phosphorus remaining above the applicable guidelines.

Routine monthly or bimonthly baseline water quality monitoring commenced in 2013 and continued through to 2018, after which quarterly monitoring continued to 2021; these compared with historical results spanning almost 40 years. Surface-water quality was generally moderately hard (i.e., with a moderately high mineral content), with mean pH of 6.1 to 8.4, and mean total dissolved solids concentrations between 88 and 1,117 mg/L, typical of northern Ontario lakes. The main COPCs in Kenogamisis Lake include arsenic and iron, largely related to historical mining activities. 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.

Among the four basins of Kenogamisis Lake, the Southwest Arm had the lowest detected mean concentrations of arsenic, while Barton Bay had the highest mean concentrations of arsenic, iron, copper, and total phosphorous. Barton Bay is affected by discharge from the municipal WTP and receives direct inputs from historical MacLeod and Little Long Lac tailings areas through groundwater discharge and surface-water drainage. The Central Basin of Kenogamisis Lake receives direct inputs from groundwater discharge and surface-water drainage associated with the historical Hardrock tailings, overflow from Hardrock No. 1 Shaft, and upstream discharge from Barton Bay, although Central Basin did have lower concentrations of metals than Barton Bay. With the commencement of dewatering the historical underground workings in September 2021, overflow from Hardrock No. 1 shaft to Central Basin ceased in late 2021.

In lakes and creeks, sampled as unaffected background or reference lakes, most metal concentrations were below CWQGs and PWQOs, except for arsenic and iron.

Historical mining activities have contributed to the degradation of groundwater and surface water quality within and around 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 in individual basins and the lake overall. The mass-balance calculations indicate that, while it is a small component of flow into the lake, 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 Kenogamisis Lake Dam. By the time water from Barton Bay 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.

20.1.9 Fish and Fish Habitat

Characterizing fish and fish habitat in the PDA and surrounding area included reviewing pre-existing background information and field studies completed between 2010 and 2021. Fieldwork included collecting fish habitat, fish community, fish tissue, sediment quality, and benthic community data. Lakes within the region of the Mine provide cool-water habitat and maintain a diversity of aquatic vegetation, cover, and substrate. Larger lakes, including Kenogamisis Lake, also provided greater bathymetric structure (e.g., humps, shoals, flats.

There was an abundance of potential spawning habitat for northern pike and yellow perch throughout most lakes. 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 in the PDA and surrounding area. These streams provide 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 streams studied. Fish species found in Kenogamisis and Goldfield Lakes include walleye, lake whitefish, northern pike, yellow perch, and burbot, spottail shriner, trout, perch, and blacknose shiner are some of the more abundant forage species observed.

Shallow, isolated ponds, and first-order watercourses in the PDA and surrounding area are likely to 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 area assessed as part of the aquatic assessment for the EIS/EA. 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.

Sediment Quality

Sediment samples were collected throughout the PDA and surrounding area in 2013 and 2015 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) in all sample replicates across the studied area in the EIS/EA, except for Wildgoose Lake. The MECP Severe Effect Level (SEL) was exceeded in replicates at several sampling stations, consisting of Lake A-322, Goldfield Lake, Mosher Lake and Barton Bay and Central Basin of Kenogamisis Lake. Common parameters that exceeded the LEL were CA$mium, chromium, copper, lead, and nickel. Exceedances of the LEL for zinc occurred in individual replicates from the Central Basin and Barton Bay of Kenogamisis Lake. Metal concentrations in Wildgoose Lake sediment were below the LELs, except for chromium in one replicate. Metals concentrations in Wildgoose Lake were the lowest among the stations sampled in baseline studies. Differences in sediment particle-size distribution in Wildgoose Lake most likely account for some of the observed differences in metals concentrations when compared to other lakes.

20.1.10 Vegetation Communities

The Mine lies on the southern boundary of the boreal forest. The boreal forest is a mix of deciduous and upland coniferous forest cover as well as wetland 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 of the Mine, with frequent occurrences of deciduous vegetation communities and species, including white birch, trembling aspen, and balsam poplar. Wetland vegetation community types occur throughout the PDA and surrounding area and are common to the boreal forest region. Anthropogenic disturbances in the Mine 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.

Two hundred and fifty three (253)species of vascular plants were recorded in the study area, of which 91% (230 species) were native and 9% (23 species) were non-native species. No plant SAR or species of conservation concern (SOCC) were recorded in the PDA and surrounding area during botanical inventories, and are assumed not to be present in the PDA and surrounding area. No known provincially significant wetlands or provincially rare communities were identified in the PDA or surrounding area.

20.1.11 Wildlife and Wildlife Habitat

Wildlife observed in the area includes various mammals, birds, reptiles, and amphibians, which are generally common and abundant to the boreal region. Mammal species observed during baseline surveys include: moose, black bear, grey wolf, red fox, lynx, pine marten, ermine, little brown myotis, northern myotis, silver-haired bat, hoary bat, striped skunk, snowshoe hare, woodchuck, red squirrel, least chipmunk, and beaver. Confirmed resident or breeding SAR and SOCC in the studied area surrounding the Mine include Canada warbler, bald eagle, eastern wood-pewee, common nighthawk, barn swallow, northern myotis, little brown myotis, and taiga alpine butterfly. Since the completion of baseline studies, the taiga alpine butterfly was downlisted from conservation rank S3 (vulnerable) to S4 (apparently secure). Although not breeding locally, American white pelican staging and foraging habitat occurred (associated with Kenogamisis Lake). No amphibian or reptile SAR or SOCC were recorded. Bank swallow nesting occurred outside of the PDA and immediate surrounding area.

Woodland caribou were not recorded, and their presence is unlikely; however, the Mine is within their historical range. Nevertheless, the Mine is not in an area identified as critical habitat for woodland caribou (Environment Canada, 2012) and is in the Caribou discontinuous distribution habitat area. Provincial significant wildlife habitat (SWH) identified within the Mine area included: moose late-winter cover; waterfowl stopover and staging habitat (aquatic); amphibian breeding habitat, turtle wintering area; taiga alpine butterfly habitat; and waterfowl nesting habitat. Other important wildlife habitats identified in the EIS/EA as occurring within the Mine area included moose foraging habitat and non-treed wetland bird breeding habitat.

20.1.12 Labour and Economy

Between 2006 and 2021 the population of Ontario and the District of Thunder Bay was increasing, the Municipality of Greenstone was decreasing by 7.1%. Available population projections indicate that the municipality will continue to see population decline without development of the Mine, 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, with the smallest population. Mining is a key component of the economy in Northwestern Ontario, with at least 18 active advanced exploration projects during 2023, as well as nine operational mines (CEDC, 2024). One challenge with the growth of the mining sector is recruiting experienced and skilled workers.

Key industries providing employment locally in the Municipality of Greenstone include trades; transport and equipment operations; processing, manufacturing, and utilities; and 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 decreasing by nearly two percentage points. In comparison, the size of the labour force in the District of Thunder Bay decreased by 3% over the same period, while the Ontario labour force increased by 4%.

GGM has implemented a hiring strategy that focuses on trying to hire at the local and regional levels. The Mine will increase employment within the region, supporting population growth and directly benefitting the economy.

20.1.13 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 Mine is 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 surrounding area include:

- Municipal features, including a park, public boat launches, and public beaches
- Kenogamisis Golf Club
- Hydro One infrastructure, including a substation and power lines
- Highway 11 and Michael Power Boulevard
- Gas station
- Ontario Provincial Police station
- MTO patrol yard.

The EIS/EA reports that some municipal services and infrastructure are at or near capacity, including wastewater systems and solid waste facilities. GGM has been assisting the Municipality of Greenstone in upgrades to the existing Geraldton WTP to allow for the connection of the Mine camp to the wastewater systems. Upgrades to the Municipality of Greenstone Geraldton WTP were implemented in 2022 and 2023.

Mine design required that a portion of Highway 11 be realigned, which included the Ministry of Transportation (MTO) patrol yard and the Ontario Provincial Police (OPP) station. GGM constructed a new segment of Highway 11 before signing ownership over to MTO. The new segment of Highway 11 was commissioned in 2023. Along with the Highway 11 realignment, GGM constructed a new MTO garage and office building in 2023; in addition, GGM completed remediation at the former MTO patrol yard in 2023, which included managing contaminated soils. GGM is planning to build a new OPP station to replace the former station, which is within the PDA.

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 underserved 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.

During the Mine construction phase, GGM constructed a temporary camp to house approximately 400 direct and indirect employees, therefore allowing for skilled workers to be flown in and out. GGM is in the process of obtaining permits to allow operations to continue providing worker accommodation. The addition of a long-term camp will also create employment opportunities for local residents.

20.1.14 Land and Resource Use

Existing land use in the area includes urban and rural uses, transportation, mineral exploration, and historical activities like logging and mining, with abundant natural landscape features. These land uses occur on a mix of patent and Crown lands. Urban uses in and adjacent to the Mine area are concentrated in Geraldton and local townsites established during previous mining activities.

The local landscape includes a brownfield area with historical tailings and mine shafts from historical mining activity, two sawdust piles from historical logging activity at the end of Lahti's Road, and various trails that may be decommissioned mining or forestry roads.

Today, the most extensive land uses are forestry and consumptive recreation such as hunting (including black bear, game birds, moose, waterfowl, and hares), trapping, fishing (walleye and northern pike), golf, and tourism (Stantec, 2017, Appendix E10). The region has numerous lakes supporting species popular with anglers. Kenogamisis Lake itself is a popular destination for sport fishing and is the location for the annual Geraldton Walleye Classic.

MacLeod Provincial Park is east of the Mine, across the central basin of Kenogamisis Lake, which offers opportunities for camping, fishing, swimming, boating, canoeing, biking, picnicking, and bird-watching.

Other outdoor recreation infrastructure in the Mine area includes hiking, cross-country skiing, and snowmobile trails. Lahti's Road provides access to the western shore of the Southwest Arm of Kenogamisis Lake. GGM anticipates that, following closure, Lahti's Road will provide public access up to the Goldfield Creek diversion. The road was previously the responsibility of the MNRF, but is now used by GGM for monitoring and maintenance. GGM will engage in discussions with MNRF on the details following closure, when this road is no longer required for maintenance or monitoring. GGM also anticipates that the East Service Road will remain following closure, providing access from the highway to the lake public access point on the Southwest Arm of Kenogamisis Lake. Post-closure ownership of the road will be discussed during the mine life.

The Kenogamisis Golf Club is on Michael Power Boulevard, just north of Highway 11. It previously had 18 holes, but has been reduced to the front nine, which Stanley Thompson designed, and which was built in 1938. The back nine was constructed in 2000 on historical MacLeod tailings following rehabilitation of the tailings in the late 1990s. GGM owns the golf course property and leases it to the Municipality of Greenstone. It is GGM's intent to continue to preserve the front nine and clubhouse, although some temporary suspensions may be needed to ensure golfer safety during blasting.

Commercial resource activities include trapping, baitfish harvesting, guide outfitting, forestry, and mineral exploration. There are also a number of active and inactive aggregate mining areas in the region.

Indigenous communities fish in the lakes and rivers throughout the area. Mammals (e.g., moose, rabbit, marten) and birds (e.g., geese, grouse) are hunted and trapped by Indigenous community members, which continue to be important traditional activities practiced for both economic and cultural reasons. Trapping areas identified during baseline studies by Indigenous community members occur throughout the Mine and surrounding area.

Cultural sites (including trails and travel ways), sacred areas, communal gathering areas, and habitation sites are used by Indigenous communities throughout the area, who continue to use traditional gathering places for socializing, harvesting, or ceremonies.

20.1.15 Heritage Resources

Archaeological 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 PDA and surrounding 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.

Architectural and Historical Resources

A Cultural Heritage Evaluation Report was 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 that the Mine may affect; the vast majority are in residential developments constructed by mining companies. Of these, 18 were determined to be situated within the PDA.

20.2 Environmental Constraints

The Mine lies in 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 Mine design, which has focused on minimizing the Mine's environmental footprint while respecting environmental features and required setbacks.

GGM now owns all property, land, and infrastructure in the PDA including provincial infrastructure related to the MTO patrol yard; Hydro One transmission and distribution power lines, and associated substation; the Discover Geraldton Interpretive Centre; and properties within the MacLeod and Hardrock townsites and Dan's General Store (Husky Gas Station). GGM is also working to relocate the OPP station, and currently has a purchase arrangement in principal with the property owner.

Through the EIS/EA and implementation of the Soil Management Plan, extensive soil characterization and delineation has occurred across the PDA. Impacted soil was identified within the northern portion of the PDA associated with historical land uses such as the MTO patrol yard, Dan's General Store, Larry's Esso, and the historical MacLeod–Mosher and Hardrock plant sites. Soil impacts are typically associated with metals (antimony, arsenic, boron, cobalt, copper, nickel, and molybdenum) with a smaller amount of soil impacted by petroleum hydrocarbons and polychlorinated biphenyls. GGM has implemented extensive regulatory-agency-approved design and rehabilitation programs to manage the legacy environmental issues with these locations.

A Soil Management Plan has been developed and implemented to provide guidance on the managing excess soil generated during Mine development and operation.

Historical mining activities have contributed to the degradation of groundwater and surface-water quality within the area of the PDA, particularly caused by seepage from historical tailings. As discussed in Section 20.1.8, it is anticipated that the Mine's activities will result in an improvement in water quality within Kenogamisis Lake, and will contribute to reduced arsenic and iron concentrations by reducing groundwater seepage and discharge from the historical MacLeod and Hardrock tailings to Kenogamisis Lake. This will be achieved by removing a portion of the historical tailings and placing them in the newly constructed TMF, installing seepage collection around the in situ historical tailings as part of the berm-and-buttress construction to address long-term physical stability; improving the 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 Mine Rehabilitation Code during preparation, and upgrades will be completed as required during closure. The majority of the mine openings will be removed during Mine development, 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, bald eagle, bank swallow, barn swallow, common nighthawk, eastern whip-poor-will, little brown myotis, northern myotis, and woodland 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 obtained appropriate permitting prior to Mine development (refer to Section 20.3.2). Where possible, considerations were also given to Mine design, construction timing, and field monitoring to reduce impacts to SAR.

Development of the Mine resulted in alteration to 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 was signed between the Municipality and GGM to support the Municipality's future plans with respect to these facilities.

MacLeod Provincial Park is 350 m east of the PDA. There are no other provincially or federally protected areas such as national parks, ecological reserves, or conservation reserves near the Mine. There are no Areas of Natural and Scientific Interest or evaluated Provincially Significant Wetlands within or near to the PDA. One sensitive and rare fen community was identified immediately adjacent to the PDA, but these have not been provincially designated. There are no areas of archaeological resources identified through baseline studies at the Mine site.

Goldfield Creek was diverted around the TMF in November 2022 to allow the final stages of construction and subsequent commissioning of the TMF. The GFC diversion was constructed to divert GFC and to offset aquatic habitat that the TMF overprinted. In May 2023, significant erosion of the lower reaches of the GFC diversion resulted in the release of sediment into the creek with subsequent deposition in Southwest ponds (SWP) 1 and 2, and a minor amount of deposition in SWP 3. The sediment release resulted in orders from the Department of Fisheries and Oceans Canada (DFO) and the MECP. GGM implemented immediate mitigation measures that included silt fencing, armouring some of the meanders with rock, and installing ESC berms within the GFC diversion channel. A temporary bypass channel was designed and constructed to divert flow around the unstable portion of the GFC diversion. The temporary bypass channel was constructed as a hydraulic function and not intended for aquatic life. GGM is working in consultation with MECP, Ministry of Natural Resources and Forestry (MNRF), Impact Assessment Agency of Canada (IAAC), DFO, and Indigenous groups, for the GFC Diversion redesign and mitigation, as well as to implement measures directed in regulatory orders.

20.3 Environmental Approval Requirements

20.3.1 Environmental Assessment

Federal EA was regulated under the *Canadian Environmental Assessment Act 2012* (*CEAA 2012*)*,* and was administered by the Canadian Environmental Assessment Agency (CEA Agency). Under *CEAA 2012*, “designated” projects included in the regulations designating physical activities require a federal EA. The Greenstone Mine (formerly Hardrock Project) has been confirmed as a designated project and a federal EA was implemented in accordance with the approved EIS guidelines the CEA Agency issued to GGM on August 5, 2014, with subsequent amendments on February 11, 2016, to include consideration of greenhouse gas (GHG) emissions, and February 12, 2016, related to changes in the list of Indigenous communities with which GGM was 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 Mine 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 was 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 filing of a single body of information (i.e., the 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. The EIS/EA was amended December 2019 to incorporate additional information, commitments, studies, reports, and modelling that were submitted after the formal submission of the EA in accordance with Condition 27 of the provincial Notice of Approval. The federal Decision Statement was amended on February 10, 2021, to accommodate minor design changes during detailed Mine design.

The federal Decision Statement was amended on February 10, 2021, to accommodate minor design changes during detailed design of the Mine. On July 18, 2024, the federal Minister of Environment determined that the Decision Statement issued for the Greenstone Mine contained conditions that could have been included in a Decision Statement issued under subsection 65(1) of the amended *Impact Assessment Act*, and that therefore the Decision Statement for the Greenstone Mine was deemed to be a Decision Statement under section 65(1) of the amended *Impact Assessment Act*.

Consultation

Consultation is a key component of both federal and provincial EA processes to engage interested parties to identify and address concerns with Mine 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 Mine 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. Section 20.4 contains further details regarding consultation and engagement activities undertaken in support of the Mine.

Preliminary Effects Assessment

The methods used to conduct the environmental effects assessment were designed to meet the combined requirements of *CEAA 2012* and the *EAA*. These methods were based on a structured approach that, particularly:

- Considered 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
- Considered the issues raised by the public, Indigenous communities, and other stakeholders during consultation and engagement activities conducted to date
- Focused on issues of greatest concern that arose from the above considerations
- Considered existing environmental conditions of the area, particularly historical activities and resulting environmental effects that might have affected baseline conditions
- Integrated engineering design and programs for mitigation and monitoring into a comprehensive environmental planning and management process that was applied during the design and implementation of the Mine
- Considered the Mine in a careful and precautionary manner, to avoid significant adverse environmental effects.

The environmental effects assessment methods addressed both Mine-related and cumulative environmental effects based on the Mine description at the completion of the final EIS/EA. Mine-related environmental effects and cumulative environmental effects were assessed using a standardized methodological framework for each VC (valued component).

The following subsections summarize the environmental effects assessment, proposed mitigations, and determination of significance for each VC from the final EIS/EA.

Atmospheric Environment

The potential environmental effects of the Mine on the atmospheric environment include changes in ambient air quality, climate change, and change in lighting. As part of the assessment, mitigation measures were identified that have been or will be applied to the Mine 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 and implementing a no idling policy to reduce emissions; applying speed limits to reduce dust from vehicles travelling on gravel roads; minimizing haul routes to reduce vehicle use; locating portable lighting equipment where, to the extent feasible, it is not visible at nearby receptors; and using directional light fixtures to avoid transmitting light outside of the PDA.

During operation, additional mitigation measures 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 by using downlighting; and implementing a GHG management plan to minimize and track GHG emissions. In addition, new mobile equipment on site meet applicable Transport Canada off-road vehicle emission requirements.

With mitigation in place, air quality emissions resulting from construction were temporary and within applicable regulatory objectives, standards, and guidelines. It is anticipated that the same will apply at closure. Overall, the Mine'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 occurred during construction and are expected in closure. During operation, the Mine is expected to emit no more than 264 kt of carbon dioxide equivalent (CO_2e) per year.

The baseline lighting conditions were characterized as being rural (i.e., characterized by low district-brightness). The change in ambient lighting outside the PDA during operation is expected to be within guidelines for rural areas.

Residual adverse environmental effects on the atmospheric environment were determined not significant.

Acoustic Environment

The potential environmental effects of the Mine on the acoustic environment include change in noise and change in vibration levels. As part of the assessment, mitigation measures were identified that have been or will be applied to the Mine to mitigate effects.

During construction, major construction activities were scheduled during daytime where possible (i.e., 07:00 to 19:00), to avoid impact during nighttime. A similar approach will be taken during Mine closure. Other noise mitigation measures include installing muffler systems on combustion exhausts, properly maintaining equipment, and implementing a procedure to address noise complaints. Blast design met the MECP's criteria for noise and vibration, and blasting occurred during the daytime.

During operation, mitigation measures include selecting quieter equipment 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 Mine-related activities is predicted to be below applicable guideline criteria and thresholds.

Residual environmental effects on the acoustic environment were determined not significant.

Groundwater

The potential environmental effects of the Mine on groundwater include a change in groundwater levels or flow, and a change in groundwater quality. As part of the assessment, mitigation measures were identified that have been or will be applied to the Mine to avoid or reduce effects.

Mitigation measures for groundwater quantity and flow include: using standard management practices throughout the Mine, including drainage control and excavation and open pit dewatering; limiting the construction footprint (i.e., the PDA) to the extent possible to reduce potential reductions in groundwater recharge and limit the number of watersheds overprinted by the PDA; using standard construction methods, such as seepage cut-off collars, to mitigate preferential flow paths where trenches extend below the water table; during operation returning to Kenogamisis Lake water generated from historical underground dewatering—treated at the ETP as required—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 in a reduced period of time.

Mitigation measures for groundwater levels and flow include: implementing progressive rehabilitation by placing vegetated soil cover to reduce infiltration into the WRSAs and TMF; designing WRSAs to increase the amount of runoff and reduce the amount of infiltration through them; relocating to managed facilities (i.e., TMF, WRSAs, or licensed off-site disposal) approximately 23% of the historical MacLeod tailings and 70% of the historical Hardrock tailings, as well as contaminated soil from historical process-plant areas; installing a subsurface seepage-collection system around the northern portion of the base of the historical MacLeod high 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 TMF; and implementing cyanide detoxification technology to reduce cyanide concentrations and precipitate metals at the process plant, resulting in improved water quality within the TMF.

Regarding groundwater levels and flow, the water table will be lowered in the Mine area due to dewatering of the open pit; however, there are no groundwater users within the area affected, and GGM either owns or leases the lands. Groundwater quality is predicted to meet regulatory criteria at the point of discharge. In addition, removing a portion of the historical tailings will reduce loading to surface-water features, and result in a positive change in the receiving environment water quality. 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 not significant.

Surface Water

The potential environmental effects of the Mine on the surface water include changes in surface-water quantity and quality. As part of the assessment, mitigation measures were identified that have been or will be applied to the Mine to avoid or reduce effects.

Mitigation measures related to surface-water quantity and quality include: limiting the Mine footprint to the extent practicable to reduce contact-water volume and management requirements; maintaining existing drainage patterns using 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 peak flow; implementing progressive rehabilitation to reduce infiltration into the WRSAs and TMF; improving water quality in 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 mine, including developing drainage controls for areas prior to developing and expanding these features.

Regarding water quantity, changes in drainage patterns will be contained within the LAA, with flow continuing to the Southwest Arm—this includes those from the Goldfield Creek diversion; there will be limited changes to flows into Kenogamisis Lake, and the flow will be within the range of background variability. With the design and mitigation for the Goldfield Creek diversion itself, the Mine is not predicted to have a significant effect on water quantity. These predictions of environmental effects regarding water quantity remain valid with the operation of the temporary bypass channel substituting for the Goldfield Creek diversion as redesign of the Goldfield Creek diversion occurs.

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 Mine are predicted, during operation, to result in a decrease in arsenic concentrations in Barton Bay Central Basin, and Outlet Basin of Kenogamisis Lake. Overall, the Mine is anticipated to improve water quality in Kenogamisis Lake, have 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 a reduction in groundwater discharge from the historical MacLeod and Hardrock tailings.

Residual adverse environmental effects on surface water were determined not be significant.

Fish and Fish Habitat

The potential environmental effects of the Mine 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 environmental effects assessment, mitigation measures were identified that have been or will be applied to the Mine to avoid or reduce adverse effects.

Mitigation measures for fish and fish habitat include,: mine design, by avoiding sensitive fish habitats where feasible; managing construction effects by creating dry work areas; performing fish salvage, 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 limit potential for fish entrainment or impingement. GGM has implemented a Spill Prevention and Response Plan, Aquatics Effects Management and Monitoring Plan, and follows DFO guidelines for the use of explosives near water.

Fish mortality can be avoided during all Mine phases such that there is no substantive residual effect on fish mortality. The Mine has been designed to reduce the potential of fish mortality through avoidance and mitigation measures. Changes to flow and drainage will alter fish habitat, but Mine designs have reduced effects on local water bodies such that significant adverse 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 of fish habitat will be lost or permanently altered, much of which consists 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 not significant.

Vegetation Communities

The potential environmental effects of the Mine on vegetation communities include change in the abundance of vegetation communities; change in function; connectivity and quality of vegetation communities; and change in the abundance of plant species of interest. As part of the assessment, mitigation measures were identified that have been or will be applied to the Mine to avoid or reduce adverse effects.

The primary mitigation for vegetation communities is progressive rehabilitation of the PDA, which will commence throughout operations. 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.

With regard to change in abundance of vegetation communities, although it was estimated 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 to be removed are generally common and widespread in the RAA, the loss of community types in the PDA is not predicted to jeopardize the long-term viability of those community types.

With regard to change in function, connectivity, and quality of vegetation communities, changes in surface water flow or drainage, dust deposition, introduction of invasive species, or fragmentation and groundwater drawdown may affect vegetation and wetland communities; however, this is not expected to threaten the long-term viability of vegetation community types in the region.

With regard to change in the abundance of plant species of interest, vegetation clearing during construction resulted in removing plant species of interest to Indigenous communities; however, plant species in the PDA are common 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 not significant.

Wildlife and Wildlife Habitat

The potential environmental effects of the Mine on wildlife and wildlife habitat include change in wildlife habitat, change in mortality risk, and change in movement. As part of the environmental effects assessment, mitigation measures were identified that have been or will be applied to the Mine to avoid or reduce adverse effects.

Mitigation of potential Mine adverse effects on wildlife and wildlife habitat will be accomplished through implementation of a Biodiversity Management and Monitoring Plan and the progressive restoration of vegetation communities and wildlife habitat. GGM obtained required authorizations under the *Endangered Species Act*, and adheres to applicable timing windows. Additionally, mitigation measures proposed for other VCs, or as part of other environmental management and monitoring plans (e.g., noise), directly or indirectly reduce effects on wildlife.

Effects on SAR and significant wildlife habitat are not predicted to adversely 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. Mine 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. Mine 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 Mine 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 not significant.

Labour and Economy

Changes to both labour and economy are predicted with the development of the Mine. As part of the environmental effects assessment, mitigation measures were identified that have been or will be applied to the Mine to avoid or reduce adverse effects.

It is expected that the Mine will result in positive effects on labour by employing local workers, including Indigenous workers, thereby reducing the unemployment rate in the local area. These positive effects do not require mitigation, but GGM commits to implementing various mechanisms for enhancing Mine 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 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 implementing the Mine's labour and training framework, which includes partnerships with Indigenous communities and educational institutions, 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 saleable 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 removing existing structures, in particular the Kenogamisis Golf Club, MacLeod–Cockshutt Mining Headframe, and the Discover Geraldton Interpretive Centre.

The overall Mine effect on labour and economy is positive given the direct, indirect, and induced benefits of Mine expenditures. The Mine will result in an increase in the size of the labour force and reductions in the unemployment rate. The Mine is also anticipated to increase household incomes, opportunities for local and Indigenous businesses, and contributions to municipal taxes.

Residual adverse environmental effects on labour and economy were determined not significant.

Community Services and Infrastructure

The potential environmental effects of the Mine on community services and infrastructure include change in housing and accommodation capacity, change in municipal and provincial service infrastructure capacity, and change in transportation services and infrastructure capacity. As part of the environmental effects assessment, mitigation measures were identified that have been or will be applied to avoid or reduce adverse effects.

The mitigation measure for housing and accommodation is using a camp to accommodate the Mine personnel. Mitigation measures for municipal and provincial services and infrastructure include maintaining communication with relevant agencies and organizations to provide information; identify and address potential Mine-related implications for services and infrastructure; and to support responsible organizations in planning for, adapting to, or benefitting from Mine-related changes in demand. GGM will offer its employees an Employee Assistance Program and require pre-employment physicals. Mine 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 are available at the Mine site and new employees are required to take mandatory safety orientations. Employees are trained in fuel handling, equipment maintenance, and fire prevention and response measures. The Mine is controlled through security measures.

Mitigation for recreation and entertainment services and infrastructure includes providing the camp with dining services and a basic recreational area. GGM will be obtaining additional permitting and upgrading the camp to accommodate the operations workforce. GGM has maintained the Kenogamisis Golf Club clubhouse and the front nine holes and acted 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 busses operations workers to and from the camp to limit Mine-related traffic, use an on-site natural gas-fueled power plant and electrical recovered-heat distribution system to supply heat and power for the Mine operation, and have Mine-dedicated sewage treatment facilities. To limit Mine-related demands on both highway and air services and infrastructure throughout operations, GGM will schedule arrivals and departures of local employee traffic to occur earlier than the existing observed morning peak hour and later than the existing observed evening peak hour, as well as scheduling alternating work shifts so that all workers do not arrive and leave the area at the same time. A third-party sewage disposal contractor is providing portable washroom facilities during construction and early operation, until the Mine WTP and sewage discharge line is commissioned. A third party sewage disposal contractor will provide portable washroom facilities during active closure when facilities are decommissioned.

Residual adverse environmental effects on community services and infrastructure were determined not significant.

Land and Resource Use

The Mine's potential environmental effects 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 environmental effects assessment, mitigation measures were identified that have been or will be applied to the Mine to avoid or reduce adverse effects.

Mitigation measures to reduce adverse effects on recreational land and resource use include: initiating revegetation as soon as practical after Mine 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 Mine components, reducing the change in viewshed and muffling nuisance noise; siting the majority of Mine 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 stabilizing and rehabilitating aggregate source areas, the northeastern portion 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 Mine progresses. Mitigation measures related to the atmospheric environment, acoustic environment, fish and fish habitat, and wildlife and wildlife habitat are also considered related to land and resource use.

In addition to the mitigation measures listed above for recreational land and resource use, GGM will maintain access to mining claims on the peninsula east of the PDA to mitigate potential effects on commercially based land and resource use.

To mitigate potential adverse effects on navigation, GGM will use established watercourse crossings and avoid obstructions to navigation; undertake construction activities in a way that limits potential for debris from flowing into a navigable water body; and implement the mitigation measures related to surface-water quantity.

GGM will continue to meet with affected tenure holders on a regular, semi-annual, basis to discuss issues and concerns, and to provide Mine updates as well as continuing 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 was and will continue to be completed in accordance with the *Crown Forest Sustainability Act* and *Crown Timber Act*, and GGM did seek a Release of Tree Reservation under the Public Lands Act to remove trees on patent lands that have timber rights reserved to the Crown. The Mine was designed to use established watercourse crossings and avoid obstructions to navigation, and signs were posted at locations around the perimeter of the PDA to alert local land and resource users of the presence of the Mine 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 operation. In consultation with agencies, stakeholders, and Indigenous communities, rehabilitation will be designed to meet desired end land uses as the Mine progresses, and end land uses are identified in the Closure Plan.

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 not significant.

Heritage Resources

The potential environmental effects of the Mine on heritage resources include change in archaeological resources and change to architectural or historical resources. As part of the assessment, mitigation measures were identified that will be or have been applied to the Mine 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 implementing procedures and mitigation. In addition, GGM will retain a licensed archaeologist, and further archaeological assessment will be conducted with the involvement of Indigenous communities. Key construction and operation staff have been trained in recognizing basic archaeological artifacts in case a potential archaeological resource is found during Mine operation, such as Aboriginal material culture (e.g., clay ceramics, lithic artifacts, and faunal remains) or Euro-Canadian material culture (e.g., refined ceramics, glassware, construction debris, and personal effects); GGM worked 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, and the Archaeology Program Unit of the Ministry of Tourism, Culture and Sport.

The mitigation strategies to be used for architectural or historical resources include implementing a 60 m buffer zone to isolate Cultural Heritage Resource 1 (house within the PDA) from Mine activities; commemorating past occupation and past mining activity, involving the associated architectural or historical resources, by creating a record of activities and 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 or historical resources were identified during construction, nor are they anticipated for future Mine phases. Consequently, no residual adverse effects carried forward for the determination of significance.

Traditional Land and Resource Use

The potential environmental effects of the Mine on traditional land and resource use (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 or 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 have been or will be applied to the Mine to avoid or reduce effects.

Potential environmental effects on TLRU were determined based on the Mine-specific Traditional Knowledge studies, Mine 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 adversely 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 have been or will be applied to avoid or limit adverse effects on components of the environment related to TLRU. Where there was interest, GGM provided opportunities to local communities for harvesting plants for traditional purposes prior to construction. GGM worked with Indigenous community representatives in detailed recording and mapping of spiritual or cultural sites, a decision was then made about the relative importance of the site and, if warranted, how to maintain and control access.

Through Mine design, the length and location of roads has been considered to reduce potential access restrictions. A Pipe Ceremony was held prior to commencement of construction under the direction of local Indigenous communities.

It is predicted that residual adverse 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, reduced access is not anticipated to be an issue, and while access to the PDA will be limited for the lifetime of the Mine, 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 not significant.

Human and Ecological Health

The potential environmental effects of the Mine on human and ecological health include change in human health and ecological health. Mine emissions include releases into the terrestrial, aquatic, and atmospheric environment. As part of the human and ecological health risk assessment, mitigation measures were identified that have been or will be applied to the Mine to avoid or reduce effects.

Several mitigation measures incorporated for the Mine to eliminate or reduce its environmental effects will also serve to address human and ecological health effects. These mitigation measures include using dust suppressants, dust collectors, and protective covers; implementing a Water Management and Monitoring Plan; and 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 to Mine-related emissions (i.e., inhalation and ingestion). With the implementation of the planned mitigation measures for air and surface water, the potential increase in health risk as a result of the Mine is negligible. As such, adverse health effects are not expected and, correspondingly, a change to human or ecological health is not expected.

Cumulative Effects Assessment

Based on the characterization of the residual cumulative effects of the Mine after mitigation has been applied, 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 Mine.

20.3.2 Obtained Permits or Approvals

A range of other permits and approvals required for mine construction and operations were obtained from numerous federal, provincial, and municipal authorities. A comprehensive list of active permits and approvals for the current operation of the Mine are provided in Table 20-2.

Table 20-2: Obtained Permits and 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 and 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: Mines	Closure Plan for the Mine.
Permit to Take Water Legislation: *Ontario Water Resources Act*, Ontario Regulation 387/04 Responsible Agency: MECP	Surface water and groundwater taking and dewatering activities as follows: • Bypass pumping and groundwater- and surface-water taking during construction • Freshwater intakes from Kenogamisis Lake • Dewatering of the open pit and historical underground workings • Pumping of the MacLeod high tailings seepage collection system.
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 Mine components and activities.
Environmental Compliance Approval—Industrial Sewage Works Legislation: *Ontario Water Resources Act* Responsible Agency: MECP	Operation of a variety of sewage works including: • The mill process water circuit • Sewage treatment plant and discharge • Effluent treatment plant and discharge • TMF and associated seepage collection system • WRSAs, ore stockpile, and overburden stockpiles contact-water collection systems • MacLeod high tailings seepage collection system • Discharge of construction dewatering • Temporary sewage holding tanks.

Permits/Approvals	Associated Activities
Environmental Compliance Approval—Waste Disposal Site Legislation: *Environmental Protection Act*, Ontario Regulation 232/98 Responsible Agency: MECP	Disposal of construction or operation waste materials at an on-site construction and demolition landfill.
Ozone Depleting Substance Registration Legislation: *Environmental Protection Act*, Ontario Regulation 463/10 Responsible agency: MECP	Discharge of a Class 1 ozone-depleting substance or anything that contains a Class 1 ozone-depleting substance.
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 from three aggregate pit locations (S1, S4, and T2) and TMF Quarry. S4 was not developed.
Permits and Licences (various) Legislation: *Fish and Wildlife Conservation Act* Responsible Agency: MNRF	• Pre-development fish and wildlife studies • Initial fish and wildlife relocation • Destruction of beaver dams, furbearer or bear dens, and nests or 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 Goldfield Creek diversion dam, Goldfield Creek diversion, Southwest Arm Tributary grade-control structures, temporary bypass channel, and water crossings.
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 *Endangered Species Act* (ESA).
License(s) 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 Highway Improvement Act* Responsible Agency: MTO	Construction of buildings or facilities close to or adjacent to a provincial highway.
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.

Permits/Approvals	Associated Activities
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.

GGM has the permits required to begin operations. It is common for mining operations to require permit amendments throughout operations to account for changes as the Mine is developed (e.g., changes in water management, water takings), or modifications to mine closure.

20.4 Social and Community Considerations

Consultation with Indigenous groups and stakeholders (e.g., community members, agencies, interested parties) is key to the success of the Mine. GGM has undertaken active participation through meetings, public open houses, published newsletters, and other events. GGM's consultation program reflects the requirements of the consultation guidelines set out in the Code of Practice for Consultation in Ontario's Environmental Process (MOECC, 2014). In addition, the consultation program was designed to follow the federal EIS Guidelines and approved provincial Terms of Reference for the Mine.

20.4.1 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 First Nation (AZA) as part of the EA.

Provincially, the MECP identified that three communities—AFN, GFN, and LLFN—hold or claim Aboriginal or treaty rights that may be adversely impacted by the Mine, 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 Mine. These communities included:

- Animbiigoo Zaagi'igan Anishinaabek (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 and mitigation and were incorporated into the final EIS/EA.

In 2020, post EIS/EA, three long-term relationship agreements with Indigenous communities were implemented. The three long-term relationship agreements are with the three First Nations (AZA, AFN, GFN), LLFN, and MNO.

The GGM Indigenous Relations team meets regularly with local Indigenous communities discussing employment, training, and procurement opportunities through the Implementation Committee (IC). The IC comprises members of each of the partnering communities and provides an ongoing forum for communication and co-operative measures for supporting Indigenous participation levels in the Mine. This provides an avenue for community members to voice concerns or questions they may have and to receive feedback from GGM.

The Environmental Sub-Committee (EAS) reports to the IC and provides a forum for timely review and consultation and comment on Project Approvals and Environmental Mitigation & Monitoring Plans. The EAS considers and recommends appropriate testing, studies, or programs. Five Environmental Monitors from AFN, AZN, GFN, LLFN, and MNO actively participate in the daily operation of the GGM Environmental Department.

20.4.2 Summary of Influence of Consultation and Engagement on the Mine

Since the initiation of the EIS/EA process, consultation has been carried out related to baseline studies, the identification and evaluation of alternatives, assessment of environmental effects, mine design, long-term monitoring programs, permitting, compliance reporting, and mine closure. 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, selection of Project alternatives, assessment of environmental effects, identification of mitigation measures, and other components of the assessment.

20.5 Follow-up Environmental Monitoring and Management Plans

As part of the EA process, a monitoring framework was advanced for all subsequent phases of the Mine and EMMPs were developed. The framework includes both compliance monitoring and effects monitoring during all phases of mine life. The EMMPs outline the environmental protection measures and commitments to be carried out by GGM and its contractor and subcontractors during each stage of mine life to avoid or reduce potential adverse effects. These EMMPs outline adaptive management and contingency measures to respond to exceedances of regulatory standards related to environmental discharges or other adverse effects of the Mine. Contingency measures specific to each EMMP are 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.

The following EMMPs have been developed and implemented:

- Air Quality 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 and Resources for Traditional Purposes Follow-up Plan
- Emergency Response Plan
- Provincial Fish Sampling Program and Fish Population Monitoring Plan
- Fish and Fish Habitat Federal EIS Follow-up Monitoring Plan
- Spill Response and Contingency Plan
- Erosion and Sediment Control Plan
- Indigenous Peoples Health Risk Assessment Follow-up Plan
- Historical Tailings Management and Relocation Plan
- Mercury Monitoring Program and Management Plan
- Multi-Media Monitoring Plan
- Policy 2 Contaminant Monitoring Program and Reporting 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 Plan.

The EMMPs are living documents that require refinement following permit amendments, modifications to monitoring programs, changes in company policies and procedures, and the evolution of industry best-management practices. Program plans are iterative by nature, and the monitoring activities associated with the Mine will be used to inform adaptive management, which is a process for continuously improving environmental management practices.

20.6 Closure, Decommissioning, and Reclamation

Prior to April 1, 2024, mining operations in Ontario required 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. In 2020, GGM submitted a Closure Plan with Financial Assurance to Mines, which received approval on March 30, 2021. Since approval of the initial Closure Plan, GGM has filed two amendments:

- December 29, 2023, amendment to account for detailed design of the TMF and other Mine infrastructure for commencement of operations.
- August 8, 2024, amendment to include the Goldfield Creek temporary bypass channel to temporarily divert water from the Goldfield Creek diversion while repairs occurred to the Goldfield Creek diversion.

Effective April 1, 2024, the *Building More Mines Act* has made changes to the *Ontario Mining Act* and created Ontario Regulation 35/24 (Rehabilitation of Lands), which replaced Ontario Regulation 240/00 (Advanced Exploration, Mine Development and Closure under Part VII of the Act). Future Closure Plan amendments will be required to follow these regulatory requirements.

The Closure Plan includes details on closure, including progressive rehabilitation, rehabilitation measures, monitoring, and expected site conditions following closure. The following sections outline the phases of reclamation as they apply to the Mine, including long-term monitoring and financial assurance requirements.

20.6.1 Phases of Reclamation

There are three key stages of rehabilitation activities that occur over the lifespan of a mine, which are:

- Progressive rehabilitation
- Closure and active rehabilitation
- Post-closure monitoring and treatment or passive closure.

Progressive rehabilitation involves rehabilitation completed throughout the mine operation, prior to closure, where practical to do so. This includes activities that contribute to the overall rehabilitation effort and would otherwise be carried out as part of the closure rehabilitation at the end of mining life. Progressive rehabilitation opportunities include:

- Removing construction-related buildings and rehabilitating laydown areas and access roads used during construction (which are not required for mine operations)
- Stabilizing and revegetating aggregate source area S4
- Stabilization and revegetation of WRSAs after deposition is completed
- Rehabilitating the portion of the TMF upon completion of tailings deposition, consisting of a vegetated soil cover on the surface of the TMF
- Backfilling the eastern portion of the open pit once the area has been mined out
- Conducting pilot-scale constructed-wetland treatment studies for contact water
- Conducting a vegetation study to determine revegetation success
- Removing hazardous and non-hazardous waste materials from site where possible on a regular basis
- Reclaiming the Goldfield Creek temporary bypass channel, constructed to temporarily redirect flows from the Goldfield Creek diversion.

Closure rehabilitation involves activities that are completed after mining operation ceases, to restore or reclaim the Mine to as close to its pre-mining condition. Such activities include demolishing and removing site infrastructure, re-vegetating disturbed areas, and other activities to achieve the requirements of the Closure Plan. The following measures will be undertaken for Final Closure and closing out of the Mine:

- Backfilling portions of the open pit with waste rock, prior to allowing it to fill with water, creating a pit lake.
- The open pit will begin to fill with water from groundwater inflow, direct precipitation, surface water runoff, and water from collection pond M1 and the TMF pond. Once the pit lake water reaches an elevation of −140 m amsl in Mine Year 16, fresh water will be pumped from the Southwest Arm of Kenogamisis Lake to promote the formation of a stratified pit lake.
- Constructing an outlet channel or spillway connecting the southern portion of the pit lake to the Southwest Arm of Kenogamisis Lake (to be implemented when the pit lake is full and the water quality meets criteria for discharge.
- Buildings and infrastructure at the site will be inventoried, cleaned as necessary, and dismantled, with the exception of infrastructure required for long-term water management (if any) and facilities needed for site management and security.
- Power supply will be stepped from to 4160 V and 600 V as required. The transmission line and on-site power distribution lines will be left intact to provide site power during closure, and to operate any mine water management systems that might be required to manage water quality following closure—until it can be demonstrated that this power source is no longer required or can be reasonably replaced by another means of power, such as a connection to the provincial power grid.
- Dismantling on-site transmission lines and poles will be implemented once site power is no longer required.
- Demolition materials will be transported to a recycling facility if appropriate, or disposed of in an approved landfill.
- Natural gas power plant will be removed.
- At the end of operations, infrastructure, equipment, and mining materials will be removed from the site.
- Salvageable machinery, equipment, and other materials will be dismantled and taken off site for sale or reuse if economically feasible. Gearboxes or other equipment containing hydrocarbons that cannot be readily cleaned will be removed from equipment and machinery and trucked offsite for disposal at a licensed facility.
- The segment of the TMF haul road that crosses the Southwest Arm tributary will be removed when no longer required.
- Transportation corridors will be closed off and revegetated to an extent consistent with the specified future use of the land and which follows applicable legislation.
- The Highway 11 realignment has been transferred to the MTO and will remain in its realigned state, managed by the MTO.
- Concrete foundations of buildings and similar structures will be demolished to less than 0.5 m below grade, infilled with non-PAG waste rock or overburden as needed, and covered with overburden to support revegetation.

- Unnecessary petroleum products, chemicals, and associated wastes will be removed from site following operations. Where practical, unused products will either be returned to the suppliers, or made available to other possible users in the general area, if appropriate.
- There are no polychlorinated biphenyls (PCB) being used in operation. Existing PCB-contaminated soil associated with historical mine areas is being managed in accordance with the Soil Management Plan reviewed and approved by the MECP.
- A soil sampling campaign will commence at closure to test areas where hydrocarbons or other chemicals may have impacted the soil. Locations will be reclaimed as required.
- At the completion of mining, the TMF will be closed out in accordance with the Ontario Mine Rehabilitation Code. All tailings, rock piles, overburden piles, and stockpiles shall be rehabilitated or treated to ensure permanent physical stability and effluent quality.
- Until the TMF is allowed to discharge passively to the environment, the TMF will be operated as set out in the TMF's OMS manual.
- At closure, the water in the TMF seepage collection ponds and ditches will continue to be pumped back into the TMF until water meets acceptable discharge criteria. The TMF seepage collection system ponds may be retrofitted with constructed wetland system if passive treatment of water quality is required prior to discharge.
- Pumps and piping used for seepage collection and water management in TMF will be removed.
- The subsurface seepage-collection system installed at the historical MacLeod high tailings may be retrofitted with a constructed wetland to improve water quality. Pilot studies may be carried out during operation to evaluate the treatment efficiencies.
- WRSA D will need to be fully rehabilitated at closure. Other WRSAs will have been progressively rehabilitation during operations.
- Once the open pit is full and water within the collection ponds meets water quality objectives and can be discharged to the environment, the ponds will be dismantled or modified into wetlands. Collection ditches will be reclaimed either through grading to natural topography, or filled in.
- Remaining overburden that is not used in reclamation will be regraded and revegetated.
- At the end of operations, stockpiled ore will be run through the mill, and the rockfill pad will be excavated and deposited in the open pit or WRSA D. The pad will then be covered with overburden and vegetated.

Once closure rehabilitation activities have been completed, a period of post-closure monitoring is required to show that the rehabilitation has been successful.

20.6.2 Financial Assurance

As defined in the *Mining Act*, A lessee shall provide financial assurance as part of rehabilitation and Closure Plan prior to site development. The cost for implementing Mine closure is estimated to be C$111 million, and assumes third party costs, no resale of scrap values, and that all materials will be treated as waste. Certain items, such as mobile equipment, may in fact have residual resale value. GGM has proposed a phased approach for the provision of Financial Assurance in a manner commensurate with the development of the Mine.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimates

21.1.1 Non-Sustaining Capital

The non-sustaining capital cost is shown in Table 21-1, and is estimated to be $318.4 million for the LOM operating period.

Table 21-1: Non-Sustaining Capital Cost LOM Summary

Capital Cost—Non-Sustaining	Total Cost ($M)
Building and Infrastructure	3.5
Machinery and Equipment	5.8
Non-Sustaining—Leased Equipment	232.2
Project Carryover	51.0
Capitalized Development	25.9
Total Cost	**318.4**

Major items included in the non-sustaining capital include the relocation cost of the Ontario Provincial Police station, the payment for the off-site laboratory purchase (located in Geraldton), the purchase of the seventh gas-powered generator, all the lease payments for the mining fleet, the cost of the MacLeod Township demolition and the rehabilitation work for the Gold Field Creek diversion.

21.1.2 Sustaining Capital

Table 21-2 shows the sustaining capital cost estimated to be $608.8 million for the LOM operating period.

Table 21-2: Sustaining Capital Cost LOM Summary

Capital Cost—Sustaining	Total Cost ($M)
Buildings and Infrastructure	44.6
Hardware/Software	2.0
Perimeter Dam	1.0
Machinery and Equipment	45.1
Major Capital Repairs	313.0
Tailings Management Facility	138.2
Fleet Purchase	64.9
Total Cost	**608.8**

Major items included in the sustaining capital include major capital repairs for the mining fleet, TMF expansions, new mining fleet equipment purchases, a new camp accommodation area, and strategic spares for the processing plant.

In addition, a portion of the major waste-stripping costs for the open pit is capitalized. If the waste stripping volume in any quarterly period is greater than the waste-stripping level of the overall average LOM stripping ratio, then it is considered as capitalized stripping. The total LOM capital stripping is $338 million.

21.2 Operating Cost Estimates

21.2.1 Operating Costs Summary

Operating costs are summarized in Table 21-3. The operating costs include mining, processing and G&A. The average operating cost is $845/oz Au or $30.22/t milled over the LOM operating period.

Table 21-3: Operating Costs Summary

Category	Total Costs ($M)	Unit Cost ($/t milled)	Cost per oz ($/oz)
Mining	2,512	17.36	485
Processing	1,076	7.44	208
G&A	784	5.42	152
Total Operating Costs	**4,372**	**30.22**	**845**

A summary of the total operating costs, by year, is presented in Figure 21-1.

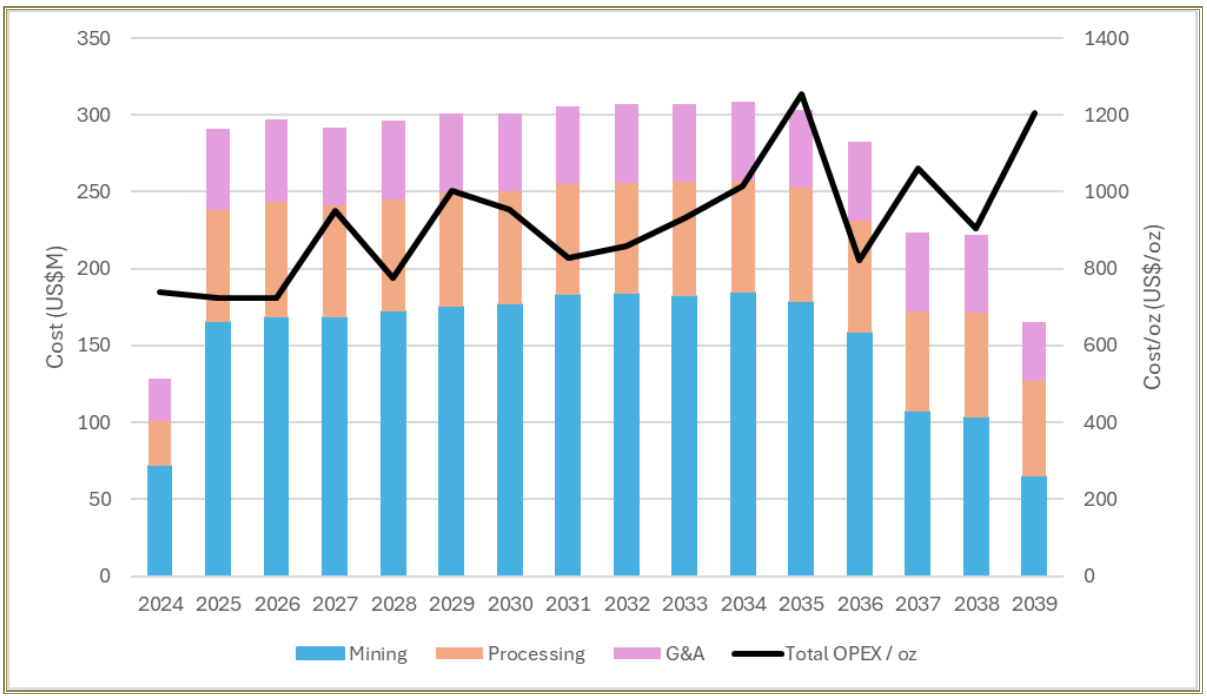


Figure 21-1: Operating Cost by Year

21.2.2 Mining Costs

The mine operating costs are estimated from first principles for all mine activities. Equipment hours required to meet production needs of the LOM plan were 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).

The average mining cost during operations is estimated at $2.70/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. This operating cost estimate excludes capital repairs, which are treated as sustaining capital.

Haulage is the major mining cost activity (38%) of total costs followed by blasting (11%), maintenance (11%) and drilling (7%). Some haulage costs have been reallocated to the TMF expansions as this represents an incremental haulage of waste rock that exceeds the haul distance to the closer waste dumps.

21.2.3 Processing Costs

The process plant operating costs were evaluated based on metallurgical testwork, recent supplier quotations, recent salary surveys and standard industry practice. The process costs are divided into several 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 cost is estimated at $7.44/t milled at a steady state plant throughput of 27,000 t/d.

The power cost of site-generated power was derived from three major components: forecasted energy price (natural gas), workforce required to operate and maintain the power plant and maintenance costs over the LOM. The processing plant electrical power requirements are based on the electrical demands specified in the equipment load lists, which considers the installed power, the utilization factor, the mechanical load factor and the process availability.

The wear parts cost category includes all the major equipment replacement parts (crusher liners, ball mill liners, HPGR tyres, etc.) and are generally based on equipment vendor recommendation and/or contractor 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.

The maintenance parts cost category includes all the minor normal operation replacement parts, such as pump casings, screen decks, chute liners, conveyor belts, etc.

Most reagents consumption data is derived from testwork. 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) the selected vendor pricing is included in the plant cost model. Grinding media consumption is based on the ore abrasion index. The ball mill power consumption and grinding media costs are used to evaluate an annual grinding media cost.

Oxygen is produced on site by a plant that is built, owned, and operated by a third party. A fixed monthly fee is associated with this service.

Sample preparation for mining and plant samples for gold assaying is preformed by an off-site lab. The geochemical lab performs pulp digestions and analysis for all other geochemical requirements and the metallurgical lab conducts bottle roll testwork 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.

21.2.4 General and Administration Costs

The annual G&A cost peaks at $54 million per year. The average G&A costs over the LOM is estimated at $5.42/t milled.

21.2.5 Operations Workforce

The operations workforce is made up 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 715 employees as shown in Table 21-4.

Table 21-4: Peak Operations Workforce

Operations Department	Peak Workforce
Mine	498
Process Plant	113
G&A	104
Total Number of Employees	**715**

22 ECONOMIC ANALYSIS

NI 43-101 regulations exempt producing issuers from the requirement to disclose economic analysis on properties currently in production, unless the technical report prepared by the issuer includes a material expansion of current production. Equinox Gold is a producing issuer, the Greenstone mine is currently being commissioned, and a material expansion is not included in the current Greenstone LOM Plan.

GMS has performed an economic analysis using the Mineral Reserves and LOM Plan presented in this Technical Report, and confirms the outcome is a positive cash flow that supports the statement of Mineral Reserves.

23 ADJACENT PROPERTIES

The QP 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 Greenstone Mine.

There are no adjacent properties that have any significant information relating to the Mine. 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 historical 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

Production Years	Bankfield Mine 1937–1942, 1944–1947	Little Long Lac Mine 1934–1954, 1956	Magnet Mine 1936–1943, 1946–1952	Talmora Long Lac Mine 1942, 1947-1948	Tombill Mine 1938–1942, 1955	Total
Ore Milled (short tons)	229,009	1,782,516	359,912	9,570	190,622	2,571,629
Ore Milled (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 (oz)	66,416	605,449	152,089	1,415	69,120	894,489
Silver (oz)	7,590	52,750	16,879	66	8,595	85,881

Source: Ferguson et al. (1971) and Mason & White (1986).

23.1 Talmora Long Lac (Past-Producer)

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, 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 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-tonne 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.

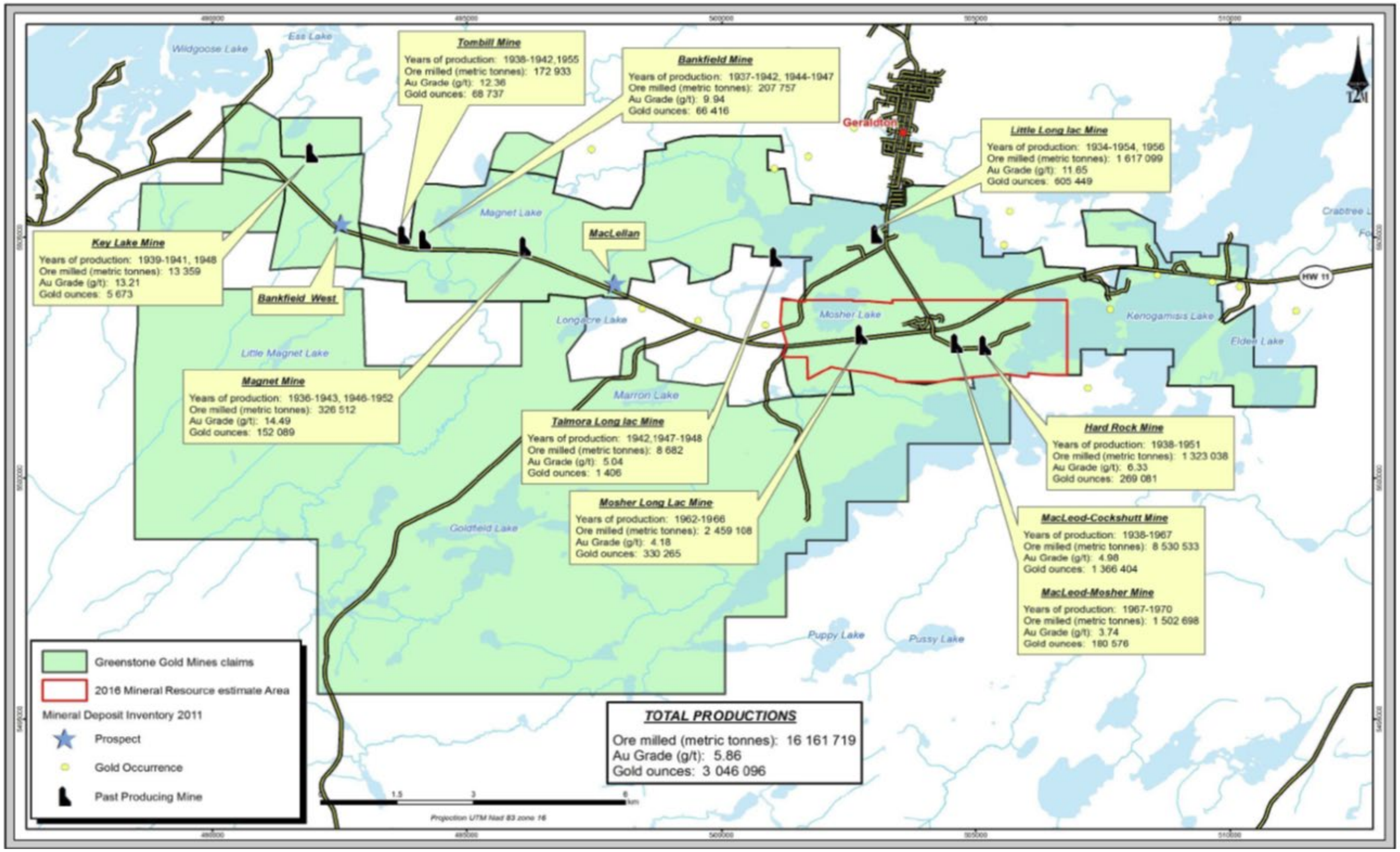
Between 1947 and 1948, Talmora Longlac Gold Mines Ltd. completed 1,663 ft (506.9 m) of drifting and 670 ft (204.2 m) of crosscutting. Diamond drilling comprised four surface holes totalling 139 ft (42.4 m) and

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, 398.5 oz of gold and 30 oz of silver were produced from 5,623 tonnes of hoisted material, for an average grade of 0.07 oz/t (2.2 g/t) Au.

At the time operations were suspended, it was estimated that about 12,000 tonnes with an average grade of 0.37 oz/t (11.5 g/t) Au 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 NI 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 sulphides are pyrite and arsenopyrite.



Source: Innovexplo (2015).

Figure 23-1: Past Gold Producers on Greenstone Gold Mine Claims

23.2 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 multiple levels from 200 ft (61.0 m) to 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 multiple levels from 2,405 ft (733.0 m)to 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-short 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 (10.6 g/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.

23.3 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 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 uncovered three mineralized zones, two of which—now known as the Magnet and Wells vein zones—showed considerable promise. 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 multiple levels from 203 ft (61.9 m) to 1,080 ft (329.2 m) from 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 DDHs totalling 1,665 ft (507.5 m) was completed. A 100-short ton amalgamation–flotation mill was built.

Between 1940 and 1952, the shaft was continued to a depth of 1,772 ft (540.1 m), with additional levels from 1,230 ft (374.9 m) to 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 from 1,884 ft (574.2 m) to 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 DDHs 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 tonnes of hoisted material. Average gold recovery was 0.42 oz/t (13.1 g/t).

The geology of the mine consists of metasediment—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 sulphides. 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).

23.4 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 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 multiple levels from 150 ft (45.7 m) to 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 multiple levels from 779 ft (237.4 m) to 1,275 ft (388.6 m). Sublevels 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 (709.6), and 10,145 ft (3,092.2 m) of unspecified drilling. A 100-short 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 (9.0 g/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 that 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°, with an average width of 7 ft (2.1 m), and is 2,000 ft long (609.6 m) including its extension into the adjacent Tombill Property. The deposits at the Bankfield Mine consist mainly of sheared and silicified greywacke and porphyry, mineralized with sulphides 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.

23.5 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 in the east-central part of Lindsley Township, about 10 km west–southwest 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-short ton capacity was erected and was later increased to 150 tonnes. 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 (11.2 g/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.

23.6 Gold Potential of the Other Historical Mines

The information presented on historical gold mines near the Greenstone Mine was obtained through the literature and is not verified by GGM. The presence of significant mineralization on these adjacent historical mines is not necessarily indicative of similar mineralization at the Greenstone Mine.

24 OTHER RELEVANT DATA AND INFORMATION

There is potential for an underground mine adjacent to the open pit that will be evaluated in the future.

25 INTERPRETATION AND CONCLUSIONS

25.1 Conclusions

The updated LOM for the Greenstone Mine is presented in Table 25-1.

Table 25-1: Greenstone Mine LOM Values

Description	Unit	Values
Gold Price—Base Case	$/oz	1,550
Exchange Rate	CAD/USD	1.28
Mine Life	operation years	15.0
Strip Ratio	W:O	5.50:1
Average Process Rate	kt/d	27.0
Average Grade	g/t Au	1.23
Average Gold Recovery	%	90.8
Average Annual Gold Production	koz	332
Total Recovered Gold	koz	5,176
Non-Sustaining Capital	$M[1]	318
Sustaining Capital	$M	609
Operating Cost	$/oz	845

Note: [1] Includes pre-production revenue credit and LTRA costs and excludes working capital.

The principal conclusions by area are detailed below.

25.1.1 Geology and Mineral Resources

Greenstone Mine

- Since the 2019 MRE, there has been significant RC and diamond drilling at the Greenstone Mine. Drilling focused on de-risking the early years of production RCGC targeting the first three benches of production), in-filling gaps in the drill pattern, and validating the new geological model. Since the start of mining, RCGC has been used to delineate the tonnage and gold grades to be mined. Depending on the various cut-off grades, the material is dispatched to various stockpiles or directly to the processing plant.
- At a cut-off grade of 0.30 g/t Au, the in-pit Measured and Indicated Mineral Resources are estimated to be 160.8 Mt at an average grade of 1.27 g/t Au for 6.6 Moz of contained gold, inclusive of Mineral Reserves. In-pit Inferred Mineral Resources are estimated to be 13.4 Mt at an average grade 1.51 g/t Au for 649 Koz of gold, inclusive of Mineral Reserves.
- At a cut-off grade of 2.00 g/t Au, the underground Indicated Mineral Resources are estimated to be 11.0 Mt at an average grade 4.20 g/t Au for 1.5 Moz of gold. Underground Inferred Mineral Resources are estimated to be 19.5 Mt at an average grade of 3.88 g/t Au for 2.4 Moz of gold.

Brookbank, Key Lake and Kailey

- The Brookbank deposit consists of in-pit Indicated Mineral Resources of 1.15 Mt at an average grade of 2.24 g/t Au, for 83 koz of gold, and Inferred Mineral Resources of 0.05 Mt at an average grade of 2.07 g/t Au, for 3 koz of gold. The underground Indicated Mineral Resource is stated at 2.28 Mt at an average grade of 7.06 g/t Au, for 517 koz of contained gold, and Inferred Mineral Resources of 0.71 Mt at an average grade of 3.38 g/t Au, for 77 koz of contained gold.
- A cut-off grade of 2.4 g/t Au was chosen for underground resources at Brookbank and is considered appropriate for reporting Mineral Resources.
- The Key Lake deposit consists of in-pit Indicated Mineral Resources of 3.76 Mt at an average grade of 1.16 g/t Au, for 141 koz of contained gold, and Inferred Mineral Resources of 1.84 Mt at an average grade of 1.39 g/t Au, for 82 koz of contained gold. No underground Mineral Resources are reported at Key Lake.
- The Kailey deposit consists of in-pit Indicated Mineral Resources of 11.28 Mt at an average grade of 0.96 g/t Au for 348 koz of contained gold and Inferred Mineral Resources of 4.86 Mt at an average grade of 0.87 g/t Au for 136 koz of contained gold. No underground Mineral Resources are reported at Kailey.

25.1.2 Mining and Mineral Reserves

- At a cut-off grade of 0.30 g/t Au, the Proven Mineral Reserves total 6.8 Mt at an average grade of 1.16 g/t Au, for 255 koz of contained gold. The Probable Mineral Reserves total 137.8 Mt at an average grade of 1.23 g/t Au, for 5,445 koz of contained gold. The total Proven and Probable Mineral Reserves is 144.7 Mt at an average grade of 1.23 g/t Au, for 5,700 koz of contained gold.
- The mining activities will occur over a period of 15-years (from start of commercial production to the end of in-pit mining) and excluding the pre-production period.
- The open pit will generate 788.6.7 Mt of overburden and waste rock (inclusive of historical tailings and underground backfill), for a strip ratio of 5.5:1.

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 Greenstone Mine were selected based on the results of this testwork and are well known technologies currently used in the mining industry.
- The gold recovery process consists of a crushing circuit; an HPGR and 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 and is operational since May 2024.
- The overall gold recovery is 90.8% and is based upon metallurgical testing completed comprising composite samples representing the full (global) deposit, early production years, lithological zones, and 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 have been relocated or purchased and dismantled. All private properties within the project area have been purchased.
- Power is provided to the site via a natural gas-fired power plant, 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) has been permanently diverted from the TMF to the northeast and ultimately to Kenogamisis Lake. The diversion dike required for the GFC diversion was designed in accordance with CDA and LRIA guidelines.
- During the 2023 spring freshet, significant erosion of the realigned GFC diversion channel occurred, resulting in the transport and deposition of sediment downstream of the constructed channel. Subsequently, a temporary bypass channel was constructed in 2024 to bypass flow around the eroded diversion channel while the realigned channel is redesigned and remediated. The temporary bypass channel is regarded as a temporary feature, and will remain in active service until the Goldfield Creek diversion has been restored, vegetated, and considered stable to receive flow.
- 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 to establish wide tailings beaches adjacent to the rockfill containment dams and to maintain the water pond against natural ground and away from the dams.
- Because of the presence of GL Silt in the foundation of most of the TMF dams, the dam raises in 2025 and subsequently will require design measures to maintain the target factors of safety. These measures are expected to include larger downstream stability berms, shear keys and in places ground improvement such as DSM.
- Should it become necessary to increase the tailings capacity beyond the current design capacity of the TMF, it would likely be necessary to construct a new TMF proximal to the existing facility. Preliminary consideration of possible expansion opportunities is underway.
- Any seepage and runoff water from the TMF is pumped back into the TMF reservoir.

25.1.5 Environmental Considerations

- The EIS/EA received federal Decision Statement on December 13, 2018 and provincial Notice of 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.
- EMMPs were developed and implemented and include measures related to both compliance and EIS/EA monitoring for all phases of the Mine. The collective monitoring activities associated with the

Mine are used to inform adaptive management for the Mine, as required. The management and monitoring requirements have been incorporated into Mine plans and budgets.

- Permits required for construction and early operation of the Mine have been obtained and conditions of the permits have been implemented, as appropriate.
- Mitigation measures for the Goldfield Creek diversion channel have been implemented to de-risk potential additional erosion of the channel as redesign and rehabilitation of the channel continues.
- Active consultation with stakeholders (community members, agencies and interested parties) and Indigenous communities has been undertaken throughout Mine planning and continued through construction and into operation of the Mine.
- Greenstone Gold Mines has established Long Term Relationship Agreements with the five local Indigenous communities. The agreements establish increased clarity regarding GGM's ability to develop the Mine and the Indigenous communities' opportunity to benefit from future mining opportunities in the region, including the potential to extend the life of the Mine.

25.1.6 Capital and Operating Costs

- The non-sustaining capital cost is estimated to be $318.4 million for the LOM operating period. The main cost items include the relocation cost of the Ontario Provincial Police station, the payment for the off-site laboratory purchase (located in Geraldton), the purchase of the seventh gas-powered generator for the power plant, all the lease payments for the mining fleet, the cost of the MacLeod Township demolition and the rehabilitation work for the Goldfield Creek diversion.
- The sustaining capital cost is estimated to be $608.8 million for the LOM operating period. The main cost items include major capital repairs for the mining fleet, TMF expansions, new mining fleet equipment purchases, a new camp accommodation area, and strategic spares for the processing plant.
- The average operating cost is $845/oz Au or $30.22/t milled over the LOM operating period. The operating costs include mining, processing and G&A.

25.2 Risks and Opportunities

25.2.1 Risks

25.2.2 Discussion of Risks

The following is a discussion of the key risks for the Mine with summaries of the related controls and risk mitigation strategies.

Gold Production

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. During operations, ongoing optimization of the metallurgical performance will be carried out via leach testwork, and throughput vs. grind-size trade-offs will be evaluated on a regular basis in conjunction with anticipated gains from the HPGR circuit due to microcracking.

Permitting

With ongoing constraints in the public sector, GGM is monitoring the risk of agencies not meeting a reasonable timeframe for any on-going or future permitting approvals. To facilitate the approval timeframes, consultation with Indigenous communities and agencies is undertaken on key permit applications prior to submission.

The permit that governs the annual mined quantities is in the process of being increased from 70 Mt/a to 72 Mt/a.

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 fieldwork. WSP and GGM carried out an operational risk assessment specific to dam safety aspects in June of 2024.

A detailed Tailings Facility Construction Management Plan, including a QA/QC program, has been implemented for construction for current and future expansions of the TMF. A tailings deposition plan and a dam-raising schedule have been developed to ensure capacity for the mill tailings during operations. An Operations Management and Surveillance (OMS) Manual following the guidelines of the Mining Association of Canada has been put in place for the TMF.

An Independent Tailings Review Board (ITRB) was established to provide oversight during the lifecycle of the TMF and is an on-going process. 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 design of the TMF.

Pit Wall Stability

A comprehensive pit slope management program is in place by the Mine's geotechnical engineering department to manage risks attributable to potential movement of the exposed rock faces. Rock mass failure is considered a low risk 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.

Stability of Historical Tailings

Attention to mining practices when mining proximal to the historical tailings have been implemented, especially focusing on controlling vibrations attributable to blasting activities. A monitoring program has been completed that indicated that controlled open pit blasting should not induce porewater pressures or cause slope instability in the adjacent historical tailings deposits. Emphasis is placed on minimizing exposure of excavated tailings slopes. Rockfill will continue to be installed on the slopes following the advancement of the excavation.

Water Management

The Mine is bordered on three sides by lakes and is cross-cut by small streams. There are several risks associated with the use, treatment and discharge of water during operations and closure. These risks and associated treatment plans are as follows:

- The risk of unacceptable contaminants such as arsenic seeping from the TMF, historical tailings, and waste rock storage areas have been mitigated by seepage collection ditches and collection ponds that allow for water to be collected and recycled to the process plant during operations to ensure the required water quality objectives are met.
- The risk of water ingress into the open pit are manageable based on historically low dewatering rates from the low permeability of the host rock. Pit dewatering will be maintained a minimum 20 m below the lowest mining bench elevation.
- The process plant relies on water collected from the historical underground workings, the open pit, and surface drainage as its main source of fresh water, which, given the overall requirement to discharge water from the permanent effluent treatment plant, should not pose a risk of a water deficit.
- The treated water from the effluent treatment plant is required to meet certain water-discharge criteria established for the Mine, which includes ammonia. An ammonia treatment option has been designed and permitted for the water treatment plant and can be implemented if ammonia concentrations within the pit (from explosives use) reach the given trigger threshold as defined in the permit.

25.2.3 Opportunities

Several potential opportunities exist to further improve the overall economics and sustainability of the Mine.

Revenue-Related Potential Opportunities

- Use of the Mine's process plant and TMF for ore processing from other GGM properties including the Greenstone underground resource and the regional exploration projects.
- The Mine is permitted for 30,000 t/d, providing an opportunity to increase throughput.
- Connecting the natural gas power plant to the provincial electrical grid to either sell spare energy during shutdowns when excess generating capacity is available or to provide electrical stability to the grid.
- Study the potential to economically remove magnetite from the tailings and produce a concentrate product for sale.
- Study the potential to economically reprocess historical tailings from the Mine's property and other nearby GGM properties.
- There is significant resource growth and discovery, such as open-pit expansion to the west, incorporating the underground deposit, and the regional potential.

OPEX-Related Potential Opportunities

- The potential to blend liquid natural gas and diesel as a fuel source for the mine haul trucks. Currently, the mine fleet only uses diesel.
- The use of new, commercially available technologies (i.e., autonomous haulage) to increase operational effectiveness and reduce costs.
- Optimizing the existing remote-assisted drilling to achieve additional labour productivity improvements.

CAPEX-Related Potential Opportunities

- Evaluation of used equipment with low usage to reduce CAPEX.

26 RECOMMENDATIONS

26.1 Mine Geology Recommendations

- Update the current block models using the drilling information RCGC performed since March 23, 2022, the cut-out date for the database used for the September 2022 Mineral Resource estimates.
- A detailed study should be undertaken to fully understand the impact of including blast hole sampling assays on the "ore control" polygons. Blasthole sampling assays do not have the same quality as DDH and RCGC and their usefulness for grade control within the main wireframes should be fully evaluated. Following this, the Mine's technical services team may consider using DDH and RCGC assay results for the design of the ore-control polygons or only using blast hole assays to find additional mineralization outside the main wireframes within the external grade shells.
- Detailed lithology and structural mapping of the pit walls should be completed regularly, and this information used to update and further refine the litho-structural model.
- A desktop study for the potential underground extension below the 2024 pit design should be conducted and should include sensitivity analysis to cut-off grades and mining methods.
- The current (and more conservative) Mineral Resource classification criteria should be reassessed with knowledge gained from further drilling and reconciliation. A 15,000-m diamond drilling program using oriented core is proposed for an all-inclusive cost of $3.0 million to reduce risk associated with the mineralization contained within the external grade-shells.
- Additional drilling should be carried out to convert Inferred Mineral Resources (exclusive of Mineral Reserves) to a higher confidence category. A program of 8,000 m of diamond drilling is proposed for an all-inclusive cost of $1.8 million.
- Continue with establishing and refining reconciliation practices and procedures to evaluate the MRE and operational effectiveness on monthly, quarterly and annual bases.

The proposed budget for these exploration and geology recommendations are presented in Table 26-1.

Table 26-1: Proposed Budget for Mine Geology-Related Recommendations

Activity	Cost ($)
Update MRE	80,000
Bench Height Study	40,000
Drilling Program of 15,000 m (Categorization)	3,000,000
Drilling Program of 8,000 m (Upside below Pit Design)	1,800,000
Desktop Study for underground extension below the Pit	150,000
MRE—External Audit	80,000
Total	**5,150,000**

26.2 Exploration Recommendations

- Undertake further Mineral Resource definition drilling at Kailey, targeting the No. 9 Zone near the surface to convert existing Inferred to Indicated Mineral Resources and to discover new gold-bearing zones in the existing pit shell.
- Undertake metallurgical testwork 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 digitize all QA/QC data from the Ontex-era drilling pre-2009 (present in drill logs and assay certificates).
- Selective sampling of gold-bearing zones was completed on much of the historic Brookbank drill core. Cut and sample wider, continuous intervals consistently along strike to confirm areas of barren rock adjacent to the main ore zones and perhaps identify new mineralized zones.
- Resample the drill core at Key Lake to increase the overall sample coverage and overcome the effects of previous under-sampling.

The proposed budget for these Exploration Recommendations are presented in Table 26-2.

Table 26-2: Proposed Budget for Exploration-Related Recommendations

Activity	Cost ($)
Metallurgical Testwork at Kailey (drilling, sampling, metallurgy)	200,000
Resampling of Drill Core at Brookbank and Key Lake	100,000
Expansion Drilling at Kailey	300,000
Total	**600,000**

26.3 Engineering Recommendations

- 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. Conduct additional laboratory testing (i.e. triaxial testing) and intact shear strength of foliation.
- As part of detailed design for each stage of raising of the TMF, optimize the combination of measures (i.e., larger stability berms, shear keys or ground improvement) to achieve the target stability factors of safety.

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28 CERTIFICATES OF QUALIFIED PERSON

28.1 Alexandre Dorval, P.Eng.

I, Alexandre Dorval, as an author of this report titled *Technical Report on the Greenstone Gold Mine, Geraldton, Ontario*, with an effective date of June 30, 2024, (the "Technical Report") prepared for Equinox Gold Corp. (the "Issuer"), do hereby certify that:

1. I am currently employed as Open Pit Mining Engineering Coordinator with G Mining Services Inc., with an office located at 5025 Boul. Lapinière, Suite 1010, Brossard, Québec, J4Z 0N5.
2. This certificate applies to the report titled *Technical Report on the Greenstone Gold Mine, Geraldton, Ontario*, with an effective date of June 30, 2024, (the "Technical Report") prepared for Equinox Gold Corp. (the "Issuer").
3. I graduated from Laval University, Canada, with a B.Sc. in Mining & Mineral Engineering in 2012. I am a professional engineer in good standing with the Ordre des Ingénieurs du Québec (#5027189), Professional Engineers of Ontario (#100214598), Professional Engineers and Geoscientists of Newfoundland & Labrador (#11042) in Canada, and the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (#L5771).
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have visited the Greenstone Gold Mine on July 24, 2024.
6. I am responsible for Sections 1.4, 1.5, 1.14.2, 1.16, 15, 16, and 25.1.2 of this Technical Report.
7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
8. I have not had prior involvement with the property that is the subject of this Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 30, 2024
Signing Date: October 1, 2024

Original Signed and Sealed

Alexandre Dorval, P.Eng.

28.2 Réjean Sirois, P.Eng.

I, Réjean Sirois, P.Eng., as an author of this report titled *Technical Report on the Greenstone Gold Mine, Geraldton, Ontario*, with an effective date of June 30, 2024, (the "Technical Report") prepared for Equinox Gold Corp. (the "Issuer") do hereby certify that:

1. I am a Geological Engineer acting as Senior Advisor, Geology & Resources at G Mining Services, with an office at 5025, Lapiniere Blvd, Suite 1010, Brossard, Quebec, J4Z 0N5.
2. I graduated of l'Université du Québec à Chicoutimi with a B.Sc. (Geological Engineering) in 1983.
3. I am a registered member in good standing with Ordre des Ingénieurs du Québec, licence no. 38754, the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) and the Prospectors & Developers Association of Canada (PDAC).
4. I have practiced my profession continuously since 1985 and have extensive experience in estimating mineral resources for various types of mineral deposits in South and North America, and in Southern and West Africa.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I have visited the Greenstone Gold Mine property and visited the site on numerous occasions since August 2016. The last visit was between July 22 to July 25, 2024.
7. I am responsible for Sections 1.3, 1.14.1, 1.16, 4–12, 14, 23, 25.1.1, 26.1, and 26.2 of this Technical Report.
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
9. I have prior involvement with the property that is the subject of this Technical Report as I was QP for the Mineral Resource sections in the Technical Report filed on January 26, 2021.
10. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
11. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 30, 2024
Signing Date: October 1, 2024

Original Signed and Sealed

Réjean Sirois, P.Eng.

28.3 Kenneth Arthur Bocking

I, Kenneth Arthur Bocking, state that:

1. I am a Fellow at:

   WSP Canada Inc.
   6925 Century Ave
   Mississauga, ON, Canada, L5N 7K2

2. This certificate applies to the technical report titled *Technical Report on the Greenstone Gold Mine, Geraldton, Ontario* with an effective date of: June 30, 2024 (the "Technical Report") prepared for Equinox Gold Corp. (the "Issuer").
3. I am a "qualified person" for the purposes of National Instrument 43-101 (NI 43-101). My qualifications as a qualified person are as follows. I am a graduate of the University of Saskatchewan with a Bachelor of Engineering degree in Civil Engineering in 1974 and an M.Sc. degree in geotechnical engineering in 1979. I am registered as a Professional Engineer in Saskatchewan, Ontario, and Northwest Territories/Nunavut. My relevant experience after graduation for the purpose of the Technical Report includes over 50 years of consulting geotechnical engineering, which has specialized in mine waste management since 1988. During that period, I have been responsible for the design, construction and closure of numerous mine waste facilities.
4. My most recent personal inspection of the Greenstone Gold Mine occurred on September 3 to 5, 2024.
5. I am responsible for Items 1.9, 18.1, 18.2, 18.3, 26.3, parts of 1.14.4 and 25.1.4 of the Technical Report.
6. I am independent of the issuer as described in section 1.5 of NI 43-101.
7. I have had prior involvement with the property that is the subject of the Technical Report. My prior involvement with the property that is as Engineer of Record for the Tailings Management Facility.
8. I have read NI 43-101 and the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and Form 43-101F1.
9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 30, 2024
Signing Date: October 1, 2024

Original Signed and Sealed

Kenneth Arthur Bocking, P.Eng.
(Ontario, Saskatchewan, Northwest Territories, and Nunavut)

28.4 Michelle Fraser, P.Geo.

I, Michelle Fraser, as an author of this report titled *Technical Report on the Greenstone Gold Mine, Geraldton, Ontario*, with an effective date of June 30, 2024, (the "Technical Report") prepared for Equinox Gold Corp. (the "Issuer"), do hereby certify that:

1. I am currently a Senior Hydrogeologist and Water Resources Technical Discipline Leader with Stantec Consulting Ltd. with an office at 300-100 Hagey Boulevard, Waterloo, Ontario, N2L 0A4.
2. I graduated from the University of Waterloo, Waterloo, Ontario, with B.Sc. in Honours Earth Sciences–Geology Specialization in 2005, and with a M.Sc. in Earth Sciences–Hydrogeology in 2007.
3. I am a Professional Geoscientist and registered member in good standing with the Professional Geoscientists Ontario, (P.Geo. Licence: 1854).
4. I have practiced my profession continuously in environmental consulting since my graduation from university. I have been involved in mining environmental studies for 10 years, including baseline studies, environmental assessments, permitting, closure plans, feasibility studies. This has included work on Crawford Project, Lynn Lake Gold Project, Marathon PGM Project, Gold River Trend Project, and Bissett Creek Project.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I have visited the Greenstone Gold Mine on July 21 and 22, 2024.
7. I am responsible for Sections 1.8, 1.10, 1.14.5, 5, 20, 25.1.5 of this Technical Report.
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
11. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 30, 2024
Signing Date: October 1, 2024

Original Signed and Sealed

Michelle Fraser, P.Geo.

28.5 Nicolas Vanier-Larrivée, P.Eng.

I, Nicolas Vanier-Larrivée, as an author of this report titled *Technical Report on the Greenstone Gold Mine, Geraldton, Ontario*, with an effective date of June 30, 2024, (the "Technical Report") prepared for Equinox Gold Corp. (the "Issuer"), do hereby certify that:

1. I am currently a Study Manager at G Mining Services with an office at 5025 Lapinière Blvd, Brossard, Quebec, Canada, J4Z 0N5.
2. I graduated from the University of Sherbrooke with a Bachelor of Applied Science (B.A.Sc.) in 2006. I am a registered member in good standing with Association of Professional Engineers and Geoscientists of Saskatchewan (License #78043) and Ordre des Ingénieurs du Québec (License No. 143023).
3. I have experience in the development, design, operation and commissioning of mineral processing plants, with a focus on gold projects. I have experience with technical reviews, preliminary economic assessments, feasibility and pre-feasibility studies, project and construction management, contracts management and cost estimation.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have visited the Greenstone Gold Mine on July 24, 2024.
6. I am responsible for Sections 1.7, 1, 18.4,18.5, 18.6, parts of 1.14.4 and 25.1.4 of this Technical Report.
7. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 30, 2024
Signing Date: October 1, 2024

Original Signed and Sealed

Nicolas Vanier-Larrivée, P.Eng.

28.6 Pierre Roy, P.Eng.

I, Pierre Roy, I, as an author of this report titled *Technical Report on the Greenstone Gold Mine, Geraldton, Ontario*, with an effective date of June 30, 2024, (the "Technical Report") prepared for Equinox Gold Corp. (the "Issuer"), do hereby certify that:

1. I am currently a Senior Metallurgist and Specialist–Mineral Processing at Soutex inc. located at 1990 rue Cyrille-Duquet, Local 204, Quebec, Province of Quebec, Canada G1N 4K8.
2. I graduated from the Université Laval (Quebec, Quebec, Canada) with a B.Sc. in Mining Engineering in 1986, and a M.Sc. in Mining in 1989.
3. I am a registered member in good standing with I am a registered member in good standing with Ordre des ingénieurs du Québec (Licence No. 45201).
4. I have practiced my profession continuously in the mining industry since my graduation from university. I have been involved in mining operations, engineering and financial evaluations for 36 years. During this time, I have been involved in mineral processing and environmental coordination at Kiena mine for six (6) years and Troilus mine for nine (9) years. I have also worked as consultant for the mineral processing industry for two (2) years at CRM in Québec and with Soutex inc. in Québec for fifteen (19) year. As consultant I have been involved in many projects with Iron, base metals and gold mining sectors.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I have visited the Greenstone Gold Mine on May 31 to June 7, 2024.
7. I am responsible for Sections 1.6, 1.14.3, 13, 17, and 25.1.3 of this Technical Report.
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
11. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 30, 2024
Signing Date: October 1, 2024

Original Signed and Sealed

Pierre Roy, P.Eng.

28.7 Carl Michaud, P.Eng.

I, Carl Michaud, P.Eng., as an author of this report titled *Technical Report on the Greenstone Gold Mine, Geraldton, Ontario*, with an effective date of June 30, 2024, (the "Technical Report") prepared for Equinox Gold Corp. (the "Issuer"), do hereby certify that:

1. I am currently a Vice-President of Mining Engineering, at G Mining Services with an office at 5025, Boul. Lapinière, Office 1010, Brossard, Québec , Canada, J4Z 0N5
2. I have graduated from l'Université du Laval with a B.Sc. (Mine Engineering) in 1996. In addition, I obtained an M.B.A. from the Université du Québec à Chicoutimi, in 2012.
3. I am a Professional Engineer registered with the "Ordre des Ingénieurs du Québec" (OIQ Licence: No. 117090).
4. I have practiced my profession continuously in the mining industry since my graduation from university. I have experience in narrow-vein gold deposits, flat and steeply dipping, bulk and selective mining methods for base metals, mine infrastructure, design and planning, mine production and financial evaluation, reserve estimation, technical reviews, feasibility and pre-feasibility studies, project and construction management, contracts management and cost estimation. . I have occupied different positions, both technical and operational, related to mining engineering, in Underground and Open Pit operation. This experience includes Kiena and Sigma Gold mine (Placer Dome), Éléonore Mine (Goldcorp) and Mont Wright Mine (Arcelor Mittal)
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I have not visited the Greenstone Gold Mine.
7. I am responsible for Sections 1.12, 1.13, 1.14.6, 21, and 25.1.6 of this Technical Report.
8. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
11. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 30, 2024
Signing Date: October 1, 2024

Original Signed and Sealed

Carl Michaud, P.Eng.

28.8 Darrol van Deventer, P.Eng.

I, Darrol van Deventer, as an author of this report titled *Technical Report on the Greenstone Gold Mine, Geraldton, Ontario*, with an effective date of June 30, 2024, (the “Technical Report”) prepared for Equinox Gold Corp. (the “Issuer”), do hereby certify that:

1. I am a currently employed as Director, Mine Engineering at Equinox Gold with an office at Suite 1501–700 West Pender Street, Vancouver, BC, Canada, V6C 1G8.
2. This certificate applies to the report titled *Technical Report on the Greenstone Gold Mine, Geraldton, Ontario*, with an effective date of June 30, 2024, (the “Technical Report”) prepared for Equinox Gold Corp. (the “Issuer”).
3. I graduated from the University of the Johannesburg with a Bachelor of Technology in Mining Engineering in 2003. I am a registered member in good standing with Professional Engineers Ontario (License No. 100542047) and the Canadian Institute of Mining, Metallurgy and Petroleum. I have practiced my profession since 1996 and have extensive experience in both open pit and underground mining operations, engineering and financial evaluations for over 25 years for various types of mineral deposits in North and South America, and in Southern and West Africa.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have visited the Greenstone Gold Mine on July 22 to July 25, 2024.
6. I am responsible for Sections 1.1, 1.2, 1.12, 1.13, 1.14.6, 2, 3, 19, 22, 24, 25.2, and 26.3 of this Technical Report.
7. I am not independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.
8. I have had prior involvement with the property that is the subject of this Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: June 30, 2024
Signing Date: October 1, 2024

Original Signed and Sealed

Darrol van Deventer, P.Eng.